NovaGold Resources Inc.

Donlin Creek Gold Project

Alaska, USA

NI 43-101 Technical Report on
Second Updated Feasibility Study




Submitted by:
Tony Lipiec, P.Eng.
Gordon Seibel, R.M. SME
Kirk Hanson, P.E.

Effective Date: 18 November 2011
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Project Number: 166549

CERTIFICATE OF QUALIFIED PERSON

Ignacy (Tony) Lipiec (P.Eng.)
AMEC Americas Ltd.,
Suite 400, 111 Dunsmuir St
Vancouver, BC., Canada
Tel: 604-664-3130; Fax: 604-664-3057
E-mail: tony.lipiec@amec.com

I, Tony Lipiec, P.Eng., am employed as the Process Manager, Vancouver, with AMEC E&C Services Inc.

This certificate applies to the Technical Report entitled "Donlin Creek Gold Project Alaska, USA, NI 43-101 Technical Report on Second Updated Feasibility Study" (the Technical Report) with an effective date of 18 November 2011.

I am a Professional Engineer in the province of British Columbia. I graduated from the University of British Columbia with a B.A.Sc. degree in Mining & Mineral Process Engineering, in 1985.

I have practiced my profession for 25 years, and have previously been involved with metallurgical design and process engineering for gold and base metal deposits employing the metallurgy and unit operations being considered for this deposit. These projects have included large, remote deposits located in North America and South America. As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Donlin Creek Gold Project.

I am responsible for Sections 1, 2, 3, 4, 5, 6, 13, 14.5.2, 14.6.2, 14.8, 14.9.4, 14.9.5, 14.9.6, 17, 18, 19, 20, 21, 22, 23, 24, 25, 26, and 27 of the Technical Report.

I am independent of NovaGold Resources Inc. as independence is described by Section 1.5 of NI 43-101.

I have been involved with the Donlin Creek Project since 2011 during preparation of a feasibility study update on the Project.

I have read NI 43–101, and the portions of the Report for which I am responsible have been prepared in compliance with that Instrument.

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

“signed”

Tony Lipiec, P.Eng.

20 January 2012

CERTIFICATE OF QUALIFIED PERSON

Gordon Seibel, R.M. SME
AMEC E&C Services Inc.
780 Vista Blvd., Suite 100
Sparks, NV., 89434
Tel (775) 331 2375
Fax (775) 331 4153
E-mail: gordon.seibel@amec.com

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

This certificate applies to the Technical Report entitled "Donlin Creek Gold Project Alaska, USA, NI 43-101 Technical Report on Second Updated Feasibility Study" (the Technical Report) with an effective date of 18 November 2011.

I am a Registered Member of the Society of Mining, Metallurgy, and Exploration.

I graduated from the University of Colorado with a Bachelor of Arts degree in Geology in 1980. In addition, I obtained a Master of Science degree in Geology from Colorado State University in 1991. I have practiced my profession for over 30 years. I have been directly involved in the development of resource models and mineral resource estimation for mineral projects in North America, South America, Africa , and Australia since 1991.

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 7, 8, 9, 10, 11, 12, 14 (excepting 14.5.2, 14.6.2, 14.8, 14.9.4, 14.9.5 and 14.9.6), and those portions of the Summary, Interpretations and 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 1.5 of NI 43-101.

I have been involved with the Donlin Creek Project since 2008 during preparation of the initial feasibility study and subsequent feasibility study updates on the Project.

I have read NI 43–101, and the portions of the Report for which I am responsible have been prepared in compliance with that Instrument.

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

“signed”

Gordon Seibel, R.M. SME

20 January 2012

CERTIFICATE OF QUALIFIED PERSON

Kirk Hanson, P.E.
AMEC E&C Services, Inc.
780 Vista Blvd., Suite 100
Sparks, NV, 89434
Tel 775 997 6559
Fax: 775-331-4153
kirk.hanson@amec.com

I, Kirk Hanson, P.E., am employed as the Technical Director, Open Pits, North America, with AMEC E&C Services Inc.

This certificate applies to the Technical Report entitled "Donlin Creek Gold Project Alaska, USA, NI 43-101 Technical Report on Second Updated Feasibility Study" (the Technical Report) with an effective date of 18 November 2011.

I am registered as a Professional Engineer in the state of Alaska (12126).

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. I have over 20 years of experience in the mining industry, predominately at hard rock open pit mines.

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 15 and 16, and those portions of the Summary, Interpretations and Conclusions and Recommendations that pertain to those Sections.

I am independent of NovaGold Resources Inc. as independence is described by Section 1.5 of NI 43-101.

I have been involved with the Donlin Creek Project since 2008, during preparation of the feasibility study and subsequent study updates on the Project.

I have read NI 43–101, and the portions of the Report for which I am responsible have been prepared in compliance with that Instrument.

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

“signed and sealed”

Kirk Hanson, P.E.

20 January 2012

AMEC E&C Services, Inc.
780 Vista Boulevard
Sparks, NV, 89434
Tel (775) 331 2375
Fax (775) 331 4153

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for NovaGold Resources Inc. (NovaGold) by AMEC Americas Limited (AMEC). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in AMEC's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by NovaGold subject to terms and conditions of its contract with AMEC. Except for the purposes legislated under Canadian provincial securities law, any other uses of this report by any third party is at that party's sole risk.

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1.0 SUMMARY

NovaGold Resources Inc. (NovaGold) requested AMEC Americas Limited (AMEC) to prepare a summary report (the Report) on the results of the second updated feasibility study (FSU2) for the Donlin Gold Project (the Project) in Alaska, USA.

The Project is a 50:50 partnership between NovaGold Resources Alaska, Inc, a wholly-owned subsidiary of NovaGold) and Barrick Gold U.S. Inc, (a wholly-owned subsidiary of Barrick). The partners use an operating company, Donlin Gold LLC (Donlin Gold) to manage the Project. For the purposes of this Report, Donlin Gold is used as a synonym for the partnership. Prior to July 2011, Donlin Gold was known as Donlin Creek LLC (DCLLC).

NovaGold is using the Report in support of a press release dated 5 December, 2011, entitled "NovaGold Passes Key Milestone On Path to Becoming Premier North American Gold Producer; Completes Positive Feasibility Study On Donlin Gold Project Natural Gas Pipeline's Economic Benefits Confirmed Capex Estimate Declines From Previous Guidance Project Ready to Advance to Permitting", and a press release dated 12 January 2012 entitled ""NovaGold Files Donlin Gold Feasibility Study Technical Report". The report was amended 20 January 2012 because the original filing was inadvertently of the review copy for Edgarizing, and not the final Sedar pdf report version, and omitted the cover page and certificates of QP.

1.1 Principal Outcomes

- Proven and Probable Mineral Reserves estimated for approximately 34 Moz contained gold:
  - Proven Mineral Reserves: 7.7 Mt at 2.32 g/t Au (0.6 Moz contained gold)
  - Probable Mineral Reserves: 497 Mt at 2.08 g/t Au (33.3 Moz contained gold).
- 25 year operating mine life
- 27 year process life at 53,500 t/d throughput (includes two years of stockpile processing at the end of the operating mine life)
- Average annual gold production:
  - 1.1 Moz over the projected life of mine
  - 1.5 Moz over the first full 5 years
  - 1.4 Moz over the first full 10 years.
- Predicted total cash costs:

- $585/oz [1] Au over the life of mine
- $409/oz over the first full 5 years
- $452/oz over the first full 10 years

- Net present after-tax cash flow (net present value (NPV) 5%)
  - At $1,000/oz gold price negative 1,342 million
  - At $1,200/oz gold price (Base Case) $547 million
  - At $1,700/oz gold price $4,581 million
  - At $2,000/oz gold price $6,722 million
  - At $2,500 oz gold price $10,243 million.
- Average annual cash flow for first full five years of production[2]
  - At $1,000/oz gold price $673 million
  - At $1,200/oz gold price $950 million
  - At $1,700/oz gold price $1,500 million
  - At $2,000/oz gold price $1,783 million
  - At $2,500 oz gold price $2,184 million.
- Increase in contained gold ounces of approximately 4.7 Moz in Proven and Probable Mineral Reserve over the previous Proven and Probable Mineral Reserve estimate of 31 December 2008.

Table 1-1 summarizes the key physical, technical, and financial parameters and the results of the FSU2 report.

1.2 Location, Climate, and Access

The Donlin deposits are situated approximately 280 miles (450 km) west of Anchorage and 155 miles (250 km) northeast of Bethel up the Kuskokwim River. The closest village is the community of Crooked Creek, approximately 12 miles (20 km) to the south, on the Kuskokwim River. There is no road or rail access to the site. All access to the Project site for personnel and supplies is by air.

The nearest roads are in the Anchorage area. Access to Bethel and Aniak, the regional centres, is limited to river travel by boat or barge in the summer and air travel year-round. The Kuskokwim River is a regional transportation route and is serviced by commercial barge lines.

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.

Table 1-1: Donlin Gold Project Financial Summary

Item	Unit	LOM	$/oz	$/t milled	$/t mined
Total Mined	Mt	3,260	—	—	—
Ore Tonnes Treated	Mt	505	—	—	—
Gold Grade	g/t	2.09	—	—	—
Gold Contained	Moz	33.849	—	—	—
Gold Recovery	%	89.8	—	—	—
Gold Recovered	Moz	30.401	—	—	—
Gold Payable	Moz	30.371	—	—	—
Gold Price	$/oz	1,200	—	—	—
Gold Gross Revenue	$M	36,481	1,200	72.27	11.19
OP Mining	$M	8,200	270	16.24	2.52
Processing	$M	7,808	257	15.47	2.39
G&A + Land Payments	$M	3,068	101	6.08	0.94
Payable Metal Deduction - Gold	$M	36	1	0.07	0.01
Doré TC+RC+Freight+Insure	$M	31	1	0.06	0.01
Direct Operating Costs + Metal Charges	$M	19,144	630	37.92	5.87
IFRS Total Capitalized Opex	$M	(1,386)	(46)	(2.75)	(0.43)
Stockpile Inventory Adjustment - Opex	$M	—	—	—	—
Total Operating Costs	$M	17,758	584	35.18	5.45
Depreciation	$M	9,846	324	19.50	3.02
Total Costs Before Taxes	$M	27,604	908	54.68	8.47
Cash Taxes	$M	2,741	90	5.43	0.84
Total Costs Including Taxes	$M	30,345	998	60.11	9.31
EBITDA	$M	18,581	611	36.81	5.70
Excluded from Cash Costs:					
Community & Social Development Costs	$M	141	—	—	—
Project Development / Start-up Expenses	$M	2	—	—	—
Funding of Closure “Trust Fund”	$M	274	—	—	—

Note: EBITDA = earnings before interest, taxes, depreciation, and amortization

The area has a relatively dry interior continental climate with typically about 20 inches (500 mm) of total annual precipitation.

1.3 Agreements

On December 1, 2007, NovaGold entered into a limited liability company agreement with Barrick that provided for the conversion of the Donlin Gold Project into a new limited liability company, the Donlin Creek LLC, which is jointly owned by NovaGold and Barrick on a 50/50 basis. In July 2011, the Board of Donlin Creek LLC voted to change the name of the company to Donlin Gold LLC.

The Donlin exploration and mining lease currently includes a total of 72 sections in the vicinity of the deposit and additional partial sections associated with the Project infrastructure leased from Calista Corporation, an Alaska Native Corporation that holds the subsurface (mineral) estate for Native-owned lands in the region. Following a renegotiation in March 2010, the lease runs through April 2031 with provisions to extend beyond that time. Title to all of these sections has been conveyed to Calista by the Federal Government. Calista owns the surface estate on 27 of these 72 sections.

A separate Surface Use Agreement with The Kuskokwim Corporation (TKC), an Alaska Native Village Corporation that owns the majority of the private surface estate in the area, grants non-exclusive surface use rights to Donlin Gold on at least 34 sections overlying the mineral deposit, with provisions allowing for adjusting that area in conjunction with adjustments to the subsurface included in the Calista lease. The term of the Surface Use Agreement runs through 5 June 2015 with provisions to extend beyond that time so long as mining, processing, or marketing operations are continuing and the Calista lease remains in effect.

The Lyman family owns a small (13 acre) private parcel in the vicinity of the deposit and holds a placer mining lease from Calista that covers approximately four sections.

1.4 Mineral Tenure

Donlin Gold has 49,261 acres (20,081 hectares) leased from Calista as mineral rights. In addition, Donlin Gold holds 242 Alaska State mining claims comprising 31,740 acres (12,845 hectares), bringing the total land package to 81,361 acres (32,926 hectares). Of these claims, three are on State-selected lands and a total of 158 are tentatively approved from conveyance from Federal to State-owned, pending survey. None of the claims held by Donlin Gold have been surveyed.

1.5 Surface Rights

Donlin Gold, through native lease agreements, holds a significant portion of the surface rights that will be required to support mining operations in the proposed mining area. Negotiations with TKC will be required for surface rights for additional lands supporting mining and access infrastructure. The currently identified Mineral Resources and the bulk of the proposed primary infrastructure (mill and waste rock facilities) are located on the leased lands.

Other lands required for offsite infrastructure, such as those required for the Jungjuk port site, road to the port site, gas pipeline, and tailings storage facility in Anaconda Creek, are categorized as Native, State of Alaska conveyed, or Bureau of Land Management (BLM or Federal) lands.

Rights-of-way will be required from the State and BLM for the road and pipeline alignments where they cross state and federal lands, respectively. Discussions regarding the extension and expansion of the TKC Surface Use Agreement and the disposition of the Lyman family land parcel and lease are ongoing.

1.6 Royalties

A net proceeds royalty is payable to Calista of equal to 8% of the net proceeds realized by Donlin Gold at the Project after deducting certain capital and operating expenses (including an overhead charge, actual interest expenses incurred on borrowed funds and a 10% per annum deemed interest rate on investments not made with borrowed funds). Part of this royalty is paid as advance, pre-set, minimum royalty payments.

There are currently no Government royalty obligations.

1.7 Environment, Permitting and Socio-Economics

There has been a focused effort for at least 15 years to collect comprehensive environmental baseline data and lay the groundwork with local and regulatory stakeholders for the successful permitting and development of a large-scale mining operation at Donlin. Baseline data collected has included studies covering wetland delineation, water quality, fish and aquatic habitats, air quality, wildlife habitats, cultural resources and heritage, subsistence, traditional knowledge, socio-economics, health, mercury data, overburden, ore and waste rock characterization studies, noise, visual aesthetics, and river and land use.

The National Environmental Policy Act (NEPA) process and formal permit applications will require the preparation of an environmental impact statement (EIS). Upon completion of the NEPA process, a Record of Decision (ROD) will be prepared that approves the preferred alternative for the Project, describes the conditions of the approval, and explains the basis for the decision. The State permitting process typically is not finalized until the NEPA process is completed.

Key environmental issues from stakeholders and regulatory authorities are likely to include mercury and cyanide management and water usage and management.

Donlin Gold and Barrick have maintained all of the necessary permits for exploration and camp facilities. Project development will require appropriate permits from both State and Federal regulatory authorities, and operational and construction permitting is likely to require at least 80 separate permits. Each Federal and State permit will have compliance stipulations requiring review and possibly negotiation by the applicant and appropriate agency. The comprehensive permitting process will determine the exact number of management plans required to address all aspects of the Project to ensure compliance with environmental design and permit criteria.

A preliminary closure plan has been prepared, which includes both concurrent reclamation during mining activities, and post-mining rehabilitation and monitoring. A modified version of the Barrick Reclamation Cost Estimator (BRCE) was used to develop the reclamation and closure cost estimate of $131.3 million. This amount is included in a Trust Fund for Reclamation, Closure costs and Post-Closure Obligations model prepared to determine the funding required to generate sufficient cash flow to cover the following costs: spillway construction from Anaconda creek to Crevice Creek; capital to construct a water treatment plant; perpetual water treatment; long-term monitoring; and associated facility and access maintenance. The total amount to cover reclamation / closure costs and post-reclamation and closure maintenance is estimated at $273.7 million, paid annually at $8.6 million over 32 years, including the construction period and 27-year life-of-mine.

1.8 Geology and Mineralization

The Donlin mineralization model is a high-level, reduced intrusion-related vein system. The Lewis–ACMA part of the district is a low sulphidation, reduced intrusion related, epizonal system with both vein and disseminated mineral zones.

The Donlin gold deposits lie in the central Kuskokwim basin of southwestern Alaska, which contains a back-arc continental margin basin fill assemblage of the Upper Cretaceous Kuskokwim Group, and Late Cretaceous volcano-plutonic complexes. The Project area is underlain by a 5 mile (8.5 km) long x 1.5 mile (2.5 km) wide granite porphyry dike and sill swarm hosted by lithic sandstone, siltstone, and shale of the Kuskokwim Group.

The deposits are hosted primarily in igneous rocks and are associated with an extensive Upper Cretaceous gold–arsenic–antimony–mercury hydrothermal system. The northeast, elongated, roughly 5,000 ft (1.5 km) wide x 10,000 ft (3 km) long cluster of gold deposits has an aggregate vertical range that exceeds 3,100 ft (945 m). These areas consist of the ACMA and 400 Zone, Aurora and Akivik mineralized areas (grouped as ACMA) and the Lewis, South Lewis, Vortex, Rochelieu and Queen mineralized areas (grouped as Lewis).

Gold occurs primarily in sulphide and quartz–carbonate–sulphide vein networks in igneous rocks and, to a much lesser extent, in sedimentary rocks. Broad disseminated sulphide zones formed in igneous rocks where vein zones are closely spaced. Sub-microscopic gold, contained primarily in arsenopyrite and secondarily in pyrite and marcasite, is associated with illite–kaolinite–carbonate–graphite-altered host rocks.

In the opinion of the QPs, knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and

Mineral Reserve estimation. The mineralization style and setting of the Project deposit is also sufficiently well understood to support Mineral Resource and Mineral Reserve estimation.

1.9 Exploration

Placer gold was first discovered at Snow Gulch, a tributary of Donlin Creek, in 1909. Early stage exploration in the modern era was performed by Resource Associates of Alaska (1974–1975), Western Gold Exploration and Mining Co. LP (WestGold) during 1988–1989 and Teck Exploration Ltd. (Teck) in 1993. Exploration included geological mapping, trenching, rock and soil sampling, an airborne magnetic and VLF survey, ground magnetic surveys, and initial Mineral Resource estimates.

The majority of the work completed on the Project has been primarily undertaken, in chronological order, by Placer Dome (1995 to 2000, and again from 2002 to 2005), NovaGold (2001 to 2002), Barrick (2006) and from 2007 to date by Donlin Gold.

Activities have included construction of infrastructure to support exploration activities, reconnaissance and geological mapping; aerial photography; rock chip and soil sampling; trenching; max-min (EM) geophysical surveys; airborne geophysical surveys; RC and core drilling for resource infill, geotechnical, engineering, condemnation, waste rock, calcium carbonate exploration and metallurgical purposes; environmental baseline studies; community consultations; detailed metallurgical test work; geotechnical and hydrogeological studies; sampling of prospective calcium carbonate source areas; exploration and auger drilling program for sand and gravel sources; a series of Mineral Resource and Mineral Reserve estimates; and initial mining and engineering studies. This work culminated in a feasibility study in 2007, and updates to this study in 2009 and 2011.

In the opinion of the QPs, the exploration programs completed to date are appropriate to the style of the deposits and prospects within the Project. The exploration and research work supports the genetic and affinity interpretations.

1.10 Exploration Potential

The Project retains exploration potential. The Akivik and East ACMA areas have good potential for lateral extensions of mineralization to the northwest and southeast of the FSU2 pit footprint. In addition, known gold mineralization is likely to extend at depth at the base of the designed pit, and in some areas immediately adjacent the planned pit floor, has been intersected by current drilling. Several drilled prospects and other exploration targets along the 3.7 mile (6 km) igneous trend north of the resource area remain under-explored, for example the Snow and Dome prospects.

1.11 Drilling

Approximately 1,834 exploration and development diamond core (90%) and reverse circulation (RC) (10%) drill holes, totalling 1,337,321 ft (407,720 m), were completed from 1988 through 2010. Approximately 50% of the core and 40% of the holes were drilled during 2006–2007. All but about 20% (district exploration, carbonate resource, facilities condemnation, hydrology, infrastructure engineering) of this drilling was utilized for the current resource model. Supporting the FSU2 model are a total of 1,396 core (89%) and RC (11%) holes totalling 1,114,324 ft (339,733 m), and 282 trenches totalling 70,344 ft (21,441 m).

Core sizes used on the Project include: NQ3 (45.1 mm core diameter), NQ (47.6 mm), HQ3 (61.2 mm), HQ (63.5 mm), and PQ (85 mm). Since 2002, core drills have been used exclusively for all resource delineation, and RC drilling was relegated to condemnation and hydrology studies.

Standard logging and sampling conventions were used to capture information from the drill core and, where applicable, RC chips. Data captured included lithology, mineralization, alteration (visual), structural and geotechnical, with provision for geologists to add comments on the core if required.

A survey of nearly 200,000 core recovery records in the database revealed an overall length-weighted average core recovery of 95%. Average recovery increases from 80 to 95% from 0 to 40 m and then ranges from 95 to 100% below 40 m where overburden and surface weathering effects are generally absent.

Collar survey methods to 2001 included Brunton compass and hip chain, a Motorola GPS system and conventional theodolite survey methods. From 2002, an Ashtech Promark2 GPS post-processed system consisting of a base unit and up to two roving units has been employed.

The Sperry Sun single-shot camera method was used through 2000 for directional surveys to determine down-hole deviation. Reflex EZ Shot instrumentation was introduced in 2001. Approximately 60% of the core holes drilled within the resource model area were oriented to collect structural information for geotechnical and geological studies. Core orientation methods included clay impression, EZ Mark, and Reflex ACT instrument.

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

Core is digitally photographed and split in half with an electric rock saw equipped with water-cooled diamond saw blades. Drill holes are sampled from the top of bedrock to the end of the hole. The maximum sample length in zones consisting of intrusive rocks or that contain appreciable sulphide/arsenic minerals is 6.6 ft (2 m), whereas sample lengths in sedimentary rock zones that lack appreciable sulphide/arsenic minerals can be 9.8 ft (3 m). A minimum of three additional 6.6 ft (2 m) sample intervals are placed before and after each intrusive rock or mineralized zone.

Specific gravity data were collected primarily in 2006 by Barrick staff, using the wax immersion, water displacement method. The weighted average of all SG data points was 2.69.

In the opinion of the QPs, sampling methods are acceptable, meet industry-standard practice, and are acceptable for Mineral Resource and Mineral Reserve estimation.

Quality assurance and quality control (QA/QC) programs have been in place since 1995, and consist of the insertion of blank, standard reference material (SRM) and duplicate samples.

1.12 Sample Analysis and Security

The primary laboratory for all assaying has been ALS Chemex in Vancouver, BC. During the exploration programs, ALS Chemex held accreditations typical for the time, including, at various times, ISO9001:2000 and ISO 9002, and from 2005, ISO/IEC 17025 accreditations.

Most core samples from 2005 through 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. Crushing requirements have been to 70% minus 10 mesh (2 mm) at the Donlin facility, and subsequently to better than 85% passing minus 200 mesh or 75 µm at ALS Chemex.

A 1 oz (30 g) subsample of the pulp was assayed by ALS Chemex using fire assay-atomic absorption spectroscopy (AAS). Before 2007, the primary gold assay method was Au-AA23, which had an analytical range of 0.005 to 10 g/t Au. The Au-AA25 gold assay method was initiated in 2007 and had an analytical range of 0.01 to 100 g/t Au.

Samples that exceeded the analytical limit for a given method were re-assayed by fire-assay and gravimetric finish or “ore grade” fire-assay AAS.

ALS Chemex determined the sulphur content of each sample according to the Leco method. The Leco method was also used to analyze samples flagged for acid base accounting (ABA) for carbon content as well as to determine neutralization potential (NP) and acid potential (AP) according to the industry-standard ALS Chemex ABA procedure.

Most trace and major element data for drill holes located within the resource model boundary were 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. Subsequent multi-element trace analyses were performed at ALS (Chemex) using aqua regia or four-acid digestions followed by ICP ± mass spectrometry.

Sample security measures practiced included moving of core form the drill site to the core shack at the end of each drill shift, and tracking of sample shipments using industry-standard procedures. Donlin Gold is of the opinion that core storage is secure because Donlin is a remote camp and access is strictly controlled.

In the opinion of the QPs, the quality of the gold and sulphur analytical data are sufficiently reliable to support Mineral Resource and Mineral Reserve estimation without limitations on Mineral Resource confidence categories and sample preparation, analysis, and security are generally performed in accordance with exploration best practices and industry standards.

1.13 Data Verification

A number of data verification programs and audits have been performed over the Project history, primarily in support of compilation of technical reports on the Project and in support of mining studies. Checks were performed in 2002 (AMEC), 2005 and 2008 (NovaGold), and AMEC (2011).

In the opinion of the QPs, the data verification programs undertaken on the data collected from the Project adequately support the geological interpretations, the analytical and database quality, and therefore support the use of the data in Mineral Resource and Mineral Reserve estimation.

1.14 Metallurgical Testwork

Testwork completed by SGS-Lakefield Research, Hazen Research, and G&T Metallurgical Services (G&T) under Barrick's supervision has shown that the Donlin ore requires pre-treatment prior to cyanidation to recover the gold. Process development work has determined that pressure oxidation is the preferred method of pre-treatment. Extensive testwork on composites has shown that acceptable gold recoveries can be produced through a combination of flotation pre-concentration, POX, and CIL cyanidation.

Air flotation using the MCF2 flowsheet provides an estimated life-of-mine (LOM) average of 93.0% recovery, with CIL recoveries after POX at approximately 96.6% for an estimated combined plant total gold recovery of 89.8%. The concentrate pull will vary from 15% to 17% and that will result in a concentrate grade of 13.0 to 12.7 g/t Au.

Process selection is supported by extensive testwork. Placer Dome undertook an initial phase of testwork from 1995 to 1999 to define the basic process. Early on, it became apparent that direct cyanidation or CIL of ore or flotation concentrate returned very low recoveries. Pre-treatment by oxidation was considered necessary.

Placer Dome testwork included grinding, gravity concentration, flotation, POX, cyanidation, and neutralization. Subsequently from 2002 to 2005, Placer Dome also explored HPGR comminution, arsenopyrite/pyrite separation, nitrogen aerated flotation, and oxidation both by bio-oxidation and pressure autoclave.

At the end of 2005, another round of work began with some testing at G&T, but this was interrupted by the acquisition of Placer Dome by Barrick Gold, which subsequently assumed management of remaining testwork.

Major programs at the bench-scale level were initiated in 2006 to test grinding, flotation, POX, and neutralization. In addition to bench-scale work, major pilot-plant runs were performed in flotation, POX, and neutralization at the Barrick Technology Centre, SGS-Lakefield, G&T, and Hazen Research (Golden, U.S.A.). Both bench-level and pilot-plant scale testwork were conducted to develop process parameters and expand engineering information.

The key testing results and recommendations considered in the FSU2 are summarized as follows:

Mineralogy

- Sulphur occurs primarily as pyrite and arsenopyrite. Marcasite is an additional minor sulphide present in the ore. Pyrite contains only a minor portion of the gold, while arsenopyrite is the main gold carrier, with gold in solid solution (sub-microscopic) form. In particular, it is the finest arsenopyrite that has the highest grade of gold. The proportion of pyrite to arsenopyrite ranges from 4 to 2:1, with 3:1 being typical.
- Mercury in the ore at ~2 ppm average is primarily hosted by pyrite in solid solution (sub-microscopic form). No mercury minerals have been observed.
- Arsenic in the ore at ~2,800 ppm average is primarily hosted by arsenopyrite. However, arsenic also occurs as native arsenic and realgar.
- Antimony in the ore at ~80 to 90 ppm average is primarily hosted by stibnite, but also occurs at trace levels hosted by tetrahedrite.
- Chloride in the ore at ~20 to 25 ppm average is primarily hosted by muscovite, but is also carried to a lesser degree by apatite.
- Carbonate in the ore at 2.4% to 2.5% (analysis specified as CO2). The most common carbonate within the Donlin ores is ferroan dolomite (impure dolomite containing varying quantities of iron) followed by ankerite. Calcite and siderite are present but not common.

Direct Leach / CIL

- The whole ore is refractory to direct and CIL cyanidation processing, with very low recoveries (<15%) from either leaching methodology. High gold recovery is achieved by destruction of the sulphidic host matrix of the gold.

Crushing / Grinding

- The ores are considered moderately hard, with an average Ball Work Index (BWI) of 15 kWh/t and an average Minnovex SAG Power Index (SPI) of 87.5 minutes.
- The ores are amenable to SAG milling with reasonable operating efficiencies.
- Ore hardness is controlled significantly by rock lithology.

Flotation

- Flotation gold recoveries are highest from intrusive ores (94.7% to 97.5%), lower from the sedimentary ores (89.7% to 91.3%), and problematic for partially geologically oxidized ores (average 75.7%).

- All the testwork showed a very close relationship between arsenic and gold recovery, indicating the presence of gold in close combination with that element.
- An MCF2 style milling, chemical addition, and flotation duplicated (mill/chemical/float, mill/chemical/float) flowsheet provides a recovery increase of 1.8% to a 7% sulphur flotation concentrate and is economically favoured for Donlin.
- Using the MCF2 flowsheet, it is possible to concentrate the gold-bearing sulphides into a 7% sulphur concentrate recovering an overall average 93.0% of the gold (including 10% oxide ore in blend) into 15% of the plant feed mass. Required flotation residence time and reagent dosages are relatively high compared to other iron sulphide flotation processes.
- The partially oxidized (altered) ores, which are predominantly near surface, perform poorly through flotation, with an average flotation recovery of 75.7%. Initial testing using sulphidizing reagents to promote flotation recovery improvement have been unsuccessful on these ores.
- CIL leaching of the flotation tailings does not yield economically justifiable recovery of gold.

Pressure Oxidation (POX)

- POX allows for 96.6% recovery of the gold in CIL following oxidation.
- A concentrate pre-acidification circuit (to dissolve carbonates) and subsequent CCD wash of the pre-acidified concentrate (using uncontaminated waters) was indicated and has been incorporated as part of the process flowsheet.
- As a precautionary measure, a mercury recovery system will be incorporated on the autoclave gas products prior to emission to the atmosphere.

Neutralization

- The presence of carbonates in the flotation tailings allows for autoclave acid solution neutralization, thus decreasing the overall lime requirements. The carbonate content in the ore is an estimated average 224% of the stoichiometric content of the total sulphur in the ore.

CIL / Gold Recovery

- Carbon in Leach (CIL) processing of washed autoclave product provides for optimized gold recoveries of 96% to 97%, requiring relatively low amounts of cyanide.

- Lime consumption of the CIL feed can be minimized by operating the CIL circuit at a pH of ~9.0 to minimize lime consumption by precipitation of magnesium hydroxide. The Donlin ore contains a naturally high content of magnesium (6,500 to 6,600 ppm average), which is liberated from the ore through reaction with acid, both in the autoclave and neutralization circuits. Since this testwork program was completed, an alternative to operating at low pH in CIL has been identified whereby soluble magnesium is removed prior to CIL to enable operation at conventional pH.
- Reagent addition is minimized by high-efficiency washing of the autoclave product to 98% or greater washing efficiency.
- Pilot testing of the CIL circuit on autoclaved blended concentrate demonstrated that CIL recovery is not sensitive to carbon gold loadings. However, the carbon elution circuit has been designed to allow for low carbon loadings in the event that preg-robbing concentrates are encountered.
- Mercury leached into solution from the autoclave product by cyanide, which is not adsorbed onto carbon, is controlled to low levels within the recirculating process water streams by precipitation as a sulphide using a Cherokee UNR reagent.
- Current design incorporates mercury gas/vapour recovery systems on the carbon regeneration kiln, electrowinning, retort system, and smelting furnace off-gases.

Environmental Considerations

- The high temperature and pressure oxidation process is considered best practice for generation of stable arsenic compounds suitable for long-term disposal in a tailings storage facility. Sufficient iron content is present in the Donlin ores to provide the recommended minimum stoichiometric ratio of 4:1 iron to arsenic.
- A portion of the arsenic in the Donlin ores is water soluble and liberates into solution within the operation of the grinding and flotation circuits alone.
- The tailings decant water from the Donlin process plant will likely contain elevated levels (above current aquatic life or drinking water standard) of As, Hg, Mn, Mo, Se, and Sb. The tailings water could also be elevated in sulphates (greater than 10 g/L), particularly due to the presence of magnesium, which increases solubility level of sulphate in solution.

1.15 Mineral Resource Estimate

The cut-off date for information used in the geologic model and resource model (termed the DC-9 model by Donlin Gold) was 1 November 2009.

The mineral estimate was prepared by Mr. Chris Valorose of Barrick and audited by AMEC. Three-dimensional solids for the geological model were constructed from polygons resulting from geologic interpretation of cross-section and level plans. Nine mineral and geological domains were assigned to the database. Geotechnical domain zone codes were input into the resource model, as required for the Lerchs–Grossmann (LG) pit optimization, using domain solids provided by BGC Engineering Inc (BGC) on 27 June 2008. A waste rock management category (WRMC) model was coded to identify overburden from the other WRMC codes.

Two specific gravity values were used: 2.65 for intrusive rocks, and 2.71 for sedimentary units.

Raw assay data were grouped by rock type, and capping values for gold were determined for each major rock type. Gold assays were capped above above 30 g/t Au. Values for neutralization potential (NP) were also capped. Total sulphur, arsenic, mercury, and antimony assays were not capped.

Composites were created down each hole at 20 ft (6 m) intervals. The composites were not broken at intrusive or sedimentary rock contact boundaries. Indicator semi-variograms generated at 0.25 g/t Au for the 6 m composites were fitted with a spherical model. Ranges of 98.4 ft (30 m) and 147.6 ft (45 m) were observed at 80% and 90% of the total sill variance.

A gold indicator model was used to estimate gold, arsenic, antimony, and mercury grades based on gold composite data. A separate sulphur indicator model was used to estimate sulphur.

Gold grades were estimated into the block model using an inverse distance to the third power methodology for two populations:

- Internal to the mineralized envelope, defined as blocks with indicator values greater than or equal to 50%
- External to the mineralized envelope, defined as blocks with indicator values less than 50%.

Interpolation of grade into the blocks was broken into five passes based upon increasing search distances, out to a maximum of 125 m. Gold grades were estimated separately for intrusive rocks, shales, and greywackes, and further sub-divided based upon whether blocks were internal or external to the mineralized envelope.

Sulphur grades were estimated using the same methods and parameters as for the gold grade estimation. Arsenic, mercury, and antimony grades were estimated using methods and parameters similar to those for the gold grade estimation. Values for CO_2, calcium, and magnesium were estimated into the block model based on an ordinary kriging method within nine estimation domains. Neutralization potential (NP) was estimated into the block model for use in the classification of waste rock.

The block model grades were validated visually against drill holes and composites in section and plan view. A nearest-neighbour block model was also generated. Grade profile plots were generated for the 6 m x 6 m x 6 m Measured and Indicated resource model as a further validation check. No estimation biases were noted from the validation reviews.

Dilution and selectivity of mineralized material were determined using a Barrick in-house program referred to as SMUman. The extent of the classified material that might have reasonable expectation for economic extraction was assessed by applying a LG pit outline using Whittle® software to the Mineral Resources. A net sales return (NSR) value per tonne was then coded into each block of the resource model. For those blocks with a resource classification of Measured or Indicated, the NSR per tonne value was calculated with the following equations:

General:

*NSR = [Au grade] * [Recovery] * [Price of Gold less Refining and Royalty Costs] – [Processing Costs+ General and Administrative Costs + Rehandling Costs] US$/tonne*

For Mineral Resources, the figures were:

*NSR = [Au grade] * [Recovery] * [US$1200 – (1.85 + ((US$1200 – 1.85) * 0.045))] – [(10.65 + (2.1874 * S%)) + 2.29 + 0.20] US$/tonne*

For Mineral Reserves the figures were:

*NSR = [Au grade] * [Recovery] * [US$975 – (1.78 + ((US$975 – 1.78) * 0.045))] – [(10.65 + (2.1874 * S%)) + 2.27 + 0.19] US$/tonne*

The NSR cut-off for Mineral Resource reporting purposes was $0.001/t milled, which represents the net sales return marginal cut-off strategy.

Mineral Resources take into account geologic, mining, processing and economic constraints, and have been confined within appropriate LG pit shells, and therefore are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves. The Qualified Person for the Mineral Resource estimate is Gordon Seibel, R.M. SME Registered Member, an employee of AMEC. Mineral

Resources are reported in Table 1-2 at a commodity price of $1,200/oz gold, have an effective date of 11 July 2011, and are inclusive of Mineral Reserves.

Factors which may affect the Mineral Resource estimate include the commodity price; changes to the assumptions used to generate the NSR cut-off; changes to the 0.25 g/t Au threshold used to define the indicator mineralized domains, changes in interpretations of fault geometry; changes to the search orientations used for grade estimation in the ACMA area, results of a review of the Measured classification criteria; and changes to the assumptions used to generate the LG pit constraining the estimate, in particular slope design assumptions.

1.16 Mineral Reserve Estimate

Mineral Reserves were optimized for all Measured and Indicated blocks assuming a gold selling price of $975/oz.

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 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.

Dilution was considered for bulk mineable (12 m bench height) and selective mineable (6 m bench height) scenarios. All blocks classified as Inferred were set to waste in the selective mining plan. In the bulk mineable plan, the entire 12 m block was assigned the highest confidence category of the sub-blocks in the plan. The grade of all Inferred blocks was set to zero at the start of the process. Therefore the combined grade of the 12 m block is derived from the Measured or Indicated metal grades only.

Pit shell generation was constrained in the northwestern part of the ACMA mining area, to prevent it from encroaching on Crooked Creek, which is a salmon-bearing stream, but was not constrained by any infrastructure considerations.

Geotechnical domains, design sectors, slope angles, and associated assumptions were provided by BGC. Mine design has incorporated geotechnical and hydrogeological considerations.

Table 1-2: Mineral Resources Summary Table, (Inclusive of Mineral Reserves) Effective Date 11 July 2011, Gordon Seibel, R.M. SME Registered Member

Category	Tonnage (kt)	Au (g/t)	Contained Au (koz)	S (%)
Measured	7,731	2.52	626	1.15
Indicated	533,607	2.24	38,380	1.08
Total Measured and Indicated	541,337	2.24	39,007	1.08
Inferred	92,216	2.02	5,993	1.08

Notes to Accompany Mineral Resources Table

1. Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
2. Mineral Resources are contained within a conceptual Measured, Indicated and Inferred optimized pit shell using the following assumptions: gold price of US$1,200/oz; variable process cost based on 2.1874 * (sulphur grade) + 10.6485; administration cost of US$2.29/t; refining, freight & marketing (selling costs) of US$1.85/oz recovered; stockpile rehandle costs of 0.20/t processed assuming that 45% of mill feed is rehandled; variable royalty rate, based on royalty of 4.5% * (Au price – selling cost)
3. Mineral resources have been estimated using a constant net sales return (NSR) cut-off of US$0.001/t milled. The NSR was calculated using the formula: NSR = Au grade * Recovery * (1,200 – (1.85 + (1,200 – 1.85) * 0.045)) (10.65 + 2.1874 * (S%) + 2.29 + 0.2) and reported in US$/tonne Assuming an average recovery of 89.54% and an average S% grade of 1.07%, the marginal gold cutoff grade would be approximately 0.46 q/t, or the gold grade that would equate to a $0.001 NSR cutoff at these same values.
4. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
5. Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

The base mining cost (before incremental mining cost with depth) is $1.51/st ($1.668/t), the average processing cost is $13.06/st ($14.39/t), and the G&A cost is $2.06/st ($2.27/t). These costs are considered reasonable.

Recoveries for non-oxide ores are quoted as a constant for each rock type, whereas recoveries for oxide ores vary with sulphur grade. Recoveries range from 88.6% in shale to 94.2% in the Akivik zone.

Mineral Reserves have been modified from Mineral Resources by taking into account geologic, mining, processing, and economic parameters and therefore are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves. The Qualified Person for the Mineral Reserve estimate is Kirk Hanson, P.E., an AMEC employee. Mineral Reserves are reported at a gold price of $975/oz gold, and have an effective date of 11 July 2011.

Mineral Reserves are summarized in Table 1-3.

Factors which may affect assumptions used in estimating Mineral Reserves include the commodity price; unrecognized structural complications in areas with relatively low drill hole density that could introduce unfavourable pit slope stability conditions;

changes in interpretation of the fault orientiations, in particular the Vortex and Lo Faults; changes in orientations of the bedding or ash layer orientations which may necessitate flatter slope angles than currently assumed; in-pit and pit wall water management if water inflows are higher than predicted; and the likelihood of obtaining required permits and social licenses to construct the gas pipeline and operate the planned mine.

Table 1-3: Proven and Probable Mineral Reserves, Effective Date 11 July 2011, K.Hanson, P.E.

Category	Tonnage (kt)	Au (g/t)	Contained Au (koz)	S (%)
Proven	7,683	2.32	573	1.12
Probable	497,128	2.08	33,276	1.06
Total Proven and Probable	*504,811*	*2.09*	*33,849*	*1.06*

Notes to Accompany Mineral Reserves Table

1. Mineral Reserves are contained within Measured and Indicated pit designs, and supported by a mine plan, featuring variable throughput rates, stockpiling and cut-off optimization. The pit designs and mine plan were optimized on diluted grades using the following economic and technical parameters: Metal price for gold of US$975/oz; reference mining cost of $1.67/t incremented $0.0031/t/m with depth from the 220 m elevation (equates to an average mining cost of $2.14/t), variable processing cost based on the formula 2.1874 x (S%) + 10.65 for each $/t processed; general and administrative cost of US$2.27/t processed; stockpile rehandle costs of 0.19/t processed assuming that 45% of mill feed is rehandled; variable recoveries by rocktype, ranging from 86.66% in shale to 94.17% in intrusive rocks in the Akivik domain; refining and freight charges of US$1.78/oz gold; royalty considerations of 4.5%; and variable pit slope angles, ranging from 23º to 43º.
2. Mineral Reserves are reported using an optimized net sales return (NSR) value based on the following equation: NSR = Au grade * Recovery * (US$975 – (1.78 + ($US975 – 1.78) * 0.045)) – (10.65 + 2.1874 * (S%) + 2.27 + 0.19) and reported in US$/tonne. . Assuming an average recovery of 89.54% and an average S% grade of 1.07%, the marginal gold cutoff grade would be approximately 0.57 g/t, or the gold grade that would equate to a $0.001 NSR cutoff at these same values.
3. The life of mine strip ratio is 5.48. The assumed life-of-mine throughput rate is 53.5 kt/d
4. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
5. Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

1.17 Proposed Mine Plan

The preferred development is for a 55 kst/d (50 kt/d) process facility with on-site power; and a mine capacity of 485 kst/d (440 kt/d) with an elevated cut-off policy applied in the initial part of the mine life. The processing rate was upgraded to 58 kst/d (53.5 kt/d) during the FSU2 design phase to take into account processing design constraints and rationalization of the proposed pressure oxidization circuit to be installed.

The ACMA ultimate pit has been divided into nine phases, the Lewis pit into six phases. The initial phases of the two pits are independent, but they partially merge later in the mine life, forming a final single pit. The mine design, complete with haulage access, includes 556,459 kst (504,811 kt) of ore containing 33,849 koz (1,052,815 kg) of in-situ gold and has a strip ratio of 5.48. The mine design is considered appropriate to the quantity of Measured and Indicated Mineral Resources estimated for the Project. AMEC notes that the engineered pit design includes approximately 5% less ore tonnage and 7% fewer Au ounces than the pit optimization shell it was based on. This is at the upper end of the generally accepted limit of a 10% reduction in tonnes. As such, there is a risk that the engineered pit design contains less ore than optimum.

Mineable pit phases were designed based on optimized nested pit shell guidance, gold grade, strip ratio, access, and backfilling of the ACMA phases. Ramps in final walls have a design width of 131 ft (40 m) and a gradient of 10%. A nominal minimum mining width of 492 ft (150 m) was used for phase design.

Dates in this paragraph are for illustrative purposes only, as no Project permits and approvals have been received, and Project development and construction has not been approved by the respective Boards of Donlin Gold, NovaGold and Barrick. Preproduction has been defined as starting in April 2018 and finishing at the end of December 2018, when the main orebody is exposed. Mill production starts in July 2019. The operating mine life is estimated to be 25 years based on a nominal processing rate of 59,000 stpd. The schedule incorporates long-term and short-term ore stockpiles. The long-term stockpile will hold all ore produced at the mine in excess of plant feed, separated into three sections according to sulphur grade for blending purposes.

The maximum long-term stockpile volume is 104.8 Mt at the end of 2031. This includes 18.5 Mt of high sulphur-grade material, 31.9 Mt of medium sulphur-grade material, and 53.9 Mt of low sulphur-grade material.

The short-term stockpile was established to cope with daily variations in plant capacity and to accommodate fluctuations in the average daily mill feed; this stockpile was assumed to have an average 45% annual re-handle.

After plant ramp-up, mill feed averages 52.7 kt/d and reaches a maximum of 54.4 kt/d in 2030 (Year 12). Contained gold in the mill feed averages approximately 1.3 Moz per year, while gold production averages 1.6 Moz per year for the first five years, with a maximum of 1.731 Moz in 2024 (Year 6).

In the opinion of Donlin Gold, the proposed plant feed supports that the amount of sulphur in the feed can be controlled through a blending strategy combining ore feed directly from the mine and from stockpiles.

A total of 2,460 Mst (2,232 Mt) of waste will be stored in a single ex-pit waste rock facility, in the American Creek Valley, east of the pit area. Another 466 Mst (423 Mt) of waste rock will be stored in the ACMA backfill dump and 18.7 Mst (17 Mt) of overburden in the overburden stockpiles for reclamation use. The remaining 114 Mst (103 Mt) is used as construction material, of which 99 Mst (90 Mt) is for tailings dam wall construction. Backfilling will commence in 2035 (Year 18) and continue until the end of mine life. In addition, 103 Mt of waste rock will be used for construction purposes, and 16.6 Mt of overburden will be stored in overburden stockpiles for reclamation purposes.

Surface ditches, a contact water pond (CWP) immediately upstream of the pit, plus diversion systems further upstream, will control surface waters in the pit and waste dump areas. Dewatering systems consisting of perimeter and in-pit vertical dewatering wells, horizontal drains, and in-pit sump pumps will be required to manage groundwater.

1.18 Process Design

Run-of-mine (ROM) ore at 59,000 stpd (53,500 t/d) from the Donlin deposits will be crushed in a gyratory crusher followed by a semi-autogenous grinding (SAG) mill and two-stage ball milling, addition of chemicals, and a flotation circuit (MCF2). The primary ball milling circuit will produce a P_{80} particle size of 120 to 150 µm as feed to the primary rougher flotation section. The secondary ball milling circuit will produce a P_{80} particle size of 50 µm as feed to the secondary rougher flotation section.

Gold-bearing sulphides, recovered by flotation, generate a concentrate containing 7% sulphur. The concentrate is refractory and will be treated in a pressure oxidation circuit prior to cyanidation. Overall gold recovery from flotation, pressure oxidation and cyanidation is estimated to be in the order of 89.83%. Excess acid from the autoclave circuit will be neutralized with flotation tailings and slaked lime. Tailings from the process will be impounded in a zero-discharge tailings storage facility; water reclaimed from here will be re-used in the process plant.

Mineralogical studies have shown that the gold is not visible. Testwork analysis indicates a high level of association of gold with arsenopyrite. Other sulphides such as pyrite and marcasite are also present, with reduced tenors of gold. Organic carbon, a potential preg robber, is present in the sedimentary ore. It is also present at lower levels in the intrusive ores, believed to be in the form of well-ordered graphite. This form of organic carbon is possibly less likely to preg-rob.

The average Bond work index for the ore is in the range of 15 kWh/t. Flotation work has shown that kinetics are initially rapid, but to achieve high recoveries, a combined primary and secondary rougher residence time over 100 minutes, together with a high reagent loading in the system, is required. Clay-like minerals will affect slurry viscosity and settling. Slurry density in the underflow will be less than 50% solids for the concentrate thickeners.

Partially geologically oxidized (altered) ore in the deposit, up to 7% of the mill feed, is the key non-performing ore type in the flotation circuit. Degradation of the sulphide ore via oxidation in the stockpile will also affect the flotation recovery, applied as 5% recovery loss within flotation on all ores stockpiled for longer than one year.

Pressure oxidation (POX) has been shown to be successful in releasing the valuable constituents, under certain conditions. To optimize oxidation conditions, the water systems design has been modified to use the highest-quality water in the oxidation circuit. The autoclave design incorporates variable level control to provide better control over operating residence time.

The oxidation circuit discharge will be washed to reduce lime load in carbon-in-leach (CIL); the washed solution will be neutralized by the use of high-carbonate flotation tails to further reduce plant lime consumption prior to tailings disposal.

Gold recovery by CIL has proven successful in treating Donlin ores and is estimated to be 96.6%. Rheological investigation and CIL testing results have determined that a relatively low CIL feed density of 35% solids should be adopted. In addition, to control lime usage, the CIL circuit will be operated at a pH of approximately 11.0.

Given the plan to use stockpiles to manage the ore blend into the process from the perspective of gold, sulphur, carbonate, and hardness, allowances were made for ore aging or stockpile degradation for the life-of-mine feed. Ore oxidized through weathering will have a slower flotation response than fresh rock. In general, ore at Donlin does not contain highly reactive sulphide species, and testwork has shown no statistical deviation over a one-year period. While data from a longer timeframe are not presently available, the testwork results for oxidized material show some degradation. Consequently, there is no effect on recovery for material stockpiled for less than one year (sulphide “fresh” material), and a recovery deduction of 5% has been applied to gold and sulphur recoveries for sulphide material stockpiled for longer than one year.

Alternative flowsheets to flotation-POX-CIL were considered, including whole ore pressure oxidation, roasting a flotation concentrate, and bio-oxidation (BIOX). None of these proved to be a viable economic alternative to the flotation-POX-CIL route.

1.19 Planned Project Infrastructure

The Project will require construction of significant infrastructure to support the planned producing facilities. Key infrastructure will include:

- Access road, 27 miles (44 km) long, from the mine site to the planned Kuskokwim River dock site at Jungjuk
- Airstrip to support DHC Dash 8 and the Hercules C-130 aircraft
- Barge cargo terminal at Jungjuk
- Marine cargo terminal at Bethel
- Two open pit mines
- Process plant site in the Anaconda valley
- Primary crusher area on a ridge on the south side of American Creek
- Fuel storage compound adjacent the process plant site
- Mining and road fleet truckshops in association with the primary crusher area
- Contact water management dams and a freshwater storage reservoir
- Water management pumping systems
- Power plant, located adjacent the process plant
- Tailings storage facility (TSF) in the Anaconda Creek basin
- Waste rock storage facility (WRF) in the American Creek valley
- Gas pipeline
- Construction (2,560 people) and permanent accommodation (maximum 638 people) camps.

In general, the design and construction of the mine site infrastructure will be relatively straightforward, although the scope of the work is extensive, especially in terms of the water systems. In addition, the Project involves several development sites considerable distances apart, incurring high infrastructure costs to provide interconnecting roads, pipelines, services, and utilities. The decision to use material from the plant site excavation as a borrow source for constructing the tailings dams is an effective way to reduce the site preparation costs.

The construction schedule for the initial phase of the TSF and the Lower CWD is aggressive, with a great deal of work to be completed in a short duration. Weather delays could affect completion on schedule.

1.20 Markets

NovaGold will be able to market gold produced from the Donlin Project. Sales contracts that could be negotiated would be expected to be within industry norms. However, the majority of production would be expected to be spot marketed.

1.21 Capital Costs

The 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% +30% of estimated final costs, per AACE Class 3 definition. All costs are expressed in second quarter (Q2) 2011 US dollars. No allowances are included for escalation, interest during construction, taxes, or duties.

The total estimated capital cost to design and build the Donlin Project described in this Report is $6,679 million, including an Owner-provided mining fleet and self-performed pre-development. Included in the estimate are:

- Direct capital costs: $4,009 million (includes gas pipeline direct cost of $758.1 M)
- Owner's costs: $414 million
- Other indirect costs: 1,271 million
- Contingency: $984 million.

Sustaining capital costs total $1,504 million. Significant areas include $649 million to replace and supplement mobile mining and support equipment and $631 million for periodic tailings storage facility capacity expansions.

AMEC notes the following in relation to the proposed natural gas pipeline. The direct costs of the pipeline are estimated at $758.1 M, with indirect costs of an additional $75.7 M ($38.7 M engineering procurement, $32.5 M construction costs and, $4.4 M Owners' costs, primarily for land), totalling $829.4 M, excluding contingency. When contingency is included, the pipeline costs are estimated to total $973 M.

1.22 Operating Costs

Operating cost estimates have been assembled by area and component, based on estimated staffing levels, consumables, and expenditures, according to the mine plan and process design. Operating costs have been prepared in second quarter (Q2) 2011 U.S. dollars with no allowances for escalation, sales tax, import duties, or contingency.

The estimated life-of-mine operating costs are $5.42/t mined or $34.99/t milled, or $581/oz.

1.23 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. Forward-looking information includes Mineral Resource and Mineral Reserve estimates, commodity prices and exchange rates, the proposed mine production plan, projected recovery rates, uncertainties and risks regarding the estimated capital and operating costs, uncertainties and risks regarding the cost estimates and completion schedule for the proposed Project infrastructure, in particular the proposed barging program, and the need to obtain permits and governmental approvals.

The overall economic viability of the Project has been assessed using both undiscounted and discounted cash flow techniques. Undiscounted techniques include total net cash flow, payback period (measured from start of production), earnings before interest, taxes, depreciation, and amortization (EBITDA), and cash costs.

Discounted values are calculated using a 5% discount rate and a discrete, end-of-year convention relative to reference dates of 1 January 2012 (FSU2) and 1 January 2014 (this Report, and Project Base Case). A period of approximately 3.5 years for permitting, starting 1 January 2012, is included prior to start of construction.

The economic evaluation of the Donlin Project was prepared by Donlin Gold and is based upon:

- Capital cost and sustaining capital cost estimates prepared by AMEC, BGC, and Hatch
- Owner's capital costs prepared by Donlin Gold
- Reclamation and closure costs prepared by SRK

- Post-closure obligations prepared by Donlin Gold
- Funding requirements for the reclamation, closure, and post-closure obligations endowment prepared by Donlin Gold
- Mine schedule prepared by Barrick
- Mineral Resource estimate prepared by Donlin Gold
- Mine equipment costs based on quotes received from equipment suppliers
- Estimated mine, process plant, and general and administration operating costs prepared by Donlin Gold, AMEC, Barrick, and Hatch, based on budget quotations, first principles, and/or costs at operating mines similar to that proposed for Donlin such as Barrick's Goldstrike operation
- An allowance for supply inventory and working capital (including doré transportation, in-process inventory, and payment delays); these values sum to zero over the life of the mine.
- Financial analysis of the Base Case (discount rate of 5%) showed the after-tax Project NPV to be $547 M and the internal rate of return (IRR) to be 6.0% (Table 1-4). The cumulative, undiscounted, after-tax cash flow value for the Project is $6,197 M and the after-tax payback period is 9.2 years.
- Sensitivity analyses were performed on the Project on a range of -20% to +20% on gold price, operating costs, and capital costs. For purposes of the sensitivity analysis, variations in the gold grade were assumed to mirror variations in the gold price.
- The Project is particularly sensitive to changes in the gold price. The Project requires a gold price of approximately $902/oz to break even on a cash flow basis and a gold price of approximately $1,141/oz to achieve an IRR of 5%. It is less sensitive to variations in operating cost and capital cost.

1.24 Preliminary Development Schedule

A preliminary Project development schedule has been generated. The schedule includes consideration of early work requirements, the environmental permitting process, EPCM and construction activities.

Table 1-4: Summary of Key Financial Evaluation Metrics (Base Case is highlighted)

Item	Unit	Value
Total Mined	Mt	3,260
Ore Tonnes Treated	Mt	505
Strip Ratio	W/O	5.46
Gold Recovered	Moz	30.401
Gold Recovery	%	89.8
Gold Price	$/oz	1,200
Total Operating Costs	$/oz	584
Total Costs Before Taxes	$/oz	908
Total Costs Including Taxes	$/oz	998
EBITDA	$M	18,581
Total Cash Flow*	$M	6,197
Jan 2012 NPV @ 5%**	$M	337
Jan 2012 IRR	%	5.6
Jan 2014 NPV @ 5%**	$M	547
Jan 2014 IRR	%	6.0
Payback Period	Years	9.2
Operation Life	Years	27.0
Initial Capital	$M	6,679
Total LOM Capital	$M	8,184

Note: EBITDA = Earnings before interest, taxes, depreciation, and amortization
* Cash flow excludes sunk costs
** Reference dates for DCF metrics are 1 January 2012 and 1 January 2014. The DCF metrics for 1 January 2014 treat funds expended before that date as sunk.
During 2012 and 2013, Donlin Gold intends to complete basic engineering and commence detailed engineering, in tandem with, and in the case of detailed engineering, subject to, progress achieved on the Environmental Impact Statement and associated permitting process. Aggregate expenditures in these years are expected to be approximately $172 million, which if excluded from the discounted cash flow analysis would result in an increased project NPV5 and IRR from 2014 onwards of $210 million and 0.4%, respectively.

1.25 Conclusions

AMEC considers that the scientific and technical information available on the Project can support proceeding with additional data collection, trade-off and engineering work and preparation of more detailed studies. However, the decision to proceed with a mining operation on the Project is at the discretion of Donlin Gold, NovaGold and Barrick.

1.26 Recommendations

Donlin Gold has completed a feasibility study and two updates on the study. A decision to proceed with any mine development plans would be made by the partners.

As a consequence, AMEC's recommendations are restricted to activities that would support permitting and detailed engineering studies. These activities are envisaged as a single phase of work, with no item or area dependent on results of another. The estimated total cost of the proposed work is in the range of $135,000 to $200,000.

2.0 INTRODUCTION

NovaGold Resources Inc. (NovaGold) requested AMEC Americas Limited (AMEC) to prepare a summary report (the Report) on the results of the second updated feasibility study (FSU2) for the Donlin Gold Project (the Project) in Alaska, USA (Figure 2-1 and Figure 2-2).

The Project is a 50:50 partnership between NovaGold Resources Alaska, Inc, (a wholly-owned subsidiary of NovaGold) and Barrick Gold U.S. Inc, (a wholly-owned subsidiary of Barrick). The partners use an operating company, Donlin Gold LLC (Donlin Gold) to manage the Project. For the purposes of this Report, "Donlin Gold" is used as a synonym for the partnership. Prior to July 2011, Donlin Gold was known as Donlin Creek LLC (DCLLC).

NovaGold is using the Report in support of a press release dated 5 December, 2011, entitled "NovaGold Passes Key Milestone On Path to Becoming Premier North American Gold Producer; Completes Positive Feasibility Study On Donlin Gold Project Natural Gas Pipeline's Economic Benefits Confirmed Capex Estimate Declines From Previous Guidance Project Ready to Advance to Permitting", and a press release dated 12 January 2012 entitled ""NovaGold Files Donlin Gold Feasibility Study Technical Report".

The report was amended 20 January 2012 because the original filing was inadvertently of the review copy for Edgarizing, and not the final Sedar pdf report version, and omitted the cover page and certificates of QP.

2.1 Terms of Reference

The second updated feasibility study was completed in October, 2011, and was a compendium of different studies by a number of companies, as indicated in Table 2-1.

AMEC used the information completed by these contributors to support information in the current Report. AMEC's QPs performed or commissioned independent due diligence reviews on the information supplied by Donlin Gold and made adjustments to the results of the FSU2 report based on the outcome of those reviews.

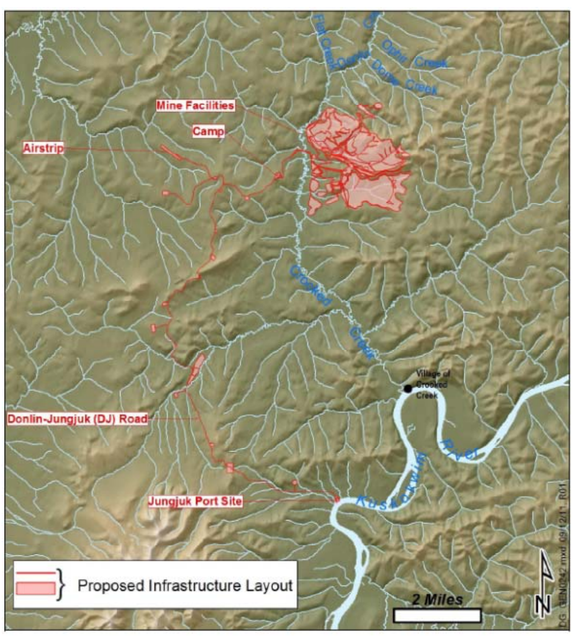
AMEC notes that the FSU2 project description has changed materially in some areas from the first updated feasibility study of February 2009 (FSU1).

The Report uses Canadian English. Unless otherwise specified in the text, monetary amounts are in US dollars and units are metric.

Figure 2-1: Regional Project Setting

Note: Figure courtesy Donlin Gold

Figure 2-2: Local Project Setting



Note: Figure courtesy Donlin Gold

Table 2-1: Consulting Firms or Entities Contributing to FSU2

Consulting Firm or Entity	Area of Responsibility in FSU2 Report Document
AMEC Americas Ltd	Overall study compilation, design of port and process facilities (excluding the autoclave and autoclave ancillary facilities), flowsheet, development of logistics program; equipment pricing, excluding equipment associated with the autoclave, oxygen plant, and mining, quantity estimation for major civil and structural components, capital cost estimates for off-site facilities, on-site facilities, and process facilities, excluding the mine, autoclave, autoclave support facilities, and oxygen plant, operating cost estimates for process, transport, and administration, excluding mining, development of Project plan and schedule
Donlin Gold and Barrick	Geologic modelling; resource and reserve estimation, specification and management of metallurgical testwork program; bench and pilot testing facilities for pressure oxidation and neutralization; specification and management of environmental and socioeconomic baseline studies, including impact analysis; permitting requirements; reclamation planning; baseline environmental data; process (excluding EPCM requirements) and mining engineering and preproduction costs; financial evaluation; mine planning; capital cost estimates for the mine; operating cost estimates for the mine; sustaining capital cost estimates for the mine
BGC Engineering Inc.	Geotechnical engineering to support the mine pits, waste rock facility, plant site, and tailings storage facility; site water management; mine waste rock management; design of the tailings storage facility and waste rock facility foundations; pit dewatering plans for the mine
CH2MHill	Routing and geotechnical studies for the selected alignment of the natural gas pipeline; pipeline design and engineering; construction execution planning and scheduling; capital and operating cost estimates for the natural gas pipeline
HATCH Ltd.	Flowsheet development of autoclave process; design of autoclave and autoclave ancillary facilities; equipment pricing for autoclave and autoclave ancillary facilities; quantity estimation for autoclave and autoclave ancillary facilities; capital cost estimate for autoclave and autoclave ancillary facilities; operating cost estimate for autoclave and autoclave ancillary facilities; logistics plan for delivery of autoclave
Lorax Environmental Services Ltd.	Water quality modelling for the mine pit lake
SRK	Acid rock drainage (ARD) and metal leaching (ML) assessment; closure cost estimate
Rowland Engineering Consultants	Geotechnical investigations to support port site, airstrip, and material borrow sources; geotechnical engineering for access roads between port site, airstrip, and plant site

2.2 Qualified Persons

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

- Gordon Seibel, R.M. SME, Principal Geologist, AMEC Reno
- Kirk Hanson, P.E., Technical Director, Open Pits North America, AMEC Reno
- Tony Lipiec., P.Eng., Manager, Process Engineering, AMEC Vancouver.

2.3 Site Visits and Scope of Personal Inspections

The QPs conducted site visits to the Project as shown in Table 2-2.

Mr Seibel completed a data verification site visit to the Project on 1 October 2008. During the visit, core logs were compared to the core, lithologies in the resource model were compared to the lithologies in the surface outcrops, and core logging, and sampling protocols were reviewed. Handling of the core and sample preparation, however, could not be observed directly as no drilling or sample preparation was being performed during the site visit.

During the October 1, 2008, site visit, Mr Hanson undertook a high-level review of the Project geology, inspected drill core, viewed the Project topography, inspected proposed pit and waste dump locations, and the locations of existing and proposed infrastructure, including road cuts and borrow pits.

In addition to these visits, other AMEC personnel have visited site during preparation of the FSU2 report, and have provided input to the AMEC QPs in the areas of their expertise in support of this Report.

AMEC considers that although completed in 2008, the site visits are still current. Since the date of the last technical report filed on the Project, Donlin Gold has completed an additional 62 drill holes (25,000 m) out of 1,740 holes (370,000 m). This drilling is not considered to comprise a material change to the Project. Changes to the Mineral Resource and Mineral Reserve statements in Section 15 of this Report are primarily driven by the increases to the gold price used in estimation.

Table 2-2: QPs, Areas of Report Responsibility, and Site Visits

Qualified Person	Site Visits	Report Sections of Responsibility (or Shared Responsibility)
Tony Lipiec	No site visit	Sections 1, 2, 3, 4, 5, 6, 13, 14.5.2, 14.6.2, 14.8, 14.9.4, 14.9.5, 14.9.6, 17, 18, 19, 20, 21, 22, 23, 24, 25, 26, and 27
Gordon Seibel	1 October 2008	Sections 7, 8, 9, 10, 11, 12, 14 (excepting 14.5.2, 14.6.2, 14.8, 14.9.4, 14.9.5 and 14.9.6), and those portions of the Summary, Interpretations and Conclusions and Recommendations that pertain to those Sections
Kirk Hanson	1 October 2008	Sections 15 and 16, and those portions of the Summary, Interpretations and Conclusions and Recommendations that pertain to those Sections.

2.4 Effective Dates

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

- Effective date of the assay database that supports estimation: 1 November 2009
- Effective date of the Mineral Resources: 11 July 2011
- Effective date of the Mineral Reserves: 11 July 2011
- Effective date of the tenure and surface rights data: 7 October 2011
- Effective date of the financial analysis that supports the updated feasibility study: 18 November 2011.

The overall effective date of the Report, based on the supply of the financial data, is 18 November 2011.

There has been no material change to the scientific and technical information on the Project between the effective date of the Report, and the signature date.

2.5 Previous Technical Reports

NovaGold has previously filed the following technical reports on the Project:

Francis, K., 2008: Donlin Creek Project, NI 43-101 Technical Report, Southwest Alaska, U.S.: unpublished technical report prepared for NovaGold Resources Inc., effective date 5 February 2008.

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

Dodd, S., 2006: Donlin Creek Project 43-101 Technical Report, unpublished NI43-101F1 Technical Report to NovaGold Resources Inc. by NovaGold Resources Inc., effective date January 19, 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, report date March 2002.

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

2.6 Information Sources and References

The primary data source for this Report is the 2011 Feasibility Study Update 2, entitled:

AMEC Americas Ltd., 2011: Donlin Creek Gold Project, Alaska, Feasibility Study Update 2, Effective Date 7 October 2011: unpublished feasibility study update prepared by AMEC for Donlin Creek LLC, dated 9 October 2011.

Reports and documents listed in the Section 3, Reliance on Other Experts and Section 27, References sections of this Report were also used to support preparation of the Report. Additional information was sought from NovaGold, Barrick, and Donlin Gold personnel where required.

3.0 RELIANCE ON OTHER EXPERTS

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

3.1 Mineral Tenure

The QPs have not reviewed the mineral tenure, nor independently verified the legal status, ownership of the Project area, underlying property agreements or permits. AMEC has fully relied upon, and disclaims responsibility for, information derived from Donlin Gold experts and experts retained by Donlin Gold for this information through the following documents:

- Sellers, T.M., 2009: Legal Opion on Mineral Title: unpublished confidential legal opinion prepared by Reeves Amodio LLC for Donlin Creek LLC, and addressed to James Fueg, Feasibility Study Manager for Donlin Gold LLC, dated 20 March 2009
- Manzer, D.S., 2011: Donlin Gold Project – Administrative Status of State of Alaska Mining Claims, Effective Date October 5, 2011: unpublished title search prepared by Alaska Land Status Inc. for Perkins Coie LLP and addressed to Robert Maynard of Perkins Coie LLP and James Fueg, Feasibility Study Manager for Donlin Gold LLC, dated 6 October 2011
- Manzer, D.S., 2011: Donlin Gold LLC's Donlin Gold Project, Updated Abstract of Record Title, Effective September 2, 2011: unpublished title search prepared by Alaska Land Status Inc. for Perkins Coie LLP and addressed to Robert Maynard of Perkins Coie LLP and James Fueg, Feasibility Study Manager for Donlin Gold LLC, dated 7 October 2011
- Maynard, R.M., 2011: Update of March 20, 2009 Reeves Amodio LLC Title Opinion Letter for Donlin Crcek Project Real Property Interests: unpublished confidential legal opinion prepared by Perkins Coie LLP for Donlin Creek LLC, and addressed to James Fueg, Feasibility Study Manager for Donlin Gold LLC, dated 10 October 2011.

This information is used in Section 4.3 of the Report and was also used to support considerations of reasonable prospects of economic extraction and declaration of Mineral Resources in Section 14.3 and 14.4, and for consideration of appropriate modifying factors for declaration of Mineral Reserves in Section 15.3.

3.2 Surface Rights

The QPs have fully relied upon and disclaim responsibility for information supplied by Donlin Gold staff and experts retained by Donlin Gold for information relating to the status of the current surface rights as follows:

- Sellers, T.M., 2009: Legal Opion on Mineral Title: unpublished confidential legal opinion prepared by Reeves Amodio LLC for Donlin Creek LLC, and addressed to James Fueg, Feasibility Study Manager for Donlin Gold LLC, dated 20 March 2009
- Manzer, D.S., 2011: Donlin Gold Project – Administrative Status of State of Alaska Mining Claims, Effective Date October 5, 2011: unpublished title search prepared by Alaska Land Status Inc. for Perkins Coie LLP and addressed to Robert Maynard of Perkins Coie LLP and James Fueg, Feasibility Study Manager for Donlin Gold LLC, dated 6 October 2011
- Manzer, D.S., 2011: Donlin Gold LLC's Donlin Gold Project, Updated Abstract of Record Title, Effective September 2, 2011: unpublished title search prepared by Alaska Land Status Inc. for Perkins Coie LLP and addressed to Robert Maynard of Perkins Coie LLP and James Fueg, Feasibility Study Manager for Donlin Gold LLC, dated 7 October 2011
- Maynard, R.M., 2011: Update of March 20, 2009 Reeves Amodio LLC Title Opinion Letter for Donlin Crcek Project Real Property Interests: unpublished confidential legal opinion prepared by Perkins Coie LLP for Donlin Creek LLC, and addressed to James Fueg, Feasibility Study Manager for Donlin Gold LLC, dated 10 October 2011.

This information is used in Section 4.5 of the report and for consideration of appropriate modifying factors for declaration of Mineral Reserves in Section 15.3.

3.3 Agreements

The QPs have fully relied upon and disclaim responsibility for information supplied by Donlin Gold staff and experts retained by Donlin Gold or NovaGold for information relating to the status of the current Property Agreements as follows:

- Sellers, T.M., 2009: Legal Opion on Mineral Title: unpublished confidential legal opinion prepared by Reeves Amodio LLC for Donlin Creek LLC, and addressed to James Fueg, Feasibility Study Manager for Donlin Gold LLC, dated 20 March 2009
- Manzer, D.S., 2011: Donlin Gold Project – Administrative Status of State of Alaska Mining Claims, Effective Date October 5, 2011: unpublished title search prepared by Alaska Land Status Inc. for Perkins Coie LLP and addressed to Robert Maynard

of Perkins Coie LLP and James Fueg, Feasibility Study Manager for Donlin Gold LLC, dated 6 October 2011

- Manzer, D.S., 2011: Donlin Gold LLC's Donlin Gold Project, Updated Abstract of Record Title, Effective September 2, 2011: unpublished title search prepared by Alaska Land Status Inc. for Perkins Coie LLP and addressed to Robert Maynard of Perkins Coie LLP and James Fueg, Feasibility Study Manager for Donlin Gold LLC, dated 7 October 2011
- Maynard, R.M., 2011: Update of March 20, 2009 Reeves Amodio LLC Title Opinion Letter for Donlin Crcek Project Real Property Interests: unpublished confidential legal opinion prepared by Perkins Coie LLP for Donlin Creek LLC, and addressed to James Fueg, Feasibility Study Manager for Donlin Gold LLC, dated 10 October 2011.

This information is used in Section 4.4 of the Report and was also used to support considerations of reasonable prospects of economic extraction and declaration of Mineral Resources in Section 14.3 and 14.4, and for consideration of appropriate modifying factors for declaration of Mineral Reserves in Section 15.3.

3.4 Royalties

The QPs have fully relied upon and disclaim responsibility for information supplied by NovaGold staff and experts retained by NovaGold for information relating to the status of the current royalties payable as follows:

- Francis, K., 2012: Confirmation Letter Regarding Royalties, Marketing and Taxation Pool: unpublished letter from Kevin Francis, Vice President Resources, Novagold to Scott Mackin, AMEC Project Manager Donlin Gold Project, 6 January 2012

This information is used in Section 4.7 of the report and was also used to support considerations of reasonable prospects of economic extraction and declaration of Mineral Resources in Section 14.3 and 14.4, and for consideration of appropriate modifying factors for declaration of Mineral Reserves in Section 15.3.

3.5 Marketing

The QPs have fully relied upon and disclaim responsibility for information supplied by NovaGold for information relating to the status of the potential Project marketing regime as follows:

- Francis, K., 2012: Confirmation Letter Regarding Royalties, Marketing and Taxation Pool: unpublished letter from Kevin Francis, Vice President Resources, Novagold to Scott Mackin, AMEC Project Manager Donlin Gold Project, 6 January 2012.

This information is used in Section 19, and was used to support considerations of reasonable prospects of economic extraction and declaration of Mineral Resources in Section 14.3 and 14.4, declaration of Mineral Reserves in Section 15.3, and the cashflow analysis in Section 22.

3.6 Taxation

The QPs have fully relied upon and disclaim responsibility for information supplied by NovaGold for information relating to the status of the potential taxation pool available to NovaGold through the following:

- Francis, K., 2012: Confirmation Letter Regarding Royalties, Marketing and Taxation Pool: unpublished letter from Kevin Francis, Vice President Resources, Novagold to Scott Mackin, AMEC Project Manager Donlin Gold Project, 6 January 2012.

This information is used in Section 22.5 of the Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Donlin deposits are situated approximately 280 miles (450 km) west of Anchorage and 155 miles (250 km) northeast of Bethel up the Kuskokwim River. The closest village is the community of Crooked Creek, approximately 12 miles (20 km) to the south, on the Kuskokwim River.

The resource areas are within T. 23 N., R. 49. W., Seward Meridian, Kuskokwim and Mt. McKinley Recording Districts, Crook Creek Mining District, Iditarod A-5 USGS 1:63,360 topography map. The mineralization is centred on approximately 540222.50 east and 6878534.36 north, using the NAD 83 datum.

These areas consist of the ACMA and 400 Zone, Aurora and Akivik mineralized areas (grouped as ACMA) and the Lewis, South Lewis, Vortex, Rochelieu and Queen mineralized areas (grouped as Lewis). The final proposed pit outline for the combined ACMA and Lewis area in relation to the Calista Lease Boundar y (refer to Section 4.3) is shown as Figure 4-1. Key deposit and prospect areas are shown in Figure 4-2.

4.2 Project Ownership History

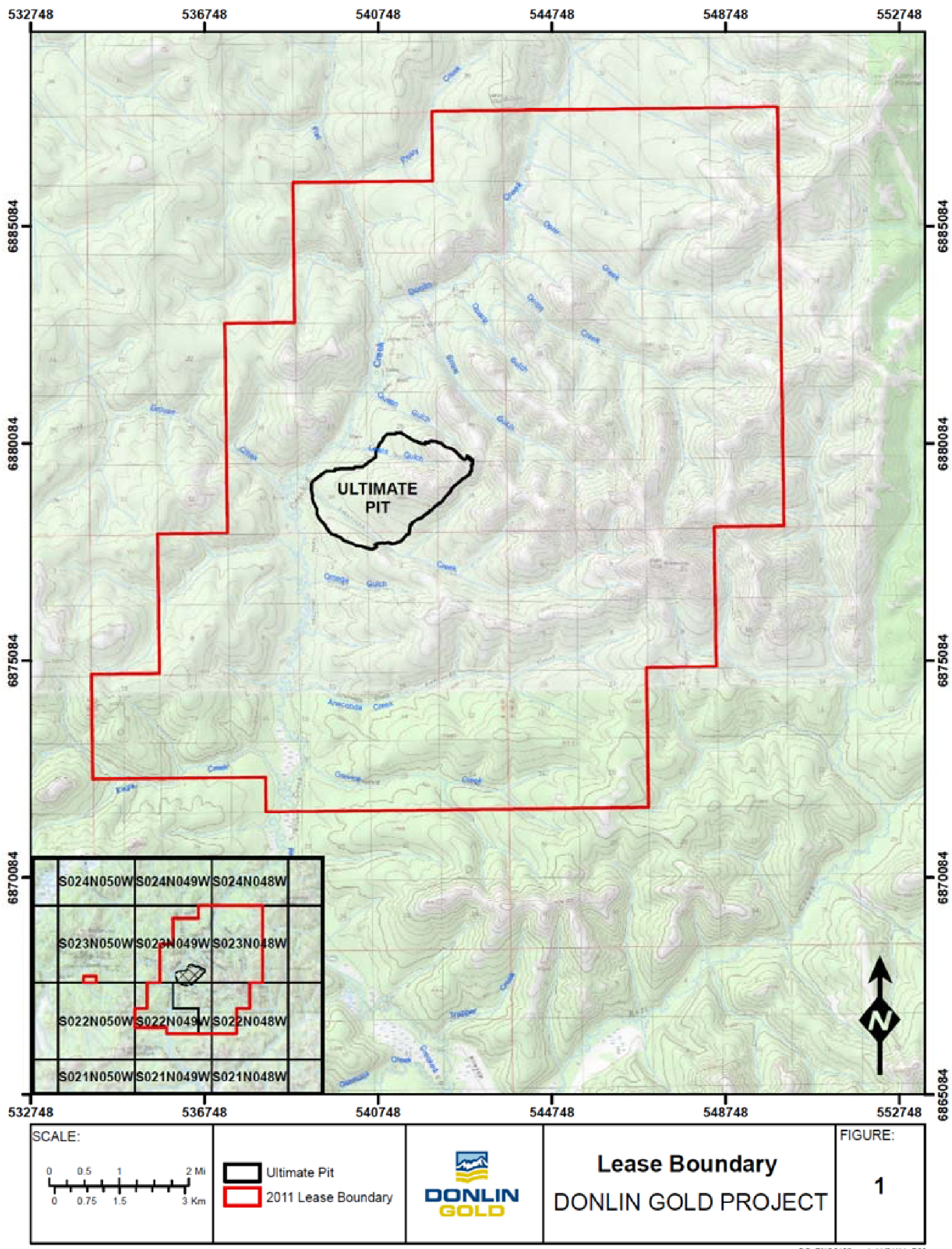
Calista Corporation, an Alaska Native Corporation, has held the mineral rights to the Project since 1974.

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.

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. Placer Dome retained an option to buy back into the Project.

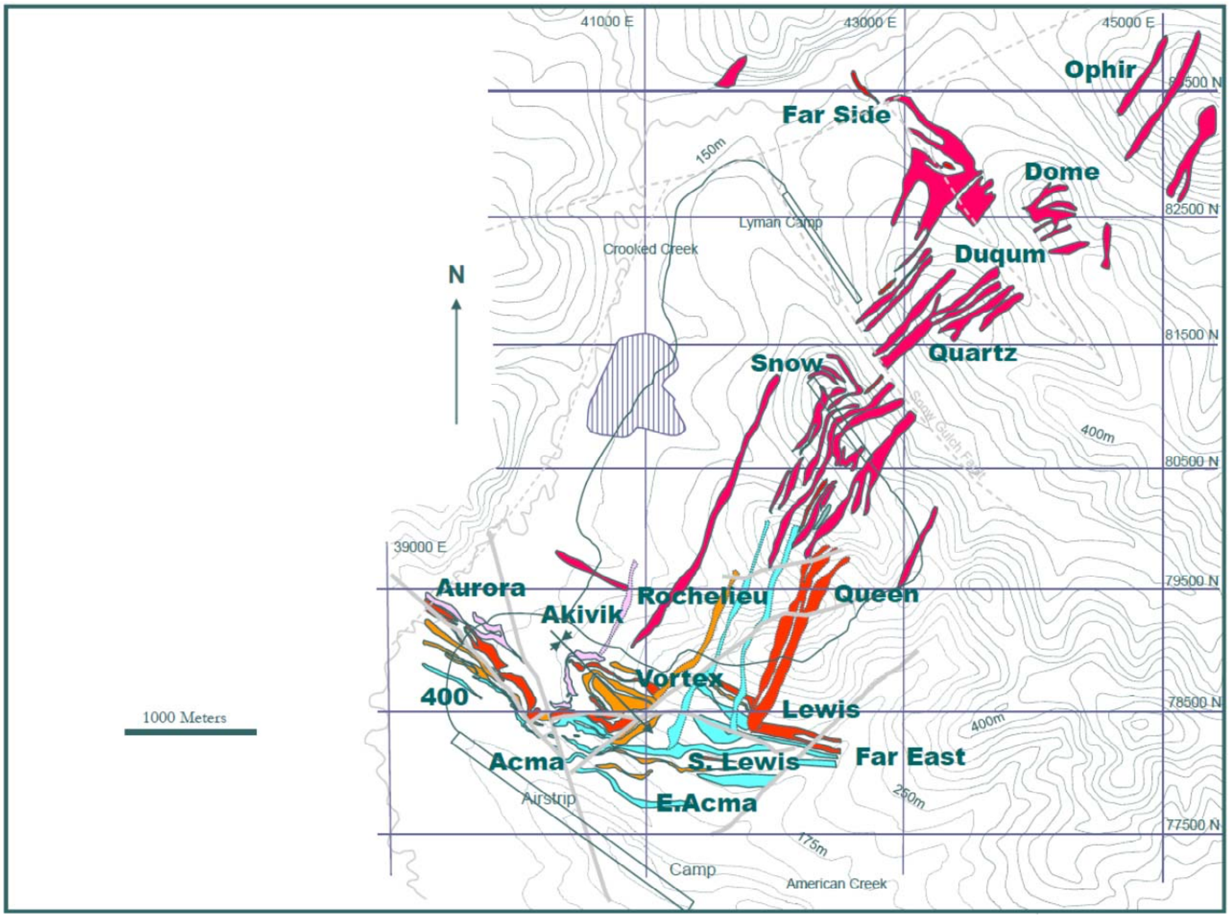
On February 11, 2003, Placer Dome exercised its back-in right and assumed management of the continued development of the Project. In January 2006, Barrick acquired Placer Dome and assumed Placer Dome's joint venture responsibilities with regard to Donlin.

Figure 4-1: Proposed Pit Location in Relation to Calista Lease Boundary



Note: Figure courtesy Donlin Gold

Figure 4-2: Key Deposit and Prospect Areas



Note: Figure courtesy NovaGold, figure dated September 2011. Coloured portions indicate different generations of intrusive dikes.

On December 1, 2007, NovaGold entered into a limited liability company agreement with Barrick (the Donlin Creek LLC Agreement) that provided for the conversion of the Project into a new limited liability company, the Donlin Creek LLC, which is jointly owned by NovaGold and Barrick on a 50/50 basis. In July 2011, the Board of Donlin Creek LLC voted to change the name of the company to Donlin Gold LLC.

As part of the Donlin Creek LLC Agreement, NovaGold has agreed to reimburse Barrick over time for approximately US$64.3 million, representing 50 percent of Barrick's approximately US$128.6 million expenditures at the Project from April 1, 2006 to November 30, 2007. NovaGold reimbursed US$12.7 million of this amount following the effective date of the agreement by paying US$12.7 million of Barrick's share of Project development costs. The remaining approximately US$51.6 million will bear interest and be paid out of future mine production cash flow. Funding is currently shared by both parties on a 50/50 basis.

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 13 acres of surface estate within the Snow Gulch area. This lease area lies immediately to the north of the current open pit shell outline but in the opinion of Donlin Gold, 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 mineralization 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.

4.3 Lease Rights

The Donlin exploration and mining lease currently includes a total of 72 sections in the vicinity of the deposit, and additional partial sections associated with the Project infrastructure, leased from Calista Corporation, an Alaska Native Corporation that holds the subsurface (mineral) estate for Native-owned lands in the region. Title to 20,081 hectares (49,261 acres) of the leased lands has been conveyed to Calista by the Federal Government. Calista owns the surface estate on a portion of these lands.

In March 2010, DCLLC renegotiated its lease with Calista. Amendments to the renegotiated lease provide for:

- The lease of certain additional lands that may be required for the development of the property
- An extension of the term of the lease to April 30, 2031 and automatically year to year thereafter, so long as either mining or processing operations are carried out on or with respect to the property in good faith on a continuous basis in such year, or Donlin Gold pays to Calista an advanced minimum royalty of US$3.0 million (subject to adjustment for increases in the Consumer Price Index) for such year
- The elimination of Calista's option to acquire a 5% to 15% participating operating interest in the Project and replacement with the payment to Calista of a net proceeds royalty equal to 8% of the net proceeds realized by Donlin Gold at the Project after deducting certain capital and operating expenses (including an overhead charge, actual interest expenses incurred on borrowed funds and a 10% per annum deemed interest rate on investments not made with borrowed funds)
- An increase in the advanced minimum royalties payable to Calista under the lease to US$0.5 million for the year ending April 30, 2010, increasing on an annual basis

thereafter until reaching US$1.0 million for each of the years 2015 to 2024 inclusive and US$2.0 million for each of the years 2025 to 2030 inclusive. All advance minimum royalties paid to Calista continue to be recoverable as a credit against Calista's existing net smelter royalty under the lease agreement, which remains unchanged.

A separate Surface Use Agreement with The Kuskokwim Corporation (TKC), an Alaska Native Village Corporation that owns the majority of the private surface estate in the area, grants non-exclusive surface use rights to Donlin Gold on at least 34 sections overlying the mineral deposit, with provisions allowing for adjusting that area in conjunction with adjustments to the subsurface included in the Calista lease. The term of the Surface Use Agreement runs through 5 June 2015, with provisions to extend beyond that time so long as mining, processing, or marketing operations are continuing and the Calista lease remains in effect.

The Lyman family owns a small (13 acre) private parcel in the vicinity of the deposit and holds a placer mining lease from Calista that covers approximately four sections.

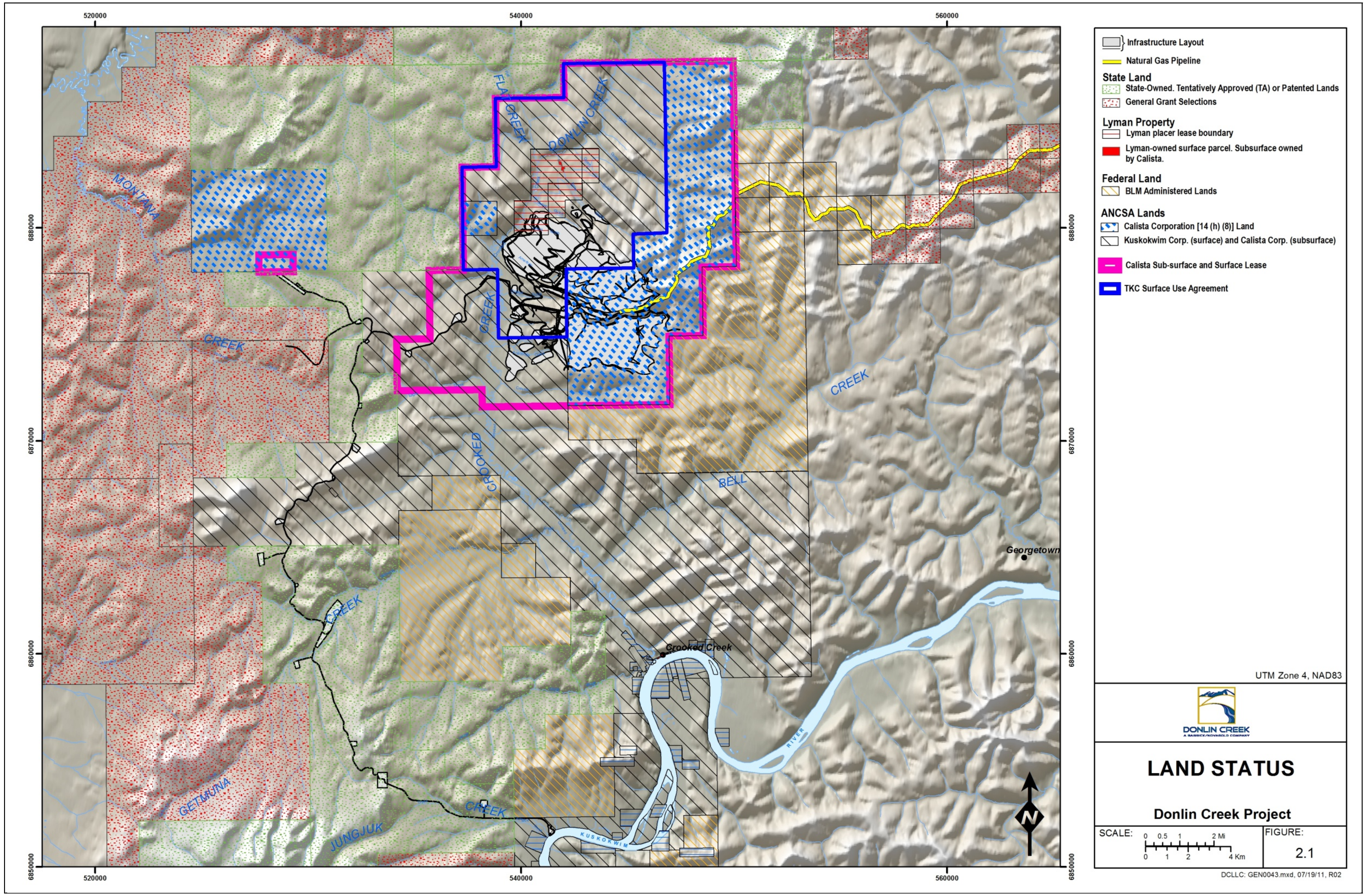
Figure 4-3 shows the land status in the Project area.

4.4 Mineral Tenure

In addition to the 49,261 acres (20,081 hectares) leased from Calista, Donlin Gold holds 242 Alaska State mining claims comprising 31,740 acres (12,845 hectares), bringing the total land package to 81,361 acres (32,926 hectares).

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. Calista reallocated its 12(b) entitlement in 1999, based on a formula that was based on original village corporation enrolments.

Figure 4-3: Donlin Gold Project Land Status Map



Note: Figure courtesy Donlin Gold

The Donlin exploration and mineral lease currently includes lands leased from Calista, which holds the subsurface (mineral) estate for Native-owned lands in the region. The leased land is believed to contain 20,081 ha. Title to all lands has been conveyed to Calista by the Federal Government. Calista owns the surface estate on a portion of these lands.

A separate Surface Use Agreement with TKC, which owns the surface estate of the remaining lands, grants non-exclusive surface use rights to Donlin Gold. All of these lands have been conveyed to TKC by the Federal Government. Donlin Gold has entered into negotiations with the TKC in regards to the Surface Use Agreement, as that agreement expires in 2015.

Donlin Gold holds 242 unpatented State mineral claims totalling 12,845 ha, primarily surrounding the leased land in the Kuskokwim and Mount McKinley recording districts. Of these claims:

- Three are on State-selected lands
- A total of 158 are tentatively approved from conveyance from Federal to State-owned, pending survey.

None of the claims held by Donlin Gold have been surveyed.

All claims are either 64.8 ha or 16.2 ha in size.

4.5 Surface Rights

Donlin Gold, through native lease agreements, holds a significant portion of the surface rights that will be required to support mining operations in the proposed mining area. Negotiations with TKC will be required for surface rights for additional lands supporting mining and access infrastructure.

Currently, Donlin Gold operates under the Mining Lease with the Calista Corporation and the Surface Use Agreement with 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 exclusive-use property.

Drilling operations for the Project are covered under the Alaska Placer Mining process and related permits.

Other lands required for off-site infrastructure, such as the gas pipeline, port site, and access road, are categorized as Native, State of Alaska conveyed, or BLM (Federal) lands.

4.6 Royalties and Encumbrances

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

- Annual advance minimum royalty increasing to $650,000 in 2014
- Annual advance minimum royalty of $1 million from 2015 to 2024
- Annual advance minimum royalty of $2 million from 2025 to 2030
- Net smelter return of 1.5% for the earlier of the first five years following commencement of commercial production or until payback
- Conversion to 4.5% after the earlier of five years or payback
- Net proceeds royalty of 8% of the net proceeds realized by Donlin Gold commencing with the first quarter in which net proceeds are first realized
- Calista shareholder hire preference and Calista 5% bidding preference on competitive contracts for all work on the property leased from Calista.

Currently there are no Government royalty obligations, and no other royalties are payable on the Project.

4.7 Permits

Donlin Gold advised AMEC that Barrick 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).

Permits required to support Project development are discussed in Section 20.

4.8 Environmental Liabilities

Environmental studies, closure plans and costs, and environmental liabilities and issues are discussed in Section 20.

4.9 Social License

The potential social and community impact assessments of the Project are discussed in Section 20.

4.10 Significant Risk Factors

Based on Project design concepts developed through the prefeasibility and feasibility engineering work and on the results of four years of community interaction, several key environmental issues of concern have been identified for the Project. These are discussed in detail in Section 20.

4.11 Comments on Section 4

In the opinion of the QPs, the following conclusions are appropriate:

- Information from Donlin Gold and legal experts supports that the mining tenure held is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves. Mineral tenures have not been surveyed. Appropriate claim payments have been made as required and required expenditure commitments had been met by Donlin Gold.
- Information from Donlin Gold indicates that surface rights are held by either TKC or Calista. This information supports that Donlin Gold has agreements with both parties as follows. Calista owns the surface estate on 27 of the 72 sections that make up the Project. TKC has granted Donlin Gold non-exclusive surface use rights to Donlin Gold on at least 34 sections overlying the mineral deposit, with provisions allowing for adjusting that area in conjunction with adjustments to the subsurface included in the Calista lease. The Lyman family owns a small (13 acre) private parcel in the vicinity of the deposit and holds a placer mining lease from Calista that covers approximately four sections. The currently identified resource and the bulk of the primary infrastructure (mill and waste rock facilities) are located on the leased lands. Additional lands required for the Jungjuk port site, road to the port site, gas pipeline, and tailings storage facility in Anaconda Creek are located on a combination of Native, State of Alaska, and Federal (Bureau of Land Management, BLM) lands. Rights-of-way will be required from the State and BLM

for the road and pipeline alignments where they cross state and federal lands, respectively. Discussions regarding the extension and expansion of the TKC Surface Use Agreement and the disposition of the Lyman family land parcel and lease are ongoing.

- Information from Donlin Gold indicates that royalty payments are associated with the Calista lease as follows:
  - annual advance minimum royalty increasing to $650,000 in 2014
  - annual advance minimum royalty of $1 million from 2015 to 2024
  - annual advance minimum royalty of $2 million from 2025 to 2030
  - net smelter return of 1.5% for the earlier of the first five years following commencement of production or until payback
  - conversion to 4.5% after the earlier of five years or payback
  - net proceeds royalty of 8% of the net proceeds realized by Donlin Gold commencing with the first quarter in which net proceeds are first realized.
- There are no Government royalty obligations
- Information from Donlin Gold indicates that a payment of an annual Aggregate Surface Use Fee of $50,000 is required to TKC, and either, once exclusive-use lands are identified, payment of an annual exclusive-use fee of 10% of the fair market value of the property, or purchase of the exclusive-use lands
- Exploration to date has been conducted in accordance with Alaskan regulatory requirements
- Additional permits will be required for Project development.
- Key areas identified by stakeholders (refer to Section 20) as areas of social and environmental concern include mercury and cyanide abatement, waste rock and water treatment and management, transportation of goods and materials to and from the Project site, and flora and fauna management. Donlin Gold is of the opinion that these issues have been, or can be, addressed and mitigated through a combination of good baseline data collection, diligent engineering and Project design, and thorough public consultation.

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

5.1 Accessibility

The Donlin deposits are approximately 280 miles (450 km) west of Anchorage and 155 miles (250 km) northeast of Bethel up the Kuskokwim River. The closest village is the community of Crooked Creek, approximately 12 miles (20 km) to the south, on the Kuskokwim River. Bethel, approximately 20 miles (30 km) upstream from the mouth of the Kuskokwim River, is the regional centre for the Yukon-Kuskokwim Delta area of Southwest Alaska. The town of Aniak, also on the Kuskokwim River and about 50 air miles (80 km) southwest of the Project site, is the regional centre for the Upper Kuskokwim Valley.

There is no road or rail access to the site. The nearest roads are in the Anchorage area. Access to Bethel and Aniak, the regional centres, is limited to river travel by boat or barge in the summer and air travel year-round. The Kuskokwim River is a regional transportation route and is serviced by commercial barge lines.

All access to the Project site for personnel and supplies is by air. The Project has an all-season, soft-sided camp with facilities to house up to 150 people. An adjacent 5,000 ft (1,500 m) long airstrip is capable of handling aircraft as large as C-130 Hercules (42,000 lb or 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 about 20 inches (500 mm) of total annual precipitation. Summer temperatures are relatively warm and may exceed 83°F (30°C). Minimum temperatures may fall to well below -45°F (-42°C) during the cold winter months.

Exploration is possible year round, though snow levels in winter and wet conditions in late autumn and in spring can make travel within the Project area difficult. It is expected that mining operations will be able to be conducted year-round.

5.3 Local Resources and Infrastructure

Local resources necessary for the exploration and possible future development and operation of the Project are located in Bethel. Some resources would likely have to be brought in from the Anchorage area.

Alaska and the adjacent Canadian Province of British Columbia have a long mining history and a large resource of equipment and skilled personnel.

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.

Infrastructure assumptions and the proposed infrastructure layout for the Project are discussed in Section 18 of the Report.

5.4 Physiography

The Project area is one of low topographic relief on the western flank of the Kuskokwim Mountains. Elevations range from 500 to 2,100 ft (150 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 and along north-facing slopes.

5.5 Sufficiency of Surface Rights

In regard to future mining operations, sufficient space is available to site the various facilities, including personnel housing, stockpiles, tailing storage facility, waste rock storage facilities and processing plants.

5.6 Comments on Section 5

In the opinion of the QPs:

- The existing and planned infrastructure, availability of staff, the existing power, water, and communications facilities, the design and budget for such facilities, and the methods whereby goods could be transported to any proposed mine, and any planned modifications or supporting studies are reasonably well-established, or the requirements to establish such, are reasonably well understood by Donlin Gold, NovaGold and Barrick, and can support the declaration of Mineral Resources and Mineral Reserves.

- There is sufficient area within the Project to host an open pit mining operation, including the proposed open pits, mine and plant infrastructure, waste rock and tailings storage facilities.
- It is a reasonable expectation that any additional surface rights to support Project development and operations can be obtained.
- It is expected that any future mining operations will be able to be conducted year-round.

6.0 HISTORY

Placer gold was first discovered at Snow Gulch, a tributary of Donlin Creek, in 1909. Intermittent small-scale placer gold production has continued to the present. Resource Associates of Alaska (RAA) carried out a regional evaluation for Calista in 1974 to 1975. This work included a soil grid and three bulldozer trenches in the Snow area immediately north of the current resource area. Calista followed up with prospecting activities between 1984 and 1986, and completed minor auger drilling in 1987.

The first substantial exploration drill program was carried out by Western Gold Exploration and Mining Co. LP (WestGold) in 1988 and 1989. WestGold completed geological mapping, trenching, rock and soil sampling, an airborne magnetic and VLF survey, and ground magnetic surveys. WestGold also tested biogeochemical sampling and ground penetrating radar with positive results. Based on this information, WestGold performed an initial Mineral Resource estimate.

Teck Exploration Ltd. (Teck) carried out a limited trenching and soil sampling program in the Lewis area in late 1993, and updated the Mineral Resource estimate.

Placer Dome US (Placer Dome) explored the property from 1995 to 2000. Placer Dome constructed an exploration camp and airstrip, undertook reconnaissance and geological mapping, aerial photography, completed rock chip and soil sampling, trenching, max-min (EM) geophysical surveys, airborne geophysical surveys, RC and core drilling, carried out detailed metallurgical test work, and prepared a series of Mineral Resource estimates and initial mining and engineering studies.

Placer Dome formed the Donlin Creek joint venture (DCJV) with NovaGold Resources, Inc. as operator in 2001. During the period of the DCJV, NovaGold undertook trenching, core and geotechnical drilling, updated Mineral Resource estimates, and completed a Preliminary Assessment.

Placer Dome reassumed management of the Project as operator in late 2002. From 2002 to 2005, work comprised additional core drilling, condemnation, geotechnical, and water drilling, geotechnical and hydrogeological studies, geological mapping and sampling of prospective calcium carbonate source areas, exploration and auger drilling program for sand and gravel resources, and updated Mineral Resource estimates.

Barrick Gold (Barrick) acquired the Placer Dome interest in the DCJV through a merger with Placer Dome in early 2006. Work completed in the period 2006-2007 included core drilling for resource infill, geotechnical, engineering, condemnation, waste rock, calcium carbonate exploration and metallurgical purposes, and updated Mineral Resource estimates.

The DCJV partners formed DCLLC in late 2007, with the subsequent name change to Donlin Gold occurring in 2011.

To 2011, work completed has consisted of soil and stream sediment sampling, core drilling for resource infill, geotechnical, engineering, condemnation, waste rock, and metallurgical purposes, and estimation of Mineral Resources and Mineral Reserves.

An initial feasibility study was completed on the Project in 2007, and updated in 2009. A second update was performed in 2011, and is the subject of this Report.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Donlin gold deposits lie in the central Kuskokwim basin of southwestern Alaska, and is a northeast-trending basin that subsided between a series of amalgamated terranes. Rocktypes within the basin include Mesozoic marine volcanic rocks, Palaeozoic clastic and carbonate rocks, and Proterozoic metamorphic rocks.

The Kuskokwim basin is predominately underlain by the Upper Cretaceous Kuskokwim Group, a back-arc continental margin basin fill assemblage that formed in response to a change in the angle of convergence between the Kula oceanic plate and the Cretaceous North American continental margin. Sediments primarily consist of a coarse- to fine-grained turbidite comprising sandstone, siltstone, and shale with minor conglomerate.

Late Cretaceous and Early Tertiary volcano-plutonic complexes intrude and overlie the Kuskokwim Group sedimentary rocks. Volcanic components of these complexes consist of intermediate tuffs and flows. Subaerial volcanic tuffs, flows, and domes are regionally extensive and dominantly andesitic, locally include dacite, rhyolite, and basalt. Associated plutons are calc-alkaline in composition, ranging from monzonite to granodiorite. Felsic to intermediate hypabyssal granite to granodiorite porphyry dikes, sills, and plugs are also widely distributed and often intruded into northeast-striking extensional faults. Volumetrically minor Upper Cretaceous intermediate to mafic intrusive bodies are also common.

The centre of the Kuskokwim basin lies between two continental-scale, dextral slip-fault zones: the Denali–Farewell Fault system to the south and the Iditarod–Nixon Fork Fault system to the north. Fold-and-thrust-style deformation formed the earliest structures in response to subduction-related compression shortly after deposition of the Kuskokwim sediments. Eastward-trending folds and thrust faults are common in the central Kuskokwim basin, including the Donlin Creek area. Younger north–northeast-trending folds are dominant near the Iditarod–Nixon Fork Fault and Denali–Farewell Fault but also formed throughout the region in response to basin-scale dextral movement. Most of the folds predate emplacement of the volcano-plutonic complexes. Pre-, syn-, and post-(?) intrusion, northeast-striking normal and oblique slip faults formed during subsequent late compressional and extensional events and focused intrusive igneous rocks and hydrothermal systems across the basin.

A regional geological plan is included as Figure 7-1.

Figure 7-1: Regional Geology of Central Kuskokwim Area

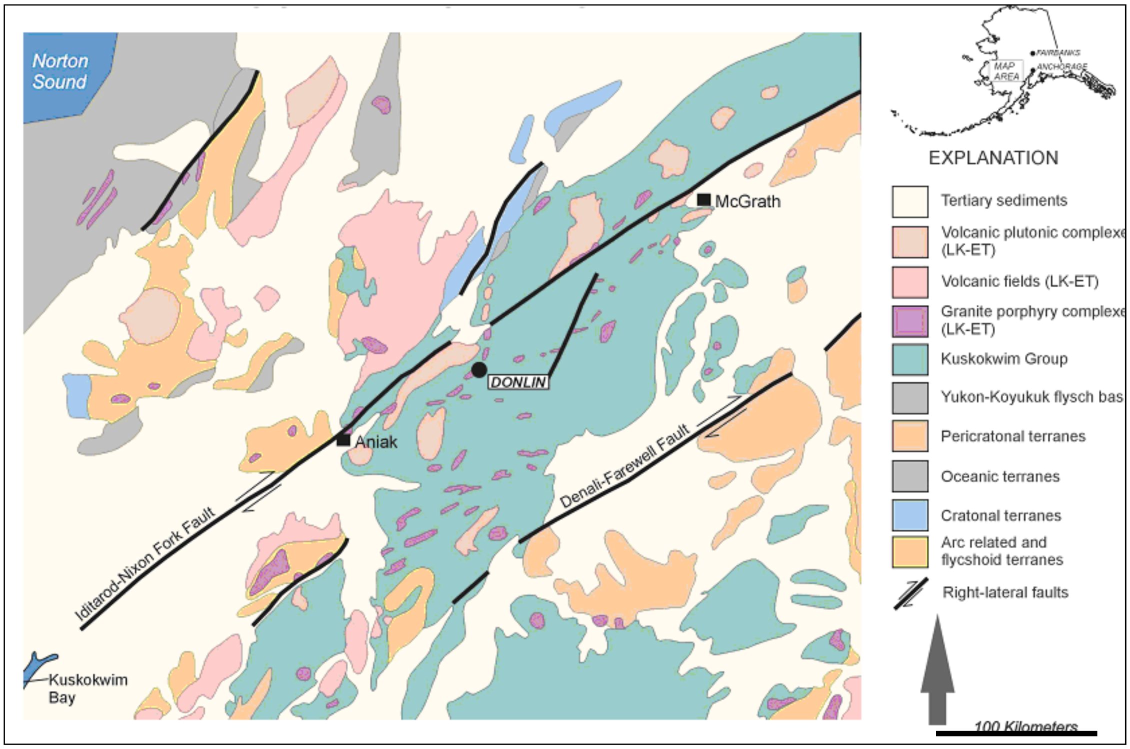


Figure courtesy Donlin Gold.

7.2 Project Geology

Because outcrop is limited and of generally poor quality, property-scale geology is largely interpreted from trenches, drill holes, aeromagnetic surveys, and soil geochemistry.

The Project area is underlain by a 5 mile (8.5 km) long x 1.5 mile (2.5 km) wide granite porphyry dike and sill swarm hosted by lithic sandstone, siltstone, and shale of the Kuskokwim Group.

7.2.1 Lithologies

The oldest igneous rocks at Donlin Creek are intermediate to mafic dikes and sills. They are not abundant but occur widely throughout the property as generally thin and discontinuous bodies. The younger and much more voluminous granite porphyry intrusive rocks vary from a few feet to 200 ft (60 m) wide and occur as west–northwest-trending sills in the southern resource area and north–northeast-trending dikes farther north. The granite porphyry dikes and sills all have similar mineralogy, and the porphyry texture indicates relatively shallow emplacement. Although these rocks belong to the regionally important granite porphyry igneous event, geologists working on the Project classify them into five textural varieties of rhyodacite. These units are chemically similar, temporally and spatially related, and probably reflect textural variations of related intrusive events.

Figure 7-2 illustrates the interpreted property-scale distribution of igneous rocks, including the mineral resource area between the Queen deposit area on the northeast and the airstrip on the southwest.

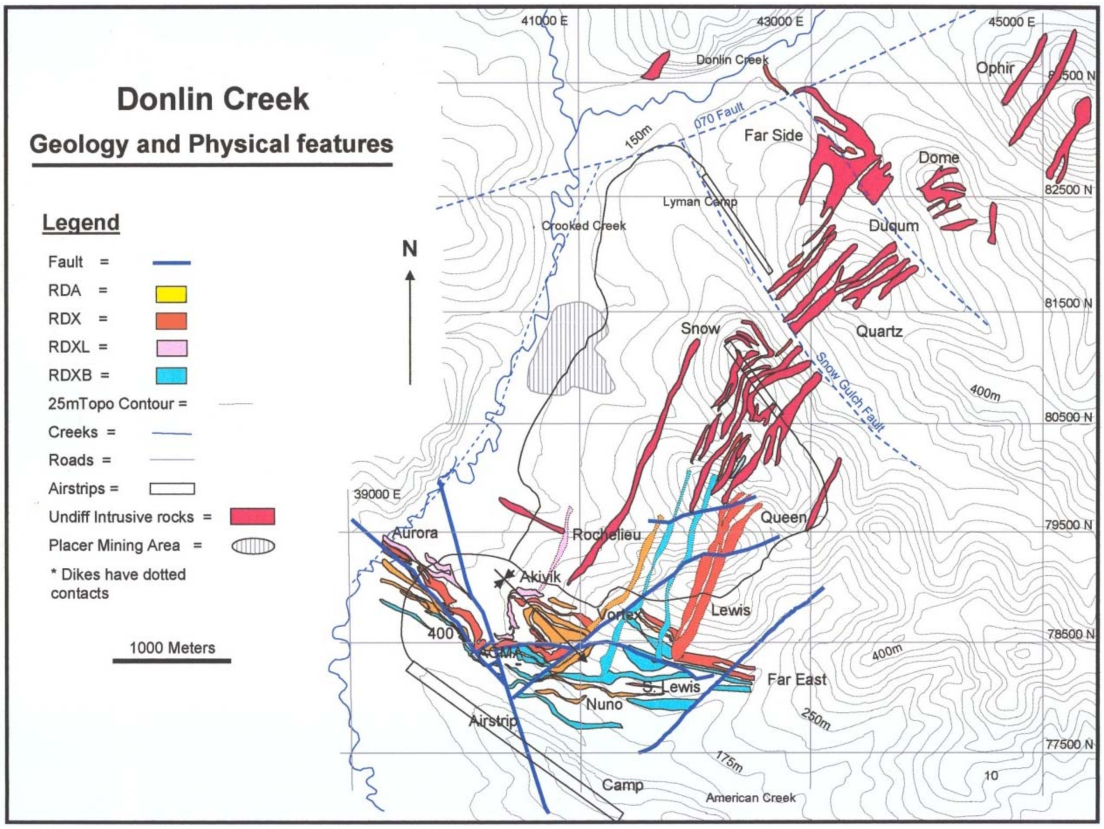
7.2.2 Structure

The Project is located in a structurally complex area about 15 miles (25 km) southeast of the Iditarod–Nixon Fault (refer to Figure 7-1). Sedimentary bedding generally strikes northwest and dips 10° to 50° to the southwest. Overall, sedimentary structure in the northern resource area is monoclinal, while sedimentary rocks in the southern resource area display open eastward-trending folds. East–southeast-trending and plunging folds or monoclinal warps are the oldest recognized structures and are associated with north-vergent thrust faults. Thrust faults are generally southwest-dipping, parallel to the bedding plane, and account for imbrication of the sedimentary rocks and locally moderate to steep southwest and northeast dips. Younger, low-amplitude north–northeast-trending folds crop out in the airstrip exposures along American Creek and are recorded on historic trench geology maps. Lack of cleavage or other evidence of dynamic recrystallization suggests that folds and thrust faults formed at relatively shallow depths.

7.3 Deposit Setting

A northeast, elongated, roughly 5,000 ft (1.5 km) wide x 10,000 ft (3 km) long cluster of gold deposits has an aggregate vertical range that exceeds 3,100 ft (945 m). The deposits are hosted primarily in igneous rocks, and are associated with an extensive Upper Cretaceous gold–arsenic–antimony–mercury hydrothermal system. Gold occurs primarily in sulphide and quartz–carbonate–sulphide vein networks in igneous rocks and, to a much lesser extent, in sedimentary rocks. Broad disseminated sulphide zones formed in igneous rocks where vein zones are closely spaced. Sub-microscopic gold, contained primarily in arsenopyrite and secondarily in pyrite and marcasite, is associated with illite–kaolinite–carbonate–graphite-altered host rocks.

Figure 7-2: Interpreted Property-Scale Igneous Rocks



Note: RDA = Aphanitic Porphyry; RDX = Crowded Porphyry; RDXL = Lath-Rich Porphyry; and RDXB = Blue Porphyry. Figure courtesy Donlin Gold.

7.4 Paragenesis

Fluid inclusion studies and field and drill hole observations define three distinct styles of gold mineralization that are locally telescoped and cross-cut one another. The earliest is a porphyry-style stockwork vein system at the Dome prospect.

Dome is located within the same dike-and-sill swarm that hosts the ACMA–Lewis resource, but the Kuskokwim sedimentary rocks are thermally metamorphosed to a siliceous hornfels. Quartz veins have a Au–Ag–Cu–Zn–Bi ± Te trace metal signature (Ebert et al., 2003c; Drexler, 2010) with up to 3% arsenopyrite–pyrite–chalcopyrite–pyrrhotite ± Fe-rich sphalerite and trace amounts of electrum, native bismuth, and bismuth tellurides and selenides. Veins cut both the hornfels and porphyry dikes.

ACMA–Lewis-style mineralization post-dates the Dome veins and consists of sparse Au–Ag–As–Sb–Hg ± W (Ebert et al., 2003c; Drexler, 2010), trace metal-bearing quartz–Fe–dolomite veins with <3% auriferous arsenopyrite–pyrite ± stibnite ± late realgar, native arsenic, and graphite. Veins and related disseminated sulphide zones are primarily hosted in illite–carbonate–kaolinite-altered rhyodacite dikes and sills but also occur in Kuskokwim Group sedimentary rocks near igneous contacts.

Variations between Dome and ACMA–Lewis vein habits, vein mineralogy, wall rock alteration, geochemical signatures, stable isotope variations (Drexler, 2010), and fluid inclusion chemistry (Ebert et al., 2003c) indicate that hydrothermal fluids were sourced at depth northeast of the Dome prospect, precipitated the base metal assemblage at Dome from metals sequestered in the vapour phase, and then migrated southwestward to the more distal ACMA–Lewis environment, where gold-bearing minerals were precipitated due to mixing with meteoric waters and boiling.

The last event consists of gold-bearing quartz–stibnite veins up to 3 ft (1 m) thick with variable carbonate, pyrite, and arsenopyrite found mainly around the margins of Dome and partially overlapping ACMA–Lewis. Quartz–stibnite veins also contain anomalous Au–As–Cu–Zn–Bi and have fluid chemistry and temperatures intermediate between Dome and ACMA–Lewis (Ebert et al., 2003). In the opinion of Donlin Gold, these veins do not contain significant gold mineralization.

7.5 Deposit Geology

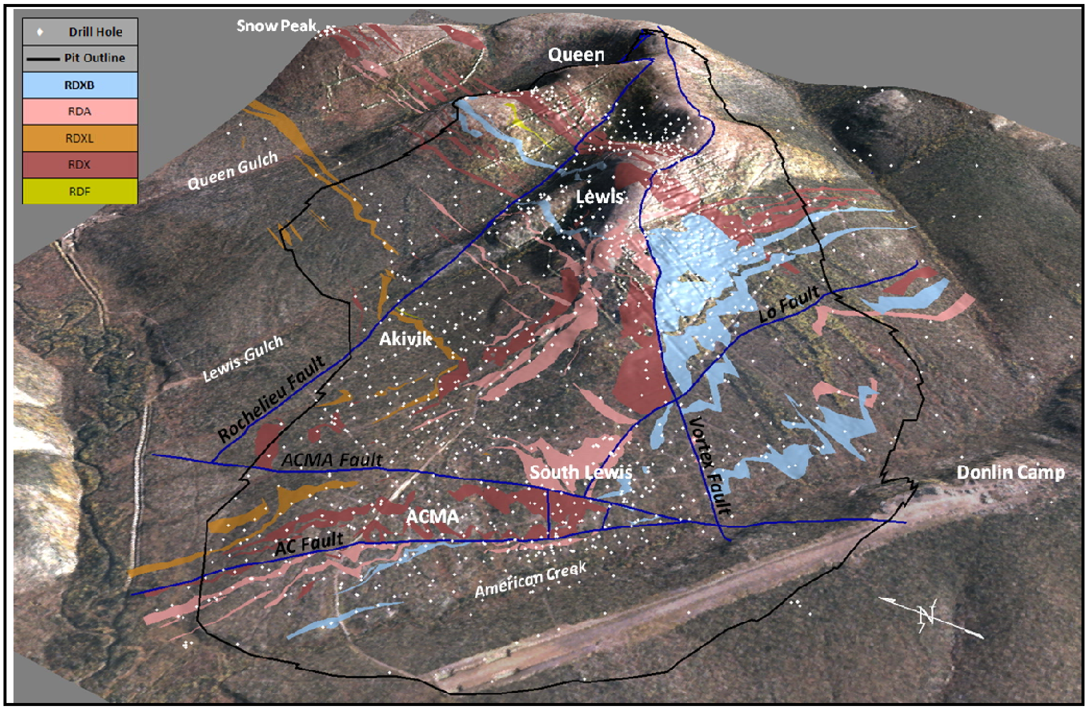
Most of the detailed trench, road cut, and outcrop maps have not yet been compiled into a geological "fact map" in the resource area. The surface geology illustration in Figure 7-3 is a projection of the 3D geological model of intrusive rock units and faults shown in a perspective view of an orthophoto-draped digital elevation model (DEM) image.

7.5.1 Sedimentary Rocks

Informal sedimentary stratigraphy in the immediate deposit area is shown in Table 7-1. The approximate thicknesses of each unit are from the southern, or ACMA, resource area.

The stratigraphy in the deposit area consists of basin margin clastic rocks (MacNeil, 2009) dominated by greywacke (lithic sandstone) units with complex transition zones of interbedded siltstone, shale, and greywacke. Marker beds are not yet recognized, so absolute stratigraphic breaks are difficult to identify. Greywacke is dominant in the northern part of the resource area (Lewis, Queen, Rochelieu, Akivik), whereas shale–siltstone-rich units are common in the southern part (South Lewis, ACMA).

Figure 7-3: Interpreted Surface Geology of Resource Area



Note: Oblique view looking northeastward showing igneous rock units, faults, drill holes, and Mineral Reserve (DC9) pit outline. Figure courtesy Donlin Gold

Table 7-1: Donlin Gold Project Stratigraphy

Assigned Nomenclature	Principal Rock Type	Apparent Thickness (ft)	Apparent Thickness (m)
Upper Greywacke	greywacke	328+	100+
Upper Siltstone	siltstone/shale	164	50
Main Greywacke	greywacke	262	80
Main Shale	shale/siltstone	up to 459 (with sills)	up to 140 (with sills)
Basal Greywacke	greywacke	656	200+

Note: After Piekenbrock and Petsel (2003)

7.5.2 Igneous Rocks

The mafic igneous rocks and the five textural varieties of rhyodacite recognized in the Donlin deposits were also shown in Figure 7-3. Table 7-2 lists the intrusive rocks, also from oldest to youngest.

Table 7-2: Donlin Gold Project Intrusive Rocks

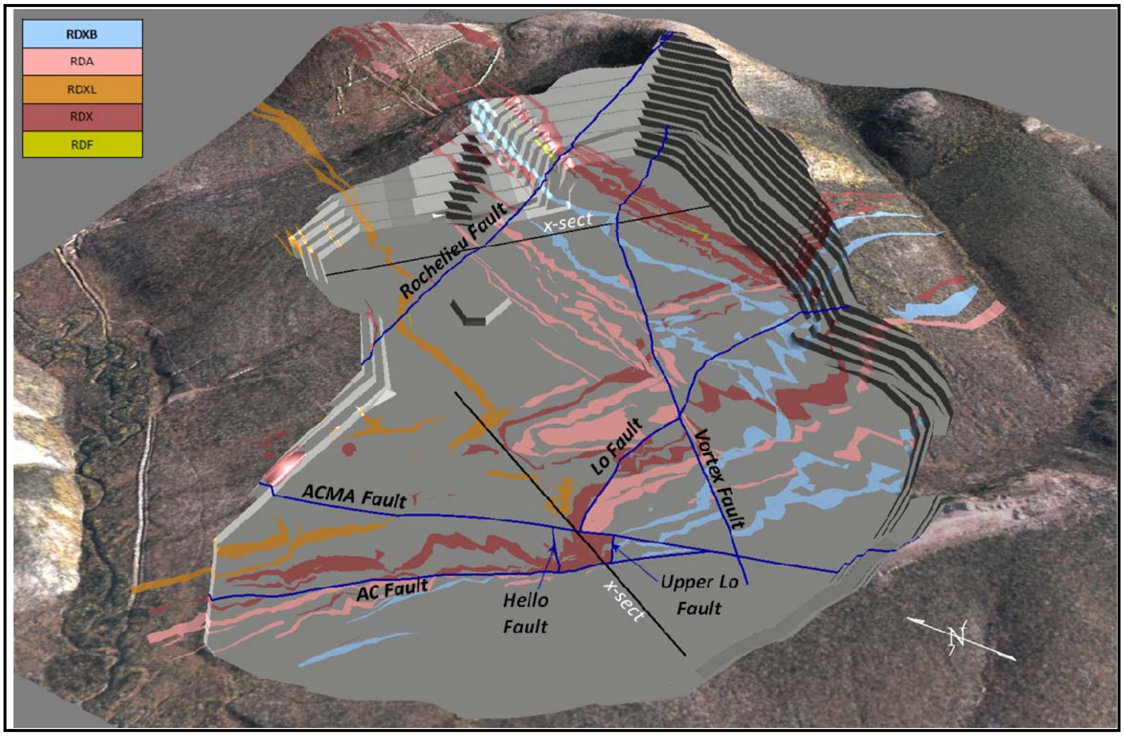
Name	Code	Age	Rock Types
Mafic Dikes/Sills	MD	oldest	Intermediate to mafic dikes and sills; locally host high-grade gold; generally less than 10 ft thick. In the transition area between Akivik and ACMA, mafic sills are extremely abundant within the Lower Greywacke, immediately below the Main Shale
Fine-Grained Porphyry	RDF	-	Earliest rhyodacite intrusions recognized. grey, typically fine-grained, felsic porphyries. RDF intrusives occur as two main northeast-striking, 16.5 to 32.8 ft (5 to 10 m) wide dikes in the Lewis zone and possible discontinuous bodies in early eastward-trending compressional faults, e.g., the Lo Fault
Crowded Porphyry	RDX	-	Volumetrically the most significant intrusive phase. Grey, characterized by a uniformly crowded feldspar porphyry texture. Present as two 164 to 328 ft (50 to 100 m) wide dike zones in the eastern edge of the north to north–northeast mineralized trend of Lewis/South Lewis. RDX is also found as sills throughout ACMA near the basal part of the sill sequence.
Lath-Rich Porphyry	RDXL	-	Characterized by sparse, elongate plagioclase laths; significant coarser-grained biotite. occurs as two important dikes in the Akivik area that strike south into the centre of the ACMA deposit. In Akivik and ACMA, 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 dikes are also present within the main Lewis area RDX dike trend, but here they are volumetrically insignificant
Aphanitic Porphyry	RDA	-	Rhyodacite rock with a salt-and-pepper texture of fine biotite phenocrysts and variable quartz and potassium feldspar phenocrysts. Numerous (up to eight) RDA dikes strike south from the Vortex/Rochelieu (Lewis) area into the East ACMA/ACMA area. The dikes are typically found west of the Vortex Fault but are also present between the Lo and Vortex faults and below the Lo Fault. An extensive sill package of RDA lies 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.
Blue Porphyry	RDXB	youngest	Final intrusive event; coarsely porphyritic with large blocky feldspars set in a graphite- and sulphide-rich matrix. locally hosts important high-grade disseminated sulphide material in addition to gold-bearing veins. RDXB occurs as two major dikes, the Lewis Blue Porphyry dike and the Vortex Blue Porphyry dike. Extensive thin RDXB sills are found in the uppermost part of the sill sequence in the South Lewis and ACMA areas, and RDXB sills are present as both distinct sills and co-mingled with RDA in the core of ACMA and in the Aurora domain.

Note: After Piekenbrock and Petsel (2003)

7.5.3 Structure

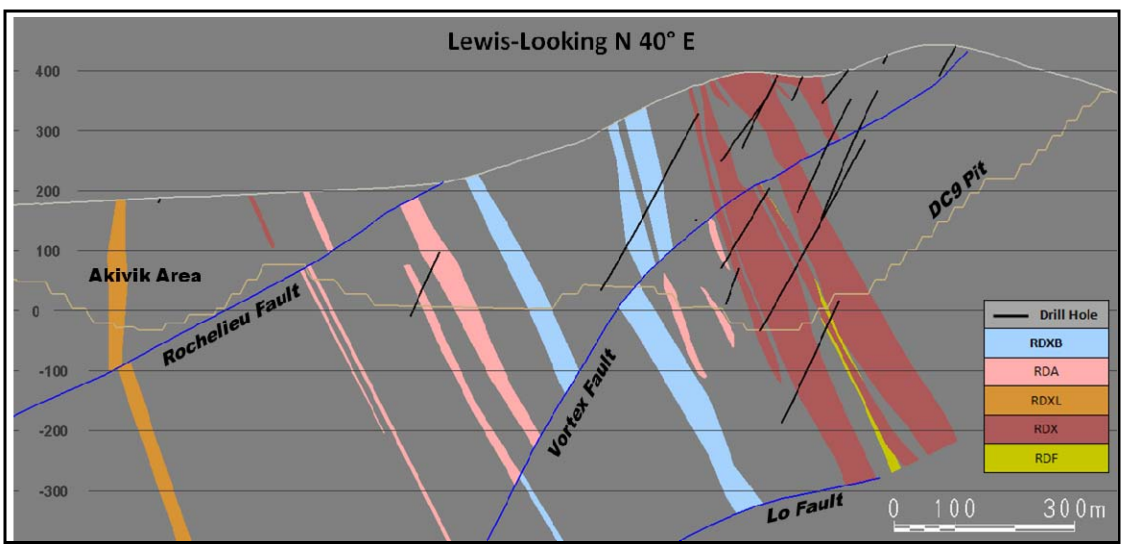
The morphology of intrusive rocks in the deposit is largely governed by the rheology of sedimentary rocks and pre-intrusion faults and folds. Faults in the geological model (from earliest to youngest) are the American Creek (AC) Fault, Lo and Rochelieu faults, Vortex Fault, and ACMA Fault. Figure 7-4 shows a plan view of the faulting in the deposit area, and cross-sections through the ACMA and Lewis areas, respectively in Figure 7-5 and Figure 7-6.

Figure 7-4: 100 m Bench Level Geology



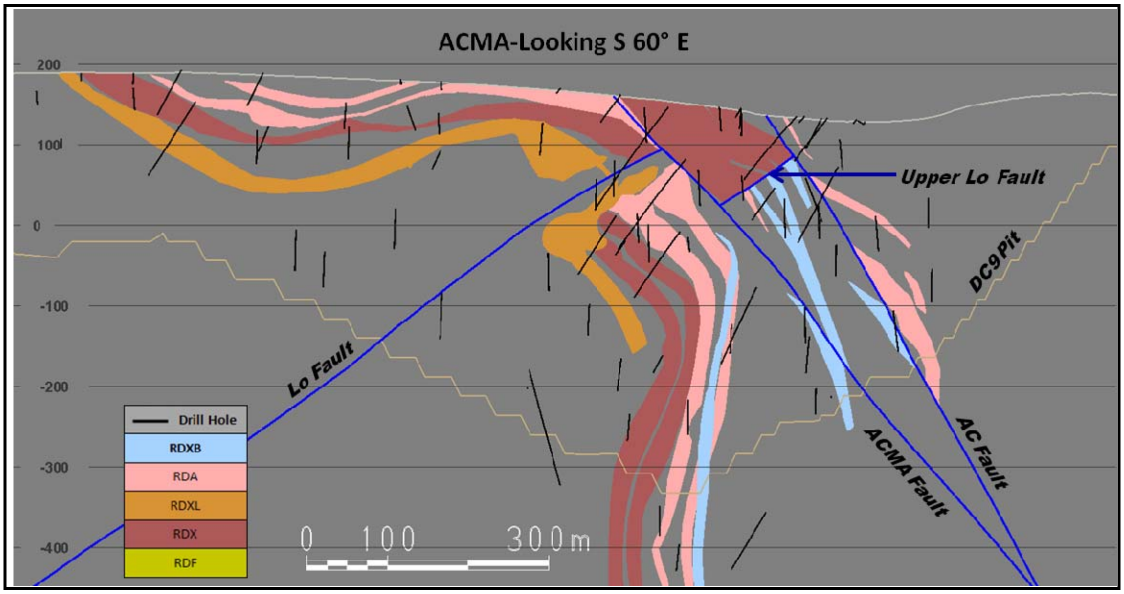
Note: Oblique view looking north-eastward of the 3D geological model projected on the 328 ft (100 m) pit bench level and the Mineral Reserves (DC9) pit outline. Note: Figure courtesy Donlin Gold.

Figure 7-5: Lewis Area Section



Note: Shows intrusive rocks, faults, drill holes, and Mineral Reserves (DC9) pit outline, looking northeast. Note: Figure courtesy Donlin Gold.

Figure 7-6: ACMA Area Section



Note: Shows intrusive rocks, faults, drill holes, and Mineral Reserves (DC9) pit, looking southeast. Note: Figure courtesy Donlin Gold.

7.6 Deposits

The Donlin deposits include eleven mineralized areas that exhibit slightly different geological settings but generally fall into two geologically similar deposit areas: ACMA and Lewis. ACMA, or the intrusive sill and shale–siltstone sedimentary setting, includes the Aurora, 400, Akivik, ACMA, and East ACMA mineralized zones. Lewis, or the massive greywacke-hosted intrusive dike setting, includes the South Lewis, Lewis, Vortex, Rochelieu, Queen, and North Akivik mineralized zones.

Veins in north–northeast-striking, east- or west-dipping faults and fracture zones are the primary control on gold distribution and are ubiquitous in all mineralized areas. Northwest- and northeast-striking veins occur locally but are relatively rare. Veins are narrow (typically <1 cm wide), highly irregular, discontinuous, and generally sparsely distributed, although vein density can locally range up to 2 to 8 per meter in higher-grade zones. Vein zones vary from 6.5 to 100 ft (2 to 35 m) wide and 300 to 1,150 ft (100 to 350 m) long. Individual vein zones generally display limited lateral and vertical continuity; however, swarms of many anastomosing vein zones form larger mineralized corridors characterized by extensive lateral and depth continuity.

Vein corridors are more apparent in the north–northeast-trending dikes of Lewis than in the west–northwest-trending ACMA sill zone. The greater width of the sill-hosted ACMA mineralized zone makes discreet corridors less obvious (but still present).

Mineralized zones follow steeply dipping dikes and sills beyond the depth limits of current drilling, or over a vertical range of at least 3,100 ft (945 m).

Veins are best developed in relatively more brittle intrusive rocks and massive greywacke. Small, irregular, carbonate-altered mafic bodies often host very high grade gold as sulphide dissemination, replacement, and breccia fill. Structural breccias in sedimentary rocks are also favourable sites for high-grade gold. Gold distribution in the deposit closely mimics the intrusive rocks, which contain about 80% of the resource. Structural zones in competent sedimentary units account for the remaining 20%. The more steeply dipping sills in the ACMA sill sequence host the highest-grade and most continuous igneous-hosted mineralized zones, particularly where intersected by northeast-striking “feeder” dikes and faults. Gold grade is directly proportional to vein density and intensity of overlapping disseminated sulphide vein aureoles. The dike-dominant Lewis deposit areas consist of sheeted veins with limited disseminated sulphide in the wall rocks and are characterized by lower-grade and less continuous mineralized zones.

Gold distribution in the planned pit area is shown in Figure 7-7, as a bench plan.

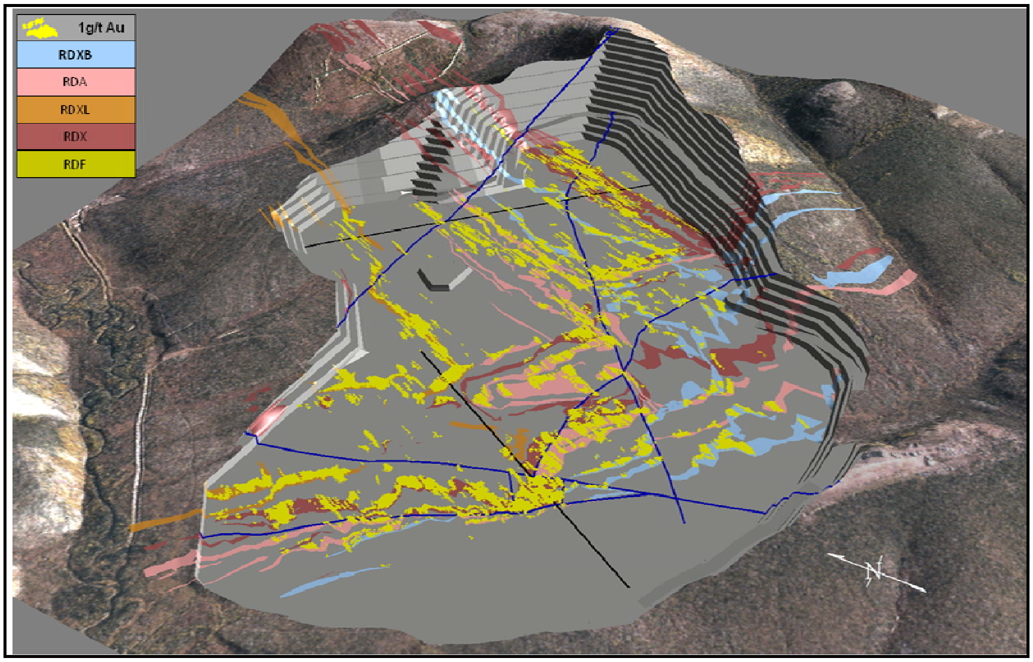
7.7 Mineralization

Gold-bearing zones are coincident with quartz–carbonate–sulphide veins and related disseminated sulphide aureoles in hydrothermally altered rhyodacite bodies and, to a lesser extent, in sedimentary rock near igneous contacts. Continuity and grade of mineralized material within the rhyodacite host rocks varies directly with vein spacing and the amount of vein and disseminated arsenopyrite, the principal gold-bearing mineral. Gold in sedimentary rocks and minor mafic igneous bodies is generally limited to small and discontinuous vein and breccia fill occurrences.

7.7.1 Vein and Disseminated Mineralization

Veins in the ACMA–Lewis area are subtle in appearance and vary from <1 mm to 20 cm wide, averaging <1 cm. They formed in brittle fractures and are typical of open-spaced fillings with vugs, drusy quartz-lined cavities, vein wall-banded and cockscomb quartz, and bladed carbonate. Veins are composed of gray to clear quartz, white to tan carbonate, and as much as 3% sulphides. Table 7-3 contains a summary of the gold-bearing vein stages.

Figure 7-7: 100 m Bench Level Gold Distribution (>1 g/t Au grade blocks)



Note: Figure courtesy Donlin Gold

Table 7-3: Vein Stages

Vein	Description
V1	Thin, irregular, and discontinuous sulphide (>50%) veins with pyrite and trace arsenopyrite, little or no quartz (<30%) or carbonate (<50%). Broad disseminated selvage of pyrite and poorly crystalline illite and Fe–carbonate alteration. Barren or very low grade
V2	Thin, discontinuous quartz (>30%) sulphide veins contain variable pyrite and arsenopyrite. May have broad, often pervasive selvages of fine-grained, needle-like arsenopyrite. Broad pyrite aureole may surround the arsenopyrite selvage. Open-space vuggy textures common. Trace amounts stibnite. Have moderate gold grade and strong illite alteration aureoles with variable Fe–carbonate replacement of the host rock.
V3a	Higher-grade veins. Thicker, more planar and continuous, open-space quartz veins with Fe-dolomite, pyrite, arsenopyrite, native arsenic, and variable amounts of stibnite. Commonly show broad arsenopyrite-rich selvages with little to no Fe–carbonate as wall rock alteration
V3b	Thicker, more continuous, and planar quartz veins with open-space textures and complex mineralogy, including pyrite, arsenopyrite, stibnite, native arsenic, realgar, and trace other sulphides in intensely illite altered material. Gold grades are commonly much higher than the average grade of the deposit
V4	Latest vein phase. Barren carbonate-quartz (>50% and <50%, respectively) vein sets that post-date mineralized veins. Primarily barren white and clear quartz veinlets and calcite ± ankerite veinlets with no sulphides.

Mineralized zones are consistently oriented sub-parallel to the main δ1 axis (024) of the compressive structural regime (Piekenbrock and Petsel, 2003). Veins in the ACMA–Lewis resource evolved through a continuum (V1 through V3) of changing mineralogy and increasing gold grade while maintaining a generally consistent NNE strike and SE dip. The final carbonate–quartz vein set (V4) has a broader range of orientation.

MacNeil (2009) found that the average vein orientation for all veins is 024/71. This orientation is generally consistent across all domains and vein types, which indicates that veins at Donlin formed during the same mineralizing event.

A comparison by host rock shows that veins in igneous rocks strike more easterly and dip more steeply than veins in sedimentary rocks, probably due to refraction across lithologic contacts.

Several quartz and carbonate phases have been recognised, including pre-gold-stage Mn–calcite veins and wall rock replacement and cockscomb quartz veins; Fe–dolomite in main gold stage veins; and post-gold-stage clear quartz veins and ankerite stringer veins.

Euhedral and porous replacement pyrite are the earliest sulphide phases, followed in order by marcasite, arsenopyrite, realgar, and native arsenic. Stibnite is most abundant in later veins. Most accessory sulphides are relatively early, while boulangerite is relatively late. Arsenopyrite occurs as both coarse (up to 1 cm) crystals and very fine (0.1 to 0.2 mm) euhedral grains. Fine-grained arsenopyrite contains five to 10 times more gold than the paragentically earlier coarse-grained phase.

7.8 Alteration

Rhyodacite bodies are ubiquitously altered to an illite–carbonate–kaolinite–chlorite / smectite ± quartz ± graphite assemblage.

Mafic igneous rocks are strongly altered by carbonate ± fuchsite and contain locally high-grade gold with disseminated, massive replacement or breccia filling sulphide.

Altered sedimentary rocks consist of relict quartz grains in a matrix of illite, kaolinite, carbonate, hematite, and <1% pyrite and trace sphalerite (Drexler, 2010).

Pyrite is widespread in all altered rocks (0.5% to 2%) but is more abundant (1% to 4%) in mineralized zones. Alteration is most intense near veins and is typically zoned outward from illite ± kaolinite to kaolinite ± illite and then to a distal zone of chlorite ± smectite ± quartz.

Silica is dominantly restricted to veins in the ACMA–Lewis area and is not generally expressed as pervasive silicification. Vein relationships show an increase in quartz content from early sulphide-dominant veins to late silica-dominant veins. Some increased silicification has been noted in the Queen area (Ebert, 2003b).

Short-wave infrared reflectance (SWIR) spectroscopy data, collected between 2007 and 2011, are interpreted by Donlin Gold show that higher gold is most strongly correlated with an alteration suite dominated by NH_4–illite (ammonia–illite), whereas kaolinite-bearing zones contain lower-grade gold.

7.9 Minor Elements

The most abundant minor elements associated with gold-bearing material are iron (Fe), arsenic (As), antimony (Sb), and sulphur (S). These are contained primarily in the mineral suite associated with hydrothermal deposition of gold, including pyrite (FeS_2), arsenopyrite (FeAsS), realgar (AsS), native arsenic (As), and stibnite (Sb_2S_3). Minor hydrothermal pyrrhotite ($Fe_{1-x}S$) and marcasite (FeS_2), and syngenetic or sedimentary pyrite, also account for some of the Fe and S.

Much less abundant elements such as copper (Cu), lead (Pb), and zinc (Zn) are contained in relatively rare or accessory hydrothermal mineral species observed in the deposit, including chalcopyrite ($CuFeS_2$), chalcocite (Cu_2S), covellite (CuS), tennantite ($Cu_{12}As_4S_{13}$), tetrahedrite ($Cu_{12}Sb_4S_{13}$), bornite (Cu_5FeS_4), native copper (Cu), galena (PbS), sphalerite (ZnS), and boulangerite ($Pb_5Sb_4S_{11}$). Small amounts of silver (Ag) 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 (Mo) occurs in rare molybdenite (MoS_2). Very minor nickel (Ni) has been observed in the secondary sulphide mineral millerite (NiS) and minor cobalt (Co) in various secondary minerals in sedimentary rocks. The Ni and Co probably have a sedimentary origin.

Three elements of particular processing significance are mercury (Hg), chlorine (Cl), and fluorine (F). Graphitic carbon and carbonate minerals also have the potential to negatively affect the metallurgical process.

7.10 Comments on Section 7

In the opinion of the QPs:

- Knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation.

8.0 DEPOSIT TYPES

According to Donlin Gold, the Donlin gold deposits share characteristics of several gold deposit genetic models. It has been classified as:

- Granite porphyry-hosted gold polymetallic (Bundtzen and Miller, 1997)
- Distal or high-level epizonal intrusion-related (Hart et al., 2002)
- Low-sulphidation epithermal (Ebert et al., 2003a)
- Orogenic- or intrusion-related (Goldfarb, 2004),
- Reduced porphyry to sub-epithermal Au–As–Sb–Hg (Ebert et al., 2003c; Hart, 2007).

Hart (2007) classifies the deposit as a high-level, reduced intrusion-related vein system to account for the reduced ilmenite series intrusions, near contemporaneous age of mineralization, and the apparent genetic relationship to the higher-temperature hydrothermal system at Dome (Drexler, 2010).

The Lewis–ACMA part of the district is clearly a low sulphidation, reduced intrusion related, epizonal system with both vein and disseminated mineral zones and conforms most closely to the Hart (2007) classification.

8.1 Comments on Section 8

In the opinion of the QPs, deposit models as used in the exploration programs have been appropriate to the style and setting of the mineralization.

9.0 EXPLORATION

A summary of the exploration programs completed on the Project is summarized in Table 9-1.

9.1 Grids and Surveys

The Project uses Universal Transverse Mercator (UTM) Zone 4 (meters). The map datum is NAD83.

The topographic surface is based on a 2004 survey by Aero-metric. The survey has an accuracy of ±6.6 ft (±2 m) within all key Project areas.

9.2 Geological Mapping

Geological mapping was performed in 1988-1989 (Westgold), and during 1996, 1998, reconnaissance mapping was undertaken by Placer Dome. This was followed in 1999 by Placer Dome completing a 1:10,000 geological mapping program over the entire Project area.

Mapping is generally limited by the poor quality and limited extent of outcrop. Information from the mapping programs was used to support more detailed data obtained from trenches and core drilling.

9.3 Geochemical Sampling

During the period 1988 to 1989, Westgold also collected over 15,000 soil, rock chip and auger samples. Westgold, in 1989, tested biogeochemical sampling, which returned positive results. Teck collected two lines of soil samples in 1993.

Placer Dome, during 2007, collected 600 soil samples in the ACMA and 400 areas, and an additional 646 soil samples and 92 rock samples were collected in 2008. During 2008, Barrick took 1,097 soil, 101 stream sediment, and 66 stream concentrate geochemical samples.

Sampling was used as part of regional prospectivity evaluations.

9.4 Geophysics

Westgold performed an airborne magnetic and VLF survey and ground magnetic surveys during 1998 to 1999. The company also trialled ground-penetrating radar.

Table 9-1: Work History Summary for Donlin Gold Project

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 was the best ever: 800 oz Au recovered.
1974, 1975	RAA	Regional mineral potential evaluation for Calista Corporation. Soil grid and three bulldozer trenches in the Snow Gulch area.	Anomalous gold values in soil, rock, and vein samples. Trench rock sample results range from 2 ppm Au to 20 ppm 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 ppm 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 ppm Au.
1988, 1989	WestGold	Airborne geophysics, ground geophysics, geological mapping, and soil sampling over most of Project area. Total of 44,362 ft (13,525 m) of D9 bulldozer trenching at all prospects. Over 15,000 soil, rock chip, and auger samples collected. Drilling included 3,106 ft (947 m) of AX core drilling, 1,325 ft (404 m) in 239 auger holes, and 34,187 ft (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. Calculated gold resource of 3 M tons at average grade of 2.50 ppm (218,908 oz) at 1 ppm cut-off. WestGold dissolved by early 1990.
1993	Teck	D-9 bulldozer trenching (4,592 ft, 1,400 m) and two 1,640 ft (500 m) soil lines in Lewis area. Petrographic, fluid inclusion, and metallurgical work.	Identified new mineralized areas and expanded property resource estimate to 3.9 Mt at average grade of 3.15 g Au/t (393,000 oz Au). Metallurgical tests not favourable, Project dropped.
1995 to 2000	Placer Dome	286,616 ft (87,383 m) of core, 39,062 ft (11,909 m) of RC drilling, and 27,857 ft (8,493 m) of trenching.	Drilled the American Creek magnetic anomaly (ACMA), discovered the ACMA deposit. Numerous mineral resource calculations.
2001, 2002	DCJV (Placer Dome/NovaGold)	152,543 ft (46,495 m) of core, 38,022 ft (11,589 m) of RC drilling, 294 ft (89.5 m) of geotechnical drilling, and 879 ft (268 m) of water monitoring holes. Mineral Resource estimate.	Expanded the ACMA resource.
2003 to 2005	DCJV	83,491 ft (25,448 m) of core and 19,611 ft (5,979 m) of RC drilling. Calcium carbonate exploration drilling; IP lines for facility condemnation studies.	Infill drilled throughout the resource area demonstrated continuity. Discovered a calcium carbonate resource. Poor quality IP data not useful for facility studies.
2006	DCJV (Barrick/NovaGold)	304,475 ft (92,804 m) of core drilling for resource conversion, slope stability, metallurgy, waste rock, carbonate exploration, facilities, and port road studies.	Geological model and internal resource updates.
2007	DCJV	Core drilling totalled 246,906 ft (75,257 m) and included resource delineation, geotechnical and engineering, and carbonate exploration. 13 RC holes for monitor wells and pit pump tests totalled 3,423 ft	Improved pit slope parameters, positive hydrogeological results, Carbonate exploration was negative.

Year	Company	Work Performed	Results
		(1,043 m). Updated Mineral Resource estimate.	
2008	DCLLC (Barrick/NovaGold)	108 core holes totalling 109,663 ft (33,425 m) for exploration and facility related geotechnical and condemnation studies. Metallurgical test work: flotation variability and CN leach. 54 test pits and 37 auger holes 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 successfully condemned. Updated resource estimates utilizing applicable data through 2007
2009	DCLLC	19 geotechnical core holes totalling 3,116 ft (950 m) in facility sites and to address hydrology. Mineral Reserve and Mineral Resource estimate update.	
2010	DCLLC	Six geotechnical core holes totalling 6,855 ft (2,090 m) to evaluate slope stability of expanded pit. Also drilled 90 auger holes totalling 1,919 ft (585 m) and dug 59 test pits to further evaluate overburden conditions and gravel supplies within TSF area. Mineral Reserve and Mineral Resource estimate update.	Pit slope stability of new pit design remained acceptable. Evaluation of construction suitability of surficial materials in TSF is ongoing.

During 1999, Placer Dome completed 25 line km of max-min (electromagnetic) geophysical survey completed in the ACMA, 400 and southern Lewis areas, and 1,800 line km of aeromagnetic survey was completed at 50 m line spacing and 50 m elevation over the property. In the same year, a total of 17.7 km of IP and resistivity lines were completed, and an additional 41.6 km of IP/resistivity lines were run in 2000. During 2003, IP surveys were undertaken in areas where infrastructure was planned, as part of condemnation evaluations.

Geophysical studies were used to support structural interpretations for geological modelling purposes, exploration targeting, and facilities condemnation drilling

9.5 Pits and Trenches

During 1988 and 1989, a total of 44,362 ft (13,525 m) of D9 bulldozer trenching was completed by Westgold at all prospects.

Teck completed 4,592 ft, (1,400 m) of D-9 bulldozer trenching during 2003.

The 1996 Placer Dome exploration program included than 8,200 ft (2,500 m) of trenches for sampling and mapping purposes in southeast Lewis area; this was followed in 1997 by 13,852 ft (4,222 m) of trenches in the Lewis area, in 1998 by 6,739 ft (2,054 m) of trenching and geological mapping in the Lewis–Vortex and ACMA areas, and in 1999 by 7,339 ft (2,237 m) of trenching and geologic mapping (Dome, Queen, Far Side, and Vortex).

Pits to provide geotechnical data were excavated in 2005 (22 test pits), 2006 (40 test pits), 2007 (55 test pits), and 2008 (54 test pits).

9.6 Petrology, Mineralogy, and Research Studies

During 1993, Teck completed petrographic and fluid inclusion studies in support of understanding of the mineralization setting and host rock lithologies.

9.7 Geotechnical and Hydrological Studies

Geotechnical and hydrological studies have been undertaken in support of mine planning, mine design and environmental considerations.

9.8 Metallurgical Studies

Metallurgical testwork is primarily based on drill core.

9.9 Exploration Potential

Exploration potential in the vicinity of the open pit design in FSU2 includes extensions along strike to the East ACMA, Lewis, and Crooked Creek dike areas. Mineralization remains open at depth uner the current pit limits. Mineralization also remains open to the north of the planned pit and has been tested by shallow trenching and soil sampling, with limited drilling undertaken to date.

The Project also retains exploration potential outside the areas that have been the subject of the mine design in FSU2. Gold mineralization is associated with an overall north–northeasterly-trending high level dike/sill complex that has been outlined in the regional aero-magnetics as a subtle 50 nT magnetic low (Figure 9-1). The zone, approximately 8 km long, and 4 km wide, consists of a northern, dike-dominated area, and a southern, more sill-dominated area (refer to Figure 9-1 and Figure 4-1).

Figure 9-2 shows a gold-in-soils compilation plan of the area indicated by the magnetic low. The ACMA/Lewis area is the southern portion of this plan. No drilling has been performed in the northern portion since initial exploration activities, and some isolated drilling in the 1990s. Exploration targets identified by NovaGold for additional work includes Far Side, Duqum, Snow, Quartz, Queen, Dome, and Ophir (refer to Figure 4-1). The following summaries of the exploration potential identified by NovaGold are sourced from Buchanan (2009), Chamois (2009) and Francis (2011).

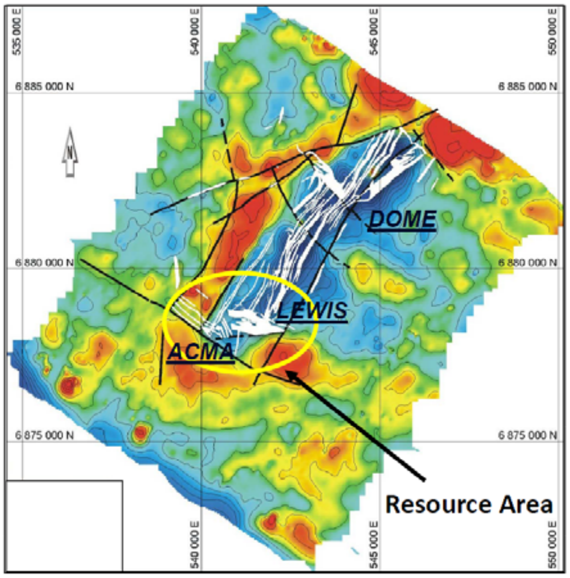
9.9.1 Far Side

The Far Side prospect has been tested by three NovaGold core drill holes (735 m) along 300 m of strike and 29 RC holes that were drilled by West Gold. Drill results for the core drilling are indicated in Table 9-2. The prospect is situated in an area where dikes of a generally easterly trend intersect the more dominant northeasterly trend.

9.9.2 Duqum

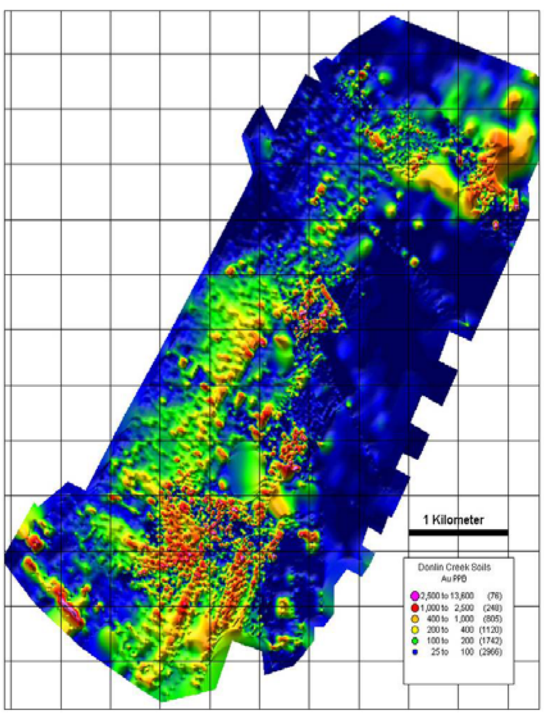
This prospect is the site of the first recorded lode gold discovery in the Donlin area. It is about 1 km long, and mineralization is associated with a narrow porphyry dike. Three core holes have been drilled (1,043 m) by NovaGold. Drill results are summarized in Table 9-3.

Figure 9-1: Regional Magnetic Image Showing Magnetic Low Intensity Zone



Note: Figure courtesy Donlin Gold

Figure 9-2: Gold-in-Soils Compilation Plan



Note: Figure courtesy Donlin Gold

Table 9-2: Far Side

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Drill Intercept From (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC96-254	6,883,395	542,905	152	320	61	23.2	16.8	4.60
DC96-255	6,883,398	542,918	68	313	65	122.0	14.0	3.00
DC96-256	6,883,347	542,843	55	317	61	130.0	15.6	5.86

Table 9-3: Duqum

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Drill Intercept From (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC97-387	6,882,477	543,488	-3	281	51	338.0	16.0	2.39
DC97-388	6,882,690	542,977	47	314	55	210.0	12.0	5.03
DC97-388	6,882,700	542,967	26	314	55	236.0	10.0	2.29
DC97-389	6,882,649	543,109	143	39	56	90.0	10.0	3.86
DC97-389	6,882,696	543,148	49	40	57	202.0	10.0	2.79
DC97-389	6,882,746	543,188	-53	36	59	320.0	16.0	3.79

9.9.3 Snow/Quartz

The Snow and Quartz prospects are hosted in a dike-related, gold-bearing corridor that is about 1.5 km wide and approximately 4 km long. In the area of this dike swarm, the porphyry dikes are 20 m to >100 m wide, discontinuous bodies. Gold mineralization is associated closely with the dikes, and is hosted either within the dikes themselves or in the adjacent sedimentary rocks. Limited drilling has been completed. Better drill results are included in Table 9-4.

9.9.4 Dome

The Dome prospect is situated under a prominent, rounded hill about 5 km north of the planned ACMA/Lewis pits. Several mineralized felsic porphyries intrude into a greywacke unit and have hornfelsed the sediment over wide intervals. Mineralization consists of stockworks of veinlets containing arsenopyrite, pyrite, pyrrhotite, and minor chalcopyrite. Preliminary metallurgical testwork undertaken by NovaGold indicates that mineralization may be less refractory than that encountered in the ACMA/Lewis area.

Fourteen widely-spaced drill holes have been completed over an area of approximately 500 m x 500 m. Mineralization is open to the north, east, south and to depth, and may be open to the west at depth. Better drill results are included in Table 9-5.

Table 9-4: Snow/Quartz

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Drill Intercept From (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC97-383	6,880,381	541,433	210	294	50	16.0	23.0	2.77
DC97-384	6,880,551	541,531	150	294	52	52.0	10.0	3.34

Table 9-5: Dome

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Drill Intercept From (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC08-1785	6,882,528	544,103	207	109	70	139.0	10.0	2.19
DC08-1785	6,882,525	544,111	185	111	70	163.0	10.0	3.89
DC08-1785	6,882,518	544,129	134	113	68	211.0	22.6	3.29
DC08-1785	6,882,512	544,142	98	114	67	248.0	25.0	2.94
DC97-392	6,882,456	544,082	232	128	65	94.0	52.0	3.21
DC97-392	6,882,431	544,113	145	128	65	185.0	61.0	3.30
DC97-392	6,882,418	544,129	100	132	66	258.0	14.0	3.99

9.9.5 Ophir

The Ophir Hill is the highest topographic feature in the Donlin district. Surface mapping over an area of about 1.5 km by 750 m indicates Cretaceous sediments have been intruded by felsic to intermediate intrusions, which may be dikes. Surface exposures are completely oxidized, but boxworks after sulphides indicate arsenopyrite, pyrite and other sulfides occur as disseminations and thin veinlets. Soil sampling has identified a strong gold-in-soil anomaly on the southwestern flanks of the hill. No drilling has been undertaken.

9.10 Comments on Section 9

In the opinion of the QPs:

- The exploration programs completed to date are appropriate to the style of the deposits and prospects within the Project
- Additional exploration potential remains in the Project area.

10.0 DRILLING

Approximately 1,678 exploration and development core (89%) and RC (11%) drill holes, totalling 1,206,960 ft (367,886 m), were completed from 1988 through 2007 in at least six separately managed campaigns. Approximately 50% of the core and 40% of the holes were drilled during 2006–2007. Another 108 core holes totalling 109,634 ft (33,425 m) were added in 2008 to explore near-pit expansions and satellite deposits, and for facility-related condemnation and geotechnical studies.

A total of 1,396 core (89%) and RC (11%) holes totalling 1,114,324 ft (339,733 m), as well as 282 trenches totalling 70,344 ft (21,441 m), were used for the FSU2 resource model.

Drill summaries for the RC and core drill programs are included in Table 10-1. Drill location plans are provided in Figure 10-1 for the Project, and in Figure 10-2 for the area where Mineral Resources and Mineral Reserves were estimated.

10.1 Drill Methods

West Gold drilling employed a Winkie rig. Drill data from this program are not contained in the Donlin database.

Boart Longyear was the coring contractor from 1995 through 2010. Core drilling was accomplished exclusively with LF-70 model drills, which were set up in heli-portable configuration, mounted on skids or on self-propelled tracked and low-ground-pressure Nodwell carriers. Standard wire line core retrieval with 5 ft or 10 ft (1.52 or 3.05 m) core barrels was used in all core drilling operations.

Core sizes used on the Project include: NQ3 (45.1 mm core diameter), NQ (47.6 mm), HQ3 (61.2 mm), HQ (63.5 mm), and PQ (85 mm). 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.

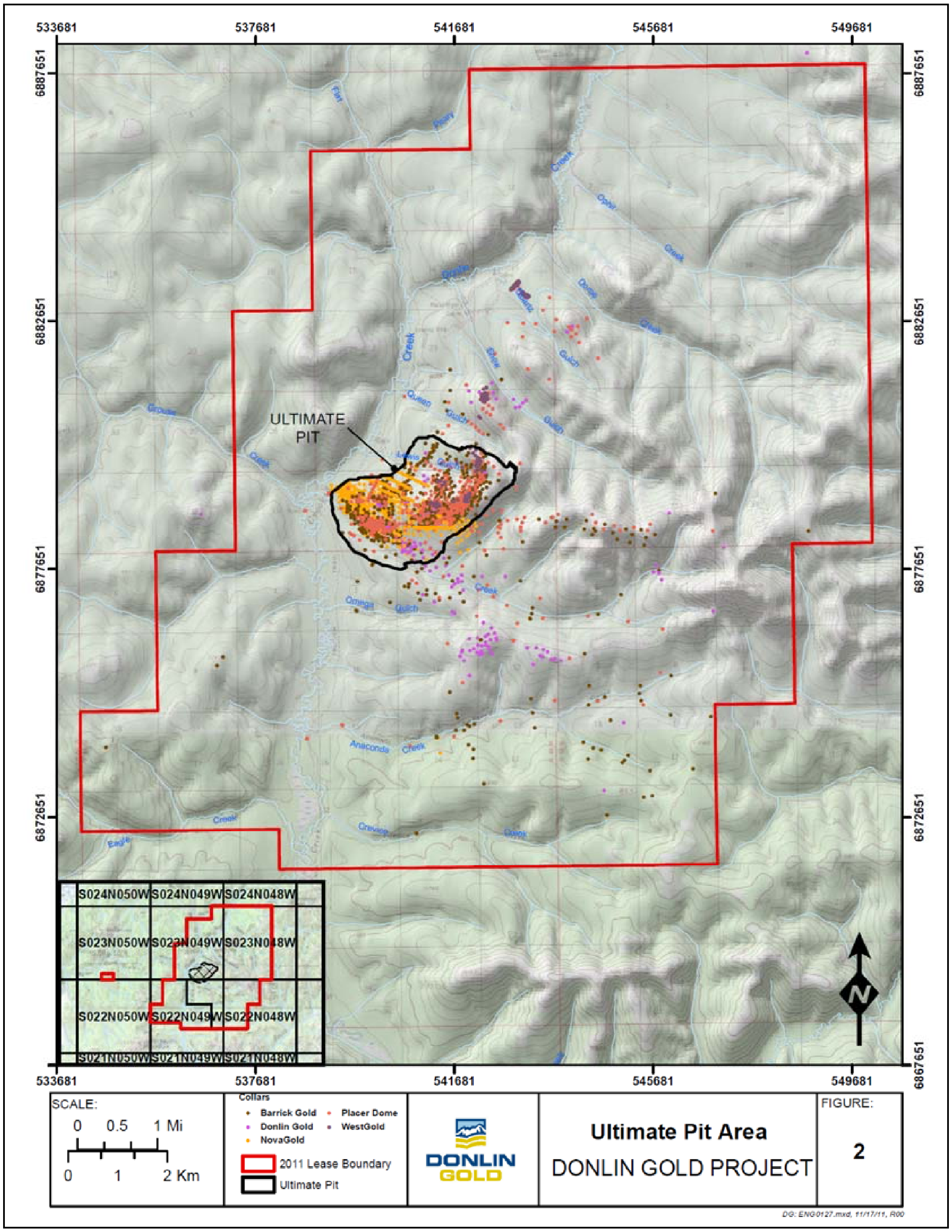
Depth limits for HQ size holes were 1,560 ft (475 m) for dry conditions and 1,785 ft (545 m) for fluid-filled holes. HQ depth was generally limited to 1,400 ft (426 m) for holes with no planned reduction.

Table 10-1: RC and Core Drill Summary Table

Year	Company	Number of Drill Holes	Hole Type	Drill Footage	Comment
1988	WestGold	33	Core	unknown	shallow (average 82 ft, or 25 m), AX-diameter
		50	Auger	unknown	shallow (average 26 ft, or 8 m)
1989	WestGold	125	RC	unknown	31 holes in Far Side, 38 in Snow, 24 in Queen, 8 in Rochelieu, and 24 in Lewis
1995	Placer Dome	32	core		30 in Lewis, 1 at Rochelieu Ridge, and 1 near the mouth of Queen Gulch
1996	Placer Dome	28	RC		Seventeen of the holes twinned earlier core holes. Four water wells (3 in camp, 1 in Lewis) were drilled with the RC drill, and 5 core holes in the 400 area were pre-collared through deep overburden.
		116	core		All but 8 of the core holes were drilled in Lewis or Queen. The others were distributed north of the current resource area in the Dome, Far Side, and Snow prospects
1997	Placer Dome	52	RC		Lewis, Queen, Rochelieu, ACMA, 400 Area, Vortex, alongside the American Ridge runway, and Snow. Includes two water wells.
		66	core		Lewis, Queen, 400 Area, ACMA, and north of the resource area at Quartz, Duqum, and Dome
1998	Placer Dome	96	core		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 in June to October to test variography. Resource expansion drilling in the Lewis, Queen, and ACMA areas was also conducted from June to October.
1999	Placer Dome	33	core		Twenty-six of these, totalling 21,949 ft (6,690 m), were resource definition holes drilled in ACMA–400
2000	Placer Dome	7	core		5 at Dome and 2 at Quartz, for an evaluation of IP anomalies and potential for high-grade deposits
2001	Placer Dome	42	core		Evaluation of the potential for significant resource growth in the ACMA area
2002	NovaGold	146	RC	38,022 ft (11,589 m)	141 exploration and resource expansion holes in the ACMA, 400, Lewis, Akivik, Rochelieu, Vortex, and Far East prospects. 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
		196	core	128,255 ft (39,092 m)	Two of the core holes are geotechnical holes in the Anaconda Creek valley.
2003	Placer Dome	16	RC		Water monitoring wells
2004	Placer Dome	17	RC	7,661 ft (2,335 m)	Condemnation holes in the Anaconda Creek and upper American Creek valleys
		3	core		Geotechnical core holes
2005	Placer Dome	30	RC	11,955 ft (3,644 m)	
		90	core	80,696 ft (24,596 m)	Infill in ACMA and Lewis

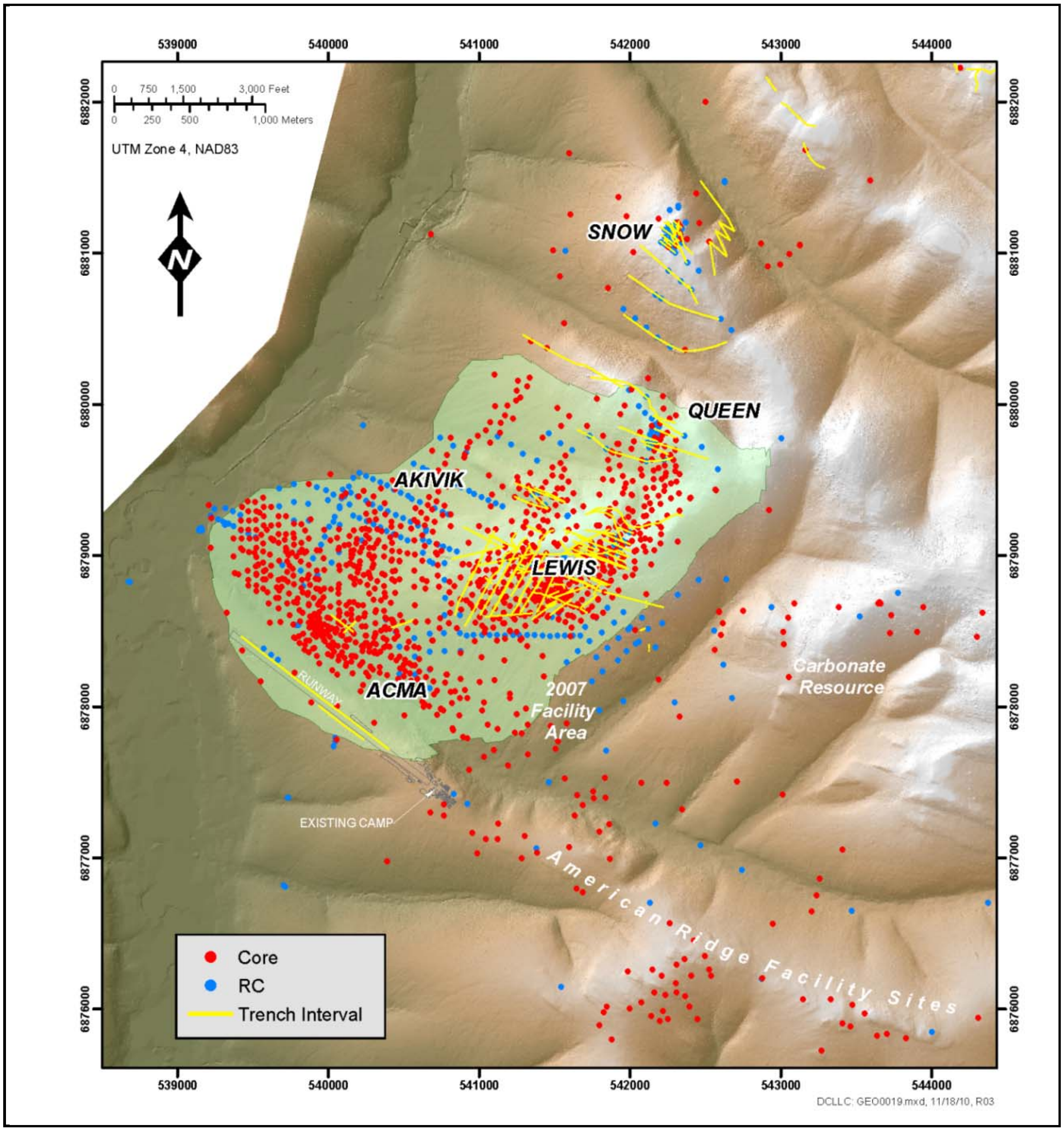
Year	Company	Number of Drill Holes	Hole Type	Drill Footage	Comment
2006	DCJV	327	core	304,475 ft (92,804 m)	Pit slope stability, metallurgy, waste rock studies, facilities condemnation, and engineering, and calcium carbonate resource bulk sampling, delineation, and exploration
2007	DCJV	13	RC	3,423 ft (1,043 m)	Monitor wells and pit pump tests
		248	core	246,906 ft (75,257 m)	Pit resource infill, pit expansion, carbonate exploration, geotechnical, and engineering studies
2008	DCLLC	108	core	109,663 ft (33,425 m)	Exploration, resource infill, condemnation, and geotechnical studies
2009	DCLLC	19	core	3,116 ft (950 m)	Geotechnical and hydrological core holes
2010	DCLLC	6	core	6,855 ft (2,090 m)	Geotechnical core holes

Figure 10-1: Project Drill Hole Location Plan



Note: Figure courtesy Donlin Gold.

Figure 10-2: Resource Area Drill Holes



Note: Figure courtesy Donlin Gold.

Drill holes planned to depths greater than 1,560 ft (426 m) were reduced depending on bit changes or logistical scheduling at a depth range of 600 to 800 ft (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.

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 mounted on low-ground-pressure, self-propelled tracked carriers equipped with high-volume air compressors, standard 4" (102 mm) dual walled pipe in 20 ft lengths, and down-hole pneumatic hammers with 5¼" (133 mm) carbide button bits were used for RC drilling. 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.

RC drilling was used by WestGold in 1988-1989 for its initial exploration, by 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 have been used exclusively for all resource delineation, and RC drilling was relegated to condemnation and hydrology studies.

10.2 Geological Logging

Standard logging conventions were developed by Placer Dome, and refined over the durations of the drilling 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 with the resulting data entered into the main database (Access© database) either by the logging geologist or a technician. Five types of data were captured in separate tables: Lithology, Mineralization, Alteration (visual), Structural and Geotechnical. Remarks were also captured. Lithology was recorded in 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 collected consisted of the type of structure, measurements relative to core axis and oriented core measurements, if applicable. The Geotechnical table recorded percent recovery and RQD for the entire hole, and fracture intensity where warranted.

RC drilling chips were logged on paper forms and the data entered into an electronic database.

10.3 Recovery

A survey of nearly 200,000 core recovery records in the database revealed an overall length-weighted average core recovery of 95%. Average recovery increases from 80 to 95% from 0 to 40 m and then ranges from 95 to 100% below 40 m where overburden and surface weathering effects are generally absent.

10.4 Collar Surveys

From 1988 through 1993, conventional theodolite survey methods were used to tie drill hole collar and trench locations to a surveyed ground control net. 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 to 1999 and 2001 drill collars and trenches. An Ashtech Promark2 GPS post-processed system consisting of a base unit and up to two roving units was introduced in 2002. The roving Promark2 instruments were operated in the field to collect stationary readings over the drill collars.

Typical reading times for periods of good satellite reception were about 20 minutes and varied from 10 minutes to four hours or more, depending on the number of satellites, roving-camp base unit distance, canopy coverage, and proximity to north- and east-facing slopes. 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 were checked for accuracy and quality control, and then copied to an Excel survey data file. This in turn was copied to the acQuire database for archival. Based on Ashtech surveys of control points, the approximate maximum horizontal and vertical variances of drill hole collar surveys under optimal conditions were considered by Donlin Gold to be 0.2 and 0.6 m, respectively.

10.5 Down-hole Surveys

The Sperry Sun single-shot camera method was used through 2000 for directional surveys to determine down-hole deviation. 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 150 ft (50 m) intervals from 20 ft (6 m) off bottom to within 100 ft (30 m) of the surface. An integrated key pad and LCD display provided for manual operation and data retrieval. Handwritten data were delivered to the geologists with the shift reports for quality control and manual entry into the acQuire database.

Approximately 60% of the core holes drilled within the resource model area were oriented to collect structural information for geotechnical and geological studies. Core orientation methods included clay impression, EZ Mark, and Reflex ACT instrument. 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 the 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.

10.6 Geotechnical and Hydrological Drilling

Geotechnical and hydrological drilling is included in the drill totals in Table 10-1, and are included in the drill location plan in Figure 10-2.

10.7 Metallurgical Drilling

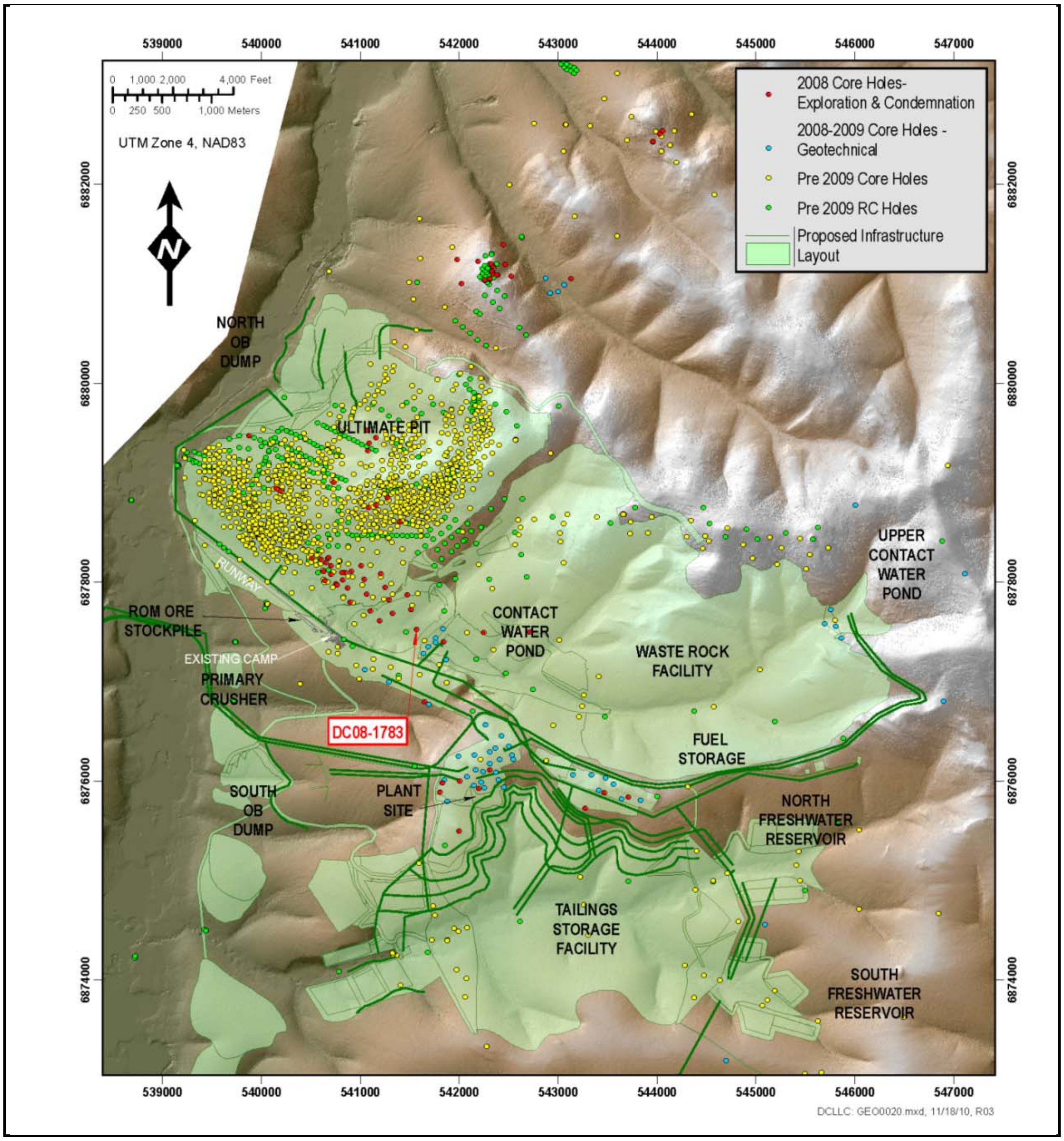
Specific drill holes were completed for metallurgical testwork. These holes, although not broken out by collar, are included in the drill location plan in Figure 10-2.

10.8 Condemnation Drilling

Condemnation drilling was performed to sterilize potential infrastructure sites in 2007. During FSU1, some infrastructure was relocated further to the east, as the host mineralized intrusions were identified to be plunging in this direction. Exploration drilling in early 2008 confirmed that the most prospective intrusive sills plunge to the east and therefore allow only modest potential for eastward pit expansion into the 2007 near-pit facility sites. Condemnation drilling in the facility sites that were relocated in 2008 did not identify near-surface mineralized material or a favourable geologic environment within 1,640 ft (500 m) of the surface.

Exploration drill hole DC08-1783, near the relocated lower contact water dam site, intersected moderately south-dipping, mineralized intrusive rocks in two zones between 1,788 ft (545 m) and 2,352 ft (717 m).

Figure 10-3: Proposed Facility Sites (FSU1 Layout) and Drill Hole Locations



Note: Figure courtesy Donlin Gold.

These intercepts indicate that a there may be potential for mineralization that may be able to be extracted using underground mining methods farther south where sedimentary rock bedding dips and the inferred intrusive sills may have a near vertical orientation. This exploration target depth exceeds 2,460 ft (750 m) beneath the surface of American Ridge and the relocated facility sites.

Locations of the condemnation drill holes are included in Figure 10-2.

10.9 Drill Orientations

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

Placer Dome conducted variography testing in the North and South Lewis areas. Fourteen core holes oriented approximately normal to the dikes and the north–northeasterly-trending mineralized zones (295° azimuth / -50° dip) and located on a grid spacing of approximately 115 ft (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–northeasterly-trending 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 115 ft (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).

Barrick (Jutras, 2006) further investigated the possibility of a gold grade bias in the resource model. Five major drill hole orientations totalling 1,298 drill 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)
- Vertical (dip of -90°, 51 holes).

Jutras (2006) found that northeasterly-oriented (sub-parallel to the north–northeasterly-trending mineral zones) drill holes were higher grade than the average grade of the deposit relative to the other orientations, but that this bias was partly caused by some clustering in a high-grade part of the ACMA deposit. The results of the study showed that the northeast-oriented holes did not present a significant bias for the estimation of the mineral resource and that they should be included in future resource estimates. A standard nothwesterly drill hole orientation (300° azimuth, -60° dip to the northwest) “normal” to the north–northeasterly-trending structural control of the gold mineralization was adopted for all resource delineation programs from 2005 onward.

10.10 Twin Drilling

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 in grade than in the twinned core holes (Szumigala, 1997).

10.11 Drilled Width versus True Thickness

Although the drill holes were designed to intersect the mineralization as perpendicular as possible; reported mineralized intercepts are longer than the true thickness of the mineralization.

10.12 Summary of Drill Intercepts

A summary of a number of drill hole intercepts from each key area is shown in Table 10-2. Examples of the drill hole geometry, and drill hole intercepts are shown in Figures 10-3 and 10-4 (ACMA) and Figures 10-5 and 10-6 (Lewis), and demonstrate that the drilling was designed to intersect the mineralization as perpendicular as possible.

10.13 Comments on Section 10

In the opinion of the QPs, the quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programs completed by Placer Dome, NovaGold, Barrick, the DCJV, and DCLLC are sufficient to support Mineral Resource and Mineral Reserve estimation as follows:

- Core logging meets industry standards for gold exploration
- Collar surveys have been performed using industry-standard instrumentation
- Downhole surveys were performed using industry-standard instrumentation
- Recovery data from core drill programs are acceptable
- Geotechnical logging of drill core meets industry standards for planned open pit operations
- Drill orientations are generally appropriate for the mineralization style, and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit area.

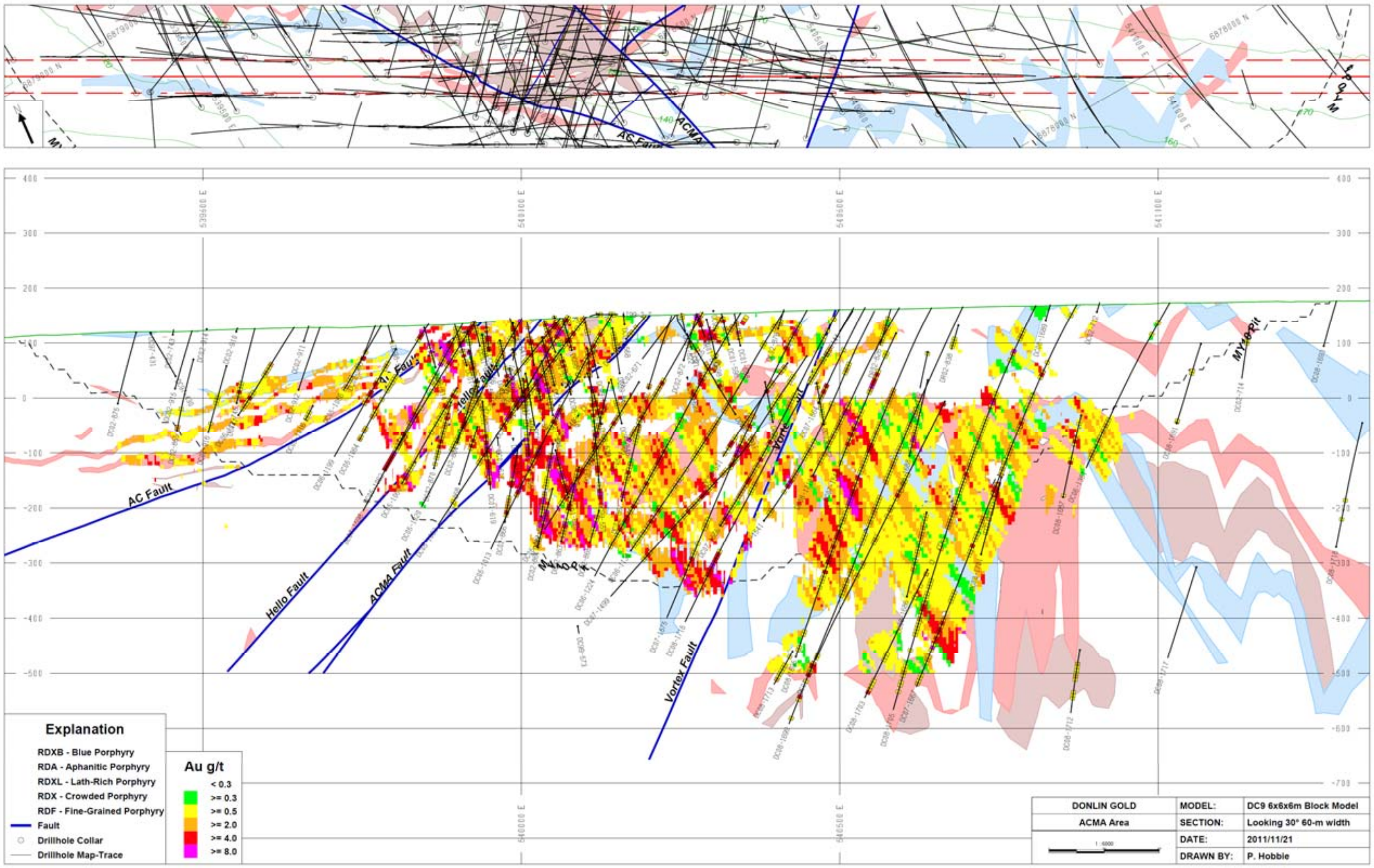
Table 10-2: Drill Hole Intercept Summary Table

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Area	Drill Intercept From (m)	Drill Intercept To (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC06-1114	6878385.36	539899.82	127.97	294.85	-65.4	ACMA	178.00	218.19	40.19	4.14
DC06-1114	6878385.36	539899.82	127.97	294.85	-65.4	ACMA	234.00	304.68	70.68	4.10
DC06-1114	6878385.36	539899.82	127.97	294.85	-65.4	ACMA	310.28	316.51	6.23	3.79
								Total	*117.10*	*4.10*
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	194.00	198.00	4.00	1.37
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	232.00	242.00	10.00	4.58
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	248.00	252.98	4.98	19.37
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	261.00	280.29	19.29	5.64
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	286.00	304.00	18.00	2.19
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	310.00	332.94	22.94	3.49
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	343.05	368.00	24.95	5.87
								Total	*104.16*	*5.01*
DC06-1120	6878411.6	539846.32	127.22	295.85	-61.2	W. ACMA	145.00	160.00	15.00	2.48
DC06-1120	6878411.6	539846.32	127.22	295.85	-61.2	W. ACMA	175.00	205.00	30.00	1.11
DC06-1120	6878411.6	539846.32	127.22	295.85	-61.2	W. ACMA	235.50	271.46	35.96	2.89
								Total	*80.96*	*2.15*
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	189.04	204.50	15.46	2.56
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	222.00	229.30	7.30	4.21
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	259.90	264.00	4.10	1.21
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	303.00	309.00	6.00	2.32
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	317.00	335.00	18.00	5.37
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	345.00	365.00	20.00	3.27
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	374.00	382.00	8.00	2.40
								Total	*78.86*	*3.43*
DC06-1134	6879104.23	539709.27	149.47	297.35	-61.3	Akivik	17.00	27.00	10.00	1.64
DC06-1134	6879104.23	539709.27	149.47	297.35	-61.3	Akivik	35.00	43.00	8.00	4.22
DC06-1134	6879104.23	539709.27	149.47	297.35	-61.3	Akivik	187.00	201.00	14.00	5.54
								Total	*32.00*	*3.99*
DC06-1136	6879210	539771.1	150.99	297.55	-59	Akivik	33.00	47.00	14.00	2.90
DC06-1136	6879210	539771.1	150.99	297.55	-59	Akivik	61.00	69.00	8.00	2.79
								Total	*22.00*	*2.86*
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	13.00	40.00	27.00	2.07
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	54.63	68.00	13.37	2.70
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	104.23	117.00	12.77	1.51
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	285.50	288.00	2.50	12.40
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	306.00	316.00	10.00	4.05
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	405.00	409.00	4.00	4.54
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	460.00	474.00	14.00	3.88
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	494.00	506.00	12.00	2.19

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Area	Drill Intercept From (m)	Drill Intercept To (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	526.00	530.00	4.00	3.25
								Total	*99.64*	*2.96*
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	5.33	23.00	17.67	1.62
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	42.00	62.00	20.00	1.84
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	94.00	106.00	12.00	5.53
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	112.00	126.00	14.00	2.33
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	128.00	148.97	20.97	2.85
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	166.00	178.00	12.00	1.21
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	187.00	193.65	6.65	2.33
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	210.00	245.50	35.50	8.35
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	254.00	276.00	22.00	1.89
								Total	*160.79*	*3.68*
DC06-1252	6878663.18	541745.81	302.68	303.35	-59.6	Lewis	45.21	53.21	8.00	3.98
DC06-1252	6878663.18	541745.81	302.68	303.35	-59.6	Lewis	104.49	108.49	4.00	1.78
DC06-1252	6878663.18	541745.81	302.68	303.35	-59.6	Lewis	209.10	215.10	6.00	3.07
DC06-1252	6878663.18	541745.81	302.68	303.35	-59.6	Lewis	240.90	259.00	18.10	2.56
DC06-1252	6878663.18	541745.81	302.68	303.35	-59.6	Lewis	265.00	281.00	16.00	4.80
DC06-1252	6878663.18	541745.81	302.68	303.35	-59.6	Lewis	297.00	309.00	12.00	2.73
								Total	*64.10*	*3.39*
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	9.00	14.33	5.33	3.00
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	26.00	32.50	6.50	1.84
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	39.69	42.50	2.81	3.39
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	74.50	85.70	11.20	1.17
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	90.00	120.00	30.00	1.60
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	293.00	305.00	12.00	1.26
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	311.00	340.50	29.50	3.29
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	349.00	363.00	14.00	1.21
								Total	*111.34*	*2.05*
DC06-1183	6879518.7	541253.71	240.06	299.65	-61.8	Rochelieu	42.00	51.00	9.00	2.22
DC06-1183	6879518.7	541253.71	240.06	299.65	-61.8	Rochelieu	60.00	63.00	3.00	2.01
DC06-1183	6879518.7	541253.71	240.06	299.65	-61.8	Rochelieu	96.00	102.00	6.00	6.72
DC06-1183	6879518.7	541253.71	240.06	299.65	-61.8	Rochelieu	110.00	116.00	6.00	3.64
								Total	*24.00*	*3.67*
DC06-1185	6879443.2	541416.5	288.87	295.35	-61.9	Rochelieu	29.80	55.93	26.13	3.61
DC06-1185	6879443.2	541416.5	288.87	295.35	-61.9	Rochelieu	63.84	78.00	14.16	1.85
DC06-1185	6879443.2	541416.5	288.87	295.35	-61.9	Rochelieu	196.00	204.77	8.77	6.66
DC06-1185	6879443.2	541416.5	288.87	295.35	-61.9	Rochelieu	237.00	251.00	14.00	6.25
DC06-1185	6879443.2	541416.5	288.87	295.35	-61.9	Rochelieu	283.00	289.00	6.00	1.82
								Total	*69.06*	*4.02*
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	131.67	135.09	3.42	1.54
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	190.00	206.00	16.00	2.41

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Area	Drill Intercept From (m)	Drill Intercept To (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	212.00	216.00	4.00	4.27
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	242.00	248.00	6.00	1.46
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	266.00	272.00	6.00	2.27
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	316.43	332.80	16.37	2.10
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	362.00	377.04	15.04	1.47
								Total	*66.83*	*2.09*
DC06-1268	6879663.53	542219.04	356.56	293.95	-58.2	Queen	159.55	165.00	5.45	2.08
DC06-1268	6879663.53	542219.04	356.56	293.95	-58.2	Queen	234.00	243.00	9.00	8.06
DC06-1268	6879663.53	542219.04	356.56	293.95	-58.2	Queen	254.00	257.00	3.00	2.43
DC06-1268	6879663.53	542219.04	356.56	293.95	-58.2	Queen	288.00	292.34	4.34	2.54
								Total	*21.79*	*4.69*

Figure 10-4: Example Drill Cross-Section ACMA



Note: Figure courtesy Donlin Gold

Figure 10-5: Vertical Cross Section through ACMA and Lewis Block Model, Looking 315°

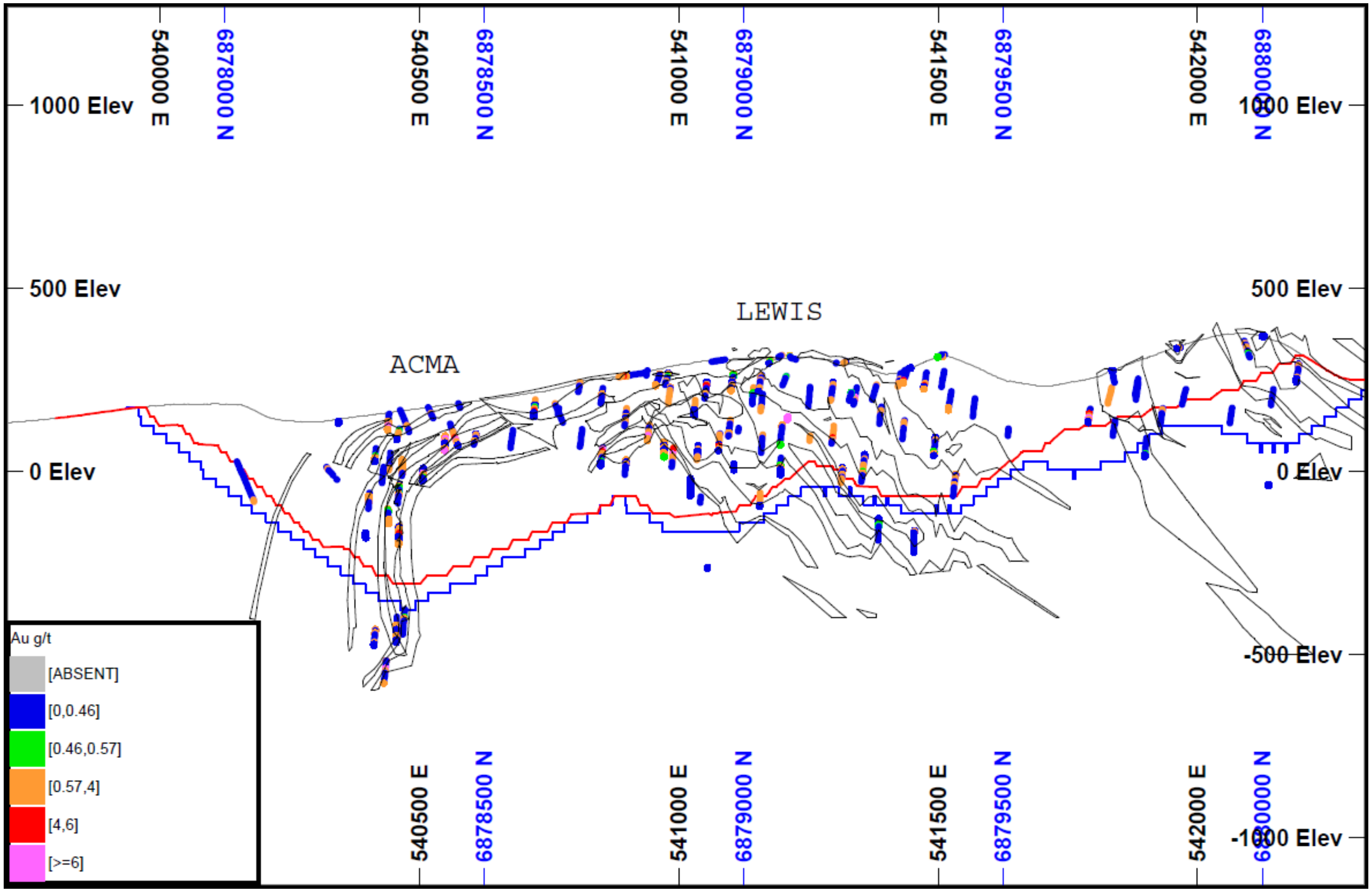
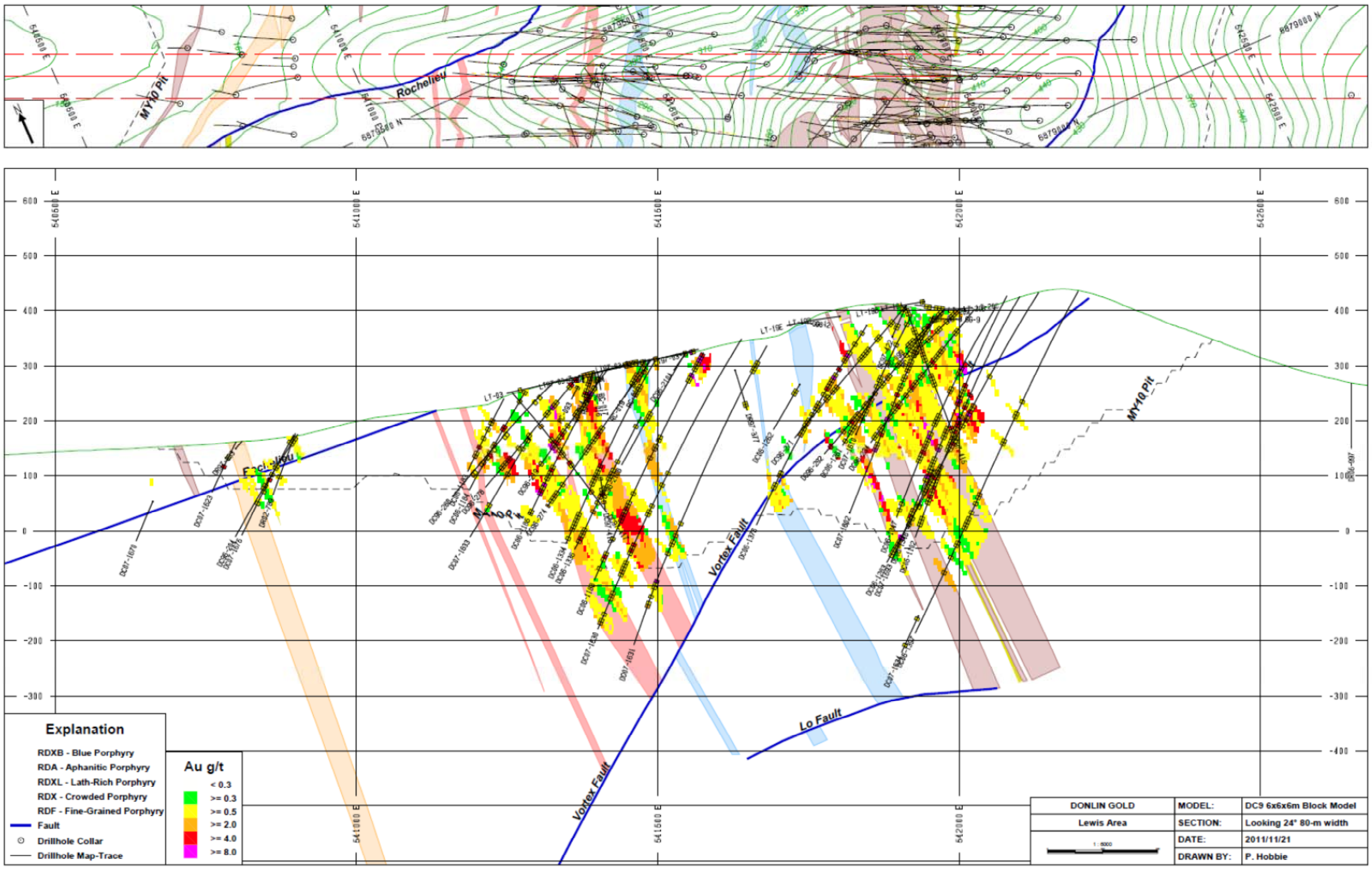
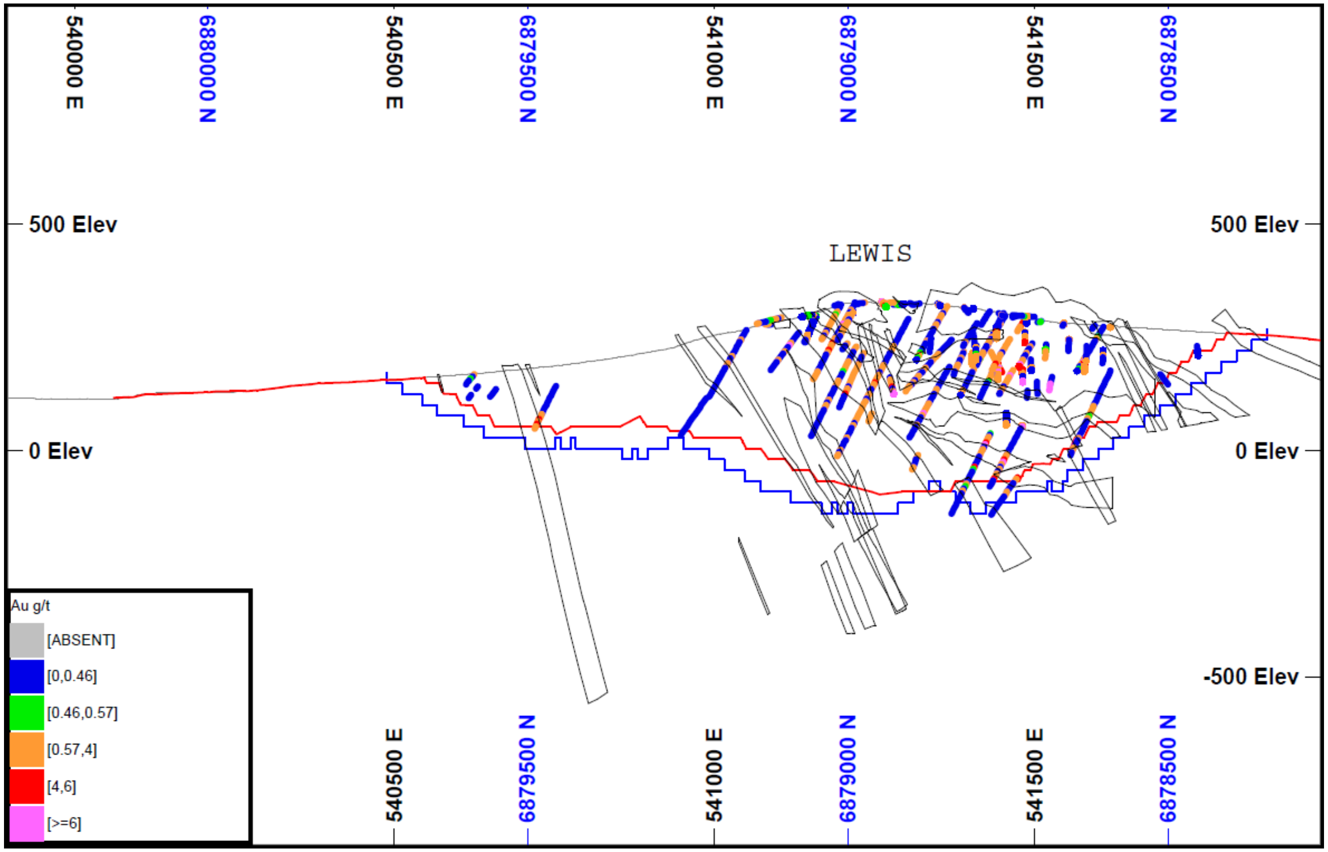


Figure 10-6: Example Drill Cross-Section, Lewis



Note: Figure courtesy Donlin Gold

Figure 10-7: Vertical Cross Section through Lewis Block Model, Looking 45°



- Drill orientations are shown in the example cross-sections included as Figures 10-3 to 10-6, and can be seen to appropriately test the mineralization
- Drill hole intercepts as summarized in Table 10-2 appropriately reflect the nature of the gold mineralization. The table demonstrates that sampling is representative of the gold grades in the deposits, reflecting areas of higher and lower grades.
- In the bottom of the ACMA pit, the preferred orientation of the drill holes and the trend of the mineralization are both northwest. Defining northwest-trending intrusives using northwest-trending drill holes, however, makes establishing the location of the mineralization more subjective than if the mineralization was defined using drill holes perpendicular to the mineralization. Figure 10-4 illustrates that although the contacts of the mineralized intrusions are well defined at higher elevations, the location of the mineralized intrusions are subjective at the bottom of the pit. Figure 10-5 is not demonstrating that the current location of the orebody is incorrect, it is only demonstrating the possibility that the location of the orebody could be different than what is currently in the model. Since non-optimized location of an orebody at the toe of the highwall could have significant economic consequences, AMEC recommends that the location of the orebody at the bottom of the pit be verified by additional drill holes drilled perpendicular to the trend of the mineralization.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling Methods

Drill hole sampling protocols were developed by Placer Dome, and refined over subsequent drill programs.

Holes are sampled from the top of bedrock to the end of the hole. Overburden, excluding the organic layer, is also sampled if core recovery was good and if the interval was 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 6.6 ft (2 m), whereas sample lengths in sedimentary rock zones that lack appreciable sulphide/arsenic minerals can be 9.8 ft (3 m). A minimum of three additional 6.6 ft (2 m) sample intervals are placed before and after each intrusive rock or mineralized zone. An aluminum 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 equipped with 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.

In December 2006 and January 2007, a total of approximately 39,360 ft (12,000 m) of whole core was shipped to an off-site logging and core-splitting facility in Anchorage. 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

11.2 Metallurgical Sampling

Typically, metallurgical sampling consisted of taking half-core samples which were used in flotation and pressure oxidation tests. Whole core samples were only taken for drop weight and SMC comminution tests.

11.3 Density/Specific Gravity Determinations

Historically, only two specific gravity (SG) values were used in tonnage calculations: 2.65 for the intrusive units and 2.71 for the sediment units. Additional SG measurements were collected in 2006 to provide better coverage of deposit rock units and geographic sub-regions. Statistical evaluations of these SG values showed that they were similar to the historical intrusive and sedimentary rock SG values. Therefore, the historic values were used for the Mineral Resource estimate in Section 14.

The following methodology was used to determine SG:

- Samples of whole core approximately 2" to 4" (5 to 10 cm) in length were first weighed dry and then weighed in water. The dry weighing tray assembly was replaced with a wire basket and the sample was submerged in a five-gallon bucket of water. A small tare weight (to compensate for the removed weighing tray) was attached midway up the wire assembly to facilitate alternating wet and dry measurements.
- The formula for SG calculation was: Weight in Air/(Weight in Air – Weight in Water). The specific gravities were automatically computed in acQuire when the weights were entered into the database.
- Measurements were collected for all rock types at a minimum frequency of one sample from all logged rock type intervals and one sample every 49.2 to 65.6 ft (15 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 were also collected to document potential variability.

The weighted average of all SG data points was 2.69. Table 11-1 summarizes the average SG values by rock type. Data points clearly identified as outliers were removed before the average was determined.

It was noted that SG values were generally similar among the rhyodacite units (2.63 to 2.67); therefore, the SG measurements were re-evaluated for the three main rock groups—rhyodacite, greywacke, and shale. Historically, only two SG values were used in tonnage calculations, 2.65 for the intrusive units and 2.71 for the sedimentary units. Because the grouped average SG values are similar enough to the historical values used in previous estimates, it was determined that the values used in Table 11-2 would be sufficient for tonnage estimation and that a block model of SG estimates was not warranted. In addition, the blocks contained within the overburden model were set with a reasonable SG value of 2.14. No new SG analysis was undertaken for the DC9 block model of 2009.

Table 11-1: Specific Gravity Values by Rock Type

Rock Code in DH Database	Rock Types	No. of Samples	Specific Gravity
ARG	Argillite	272	2.67
CGL	Conglomerate	9	2.71
FTZ	Fault Zone	25	2.75
GWK	Greywacke	2,368	2.71
MD	Mafic Dyke	473	2.73
MZD	Monzodiorite	2	2.70
RDA	Rhyodacite Aphanitic Porphyry	499	2.64
RDF	Rhyodacite Fine-Grained Porphyry	315	2.67
RDX	Rhyodacite Coarse-Grained Porphyry	1,339	2.66
RDXB	Rhyodacite Coarse-Grained Blue Porphyry	520	2.63
RDXL	Rhyodacite Lath-Rich Porphyry	216	2.64
SLT	Siltstone	838	2.72
SHL	Shale	387	2.70
All Rock Types		*7,370*	*2.69*

Table 11-2: Specific Gravity Values by Grouped Rock Type

Grouped Rock Type	Individual Rock Types	No of Samples	Specific Gravity
Intrusive Rocks	RDA, RDX, RDXB, RDXL & RDF	2,889	2.65
Greywacke	GWK & CGL	2,377	2.71
Shale	SHL, SLT & ARG	1,497	2.70
All Rock Types		*6,763*	*2.68*

11.4 Analytical and Test Laboratories

The primary laboratory for all assaying has been ALS Chemex in Vancouver, BC. During the exploration programs, ALS Chemex held accreditations typical for the time, including, at various times, ISO9001:2000 and ISO 9002, and from 2005, ISO/IEC 17025 accreditation.

Metallurical test facilities have included SGS-Lakefield Research, Hazen Research, and G&T Metallurgical Services (G&T), who are independent, recognized metallurgical testing laboratories. Work has also been performed by test facilities operated by Placer Dome and Barrick. Metallurgical test facilities are not typically accredited.

11.5 Sample Preparation and Analysis

Most core samples from 2005 onward were crushed at the Donlin camp sample preparation facility and pulverized at the ALS Chemex Vancouver laboratory facility. Samples of 2006 core that were split in Anchorage were shipped to an ALS Chemex preparation laboratory for crushing and pulverizing.

The Donlin camp preparation laboratory is housed in a heated steel building. The facility was rebuilt before the 2007 drill campaign to improve process flow and to upgrade ventilation and dust control.

Sample preparation procedures are as follows:

- The entire bagged sample is dried in an oven heated to 90° to 95°C for 12 hours.
- The sample and sample tag are placed into trays for processing.
- Blank samples (one of three QA/QC control samples) are inserted into the sample stream.
- The sample is crushed in a TM Terminator jaw crusher until the end product passes 70% minus 10 mesh (2 mm). Sieve analyses are performed daily to check crush quality, and the crusher jaws are adjusted as necessary. The crushers are cleaned with blank material four times per 12-hour shift and before a new hole is started. Cutting lists also specify special 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 9 oz (250 g) split. This subsample 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 through the jaw crusher and riffle-splitting it to ~7 oz (200 g). When a duplicate is required, the crushed core sample is passed once through the riffle splitter, and each half is split repeatedly to obtain a ~7 oz (200 g) sample.

Final sample preparation and chemical analysis at the ALS Chemex laboratory in Vancouver consisted of the following:

- Splits of crushed core were reduced to rock flour or "pulp" (better than 85% passing minus 200 mesh or 75 μm) in a ring-and-puck grinding mill.
- A 1 oz (30 g) subsample of the pulp was assayed by fire assay-atomic absorption spectroscopy (AAS). Before 2007, the primary gold assay method was Au-AA23, which had an analytical range of 0.005 to 10 g/t Au. The Au-AA25 gold assay method was initiated in 2007 and had an analytical range of 0.01 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 exceeded the analytical limit for a given method were re-assayed by fire-assay and gravimetric finish or "ore grade" fire-assay AAS. Significant drill hole assay intercepts for the 2005 through 2007 programs, as tabulated in Appendix B3, were based on a 1 g/t Au cut-off grade with up to 13.12 ft (4 m) internal dilution and a minimum width of 9.8 ft (3 m). Chemex determined the sulphur content of each sample according to the Leco method. The Leco method was also used to analyze samples flagged for acid base accounting (ABA) for carbon content as well as to determine neutralization potential (NP) and acid potential (AP) according to the industry-standard Chemex ABA procedure.
- Most trace and major element data for drill holes located within the resource model boundary were acquired prior to the 2005 program by various labs using industry-standard acid digestions followed by atomic absorption (AA) or inductively coupled plasma (ICP) instrumental determinations. Subsequent multi-element trace analyses were performed at ALS (Chemex) using aqua regia or four-acid digestions followed by ICP ± mass spectrometry.

On occasion, if the ALS Chemex Vancouver laboratory was unable to complete the preparation, another ALS Chemex laboratory could be used.

11.6 Quality Assurance and Quality Control

11.6.1 1995–2002 QA/QC Protocol

Placer Dome initiated the first QA/QC program during the 1995 drilling campaign. Coarse reject duplicate splits from 10% of the drill hole samples were submitted to an outside lab (Bondar Clegg). Standard reference material (SRM) assay standards and blanks were added in 1996, and an outside lab (Chemex) performed check assays, presumably of coarse reject duplicates. Check assays by a secondary assay lab were apparently discontinued after 1996. A more structured assay QA/QC program, consisting of SRMs, blanks, and duplicates inserted in rotation every 50 ft (15 m) down-hole, 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.

11.6.2 2005–2006 QA/QC Protocol

No resource delineation drilling was conducted in 2003 and 2004. Placer Dome implemented a slightly modified QA/QC protocol in 2005, which Barrick continued in 2006. Three QA/QC samples, consisting of one blank, one coarse reject duplicate, and one SRM, were randomly inserted into every block of 20 sample numbers. Thus, every block of 20 sample numbers contained 17 drill hole samples and 3 QA/QC control samples.

11.6.3 2007–2010 QA/QC Protocol

The batch size submitted to ALS Chemex was increased from 20 samples to 78 in 2007. To avoid sample mixing with products from other sources in the fusion process, the ALS Chemex protocol was based on a fusion batch size of 84 samples, where the lab added six internal control samples, leaving space for 78 client samples in a given batch.

Each batch of 78 samples shipped to ALS Chemex for sample preparation and analysis contained 9 control samples (12%) consisting of 3 each of standards, blanks, and crushed duplicates. Spacing of the SRMs within the batch was left to the judgement of the geologist. Up to 5% field duplicates (remaining half split of core) were added to the sample batch at the discretion of the geologists for geologic reasons.

11.6.4 Standard Reference Materials

There is no information available on SRMs used prior to the 2002 drilling campaign. Two standards, (Std-C and Std-D), were used during the 2002 drill campaign.

Standard reference materials remaining from the 2002 campaign were used at the beginning of the 2005 season. Additional reference material was purchased from Analytical Solutions (OREAS 6Pb and OREAS 7Pb) and CDN Laboratories (CDN-GS-3) when these SRMs were depleted. After the 2005 season, two additional SRMs (Std-G and Std-H) were created from Donlin coarse reject material. These two new standards and CDN-GS-3 were used during the 2006 season.

Nine new “matrix matched” SRMs of varying gold grade were added in early 2007, and the older standards were eventually phased out. The new SRMs were created from coarse reject samples from throughout the deposit. Composites of this material were pulverized and homogenized at CDN Laboratories in Vancouver, BC. A Barrick geochemist certified the 2007 SRMs after industry-accepted round-robin assay and statistical analyses.

The final SRMs included four each from unoxidized igneous and sedimentary host rocks and one oxidized igneous rock SRM.

11.6.5 Blank Materials

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 for all subsequent drill campaigns.

11.7 Databases

The work completed by Placer Dome and predecessors before 2001 was collected and compiled into a main Microsoft Access database. NovaGold compiled the Placer Dome database into an updated Access database and added information from work completed in 2001 and 2002.

Placer Dome contracted ioDigital to convert 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 into the acQuire database.

Barrick subsequently used acQuire software to capture 2006 and 2007 drill hole data, which were 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 rest of 2006 and through 2007. Geologic and sample data were entered into the Donlin camp acQuire database and merged into the master database as needed. The acQuire database was converted from the standard acQuire data model to the more robust acQuire "Corp" data model in early 2007.

These database procedures were continued for all subsequent programs.

11.8 Sample Security

For all drill programs following the initial involvement of Placer Dome in the Project, core samples are transported from the field and 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 a Barrick 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.

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

Whole core shipped from camp to the facility was transported by 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. Similar protocols were followed for split core samples shipped from camp to the ALS Chemex Fairbanks laboratory. Bagged split core samples were tied into shipping bags and loaded into palletized supersacks 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 camp.

11.9 Comments on Section 11

Sample collection, preparation, analysis and security for all Placer Dome, NovaGold, Barrick, DCJV, and Donlin Gold core drill programs are in line with industry-standard methods for gold deposits:

- Drill programs included insertion of blank, duplicate and standard reference material samples
- QA/QC program results do not indicate any problems with the analytical programs (refer to discussion in Section 12)
- Data is subject to validation, which includes checks on surveys, collar co-ordinates, lithology data, and assay data. The checks are appropriate, and consistent with industry standards (refer to discussion in Section 12)
- Independent data audits have been conducted, and indicate that the sample collection and database entry procedures are acceptable
- All core has been catalogued and stored in designated areas

The QPs are of the opinion that the quality of the gold analytical data from the Placer Dome, NovaGold, Barrick, DCJV, and Donlin Gold drill programs are sufficiently reliable to support Mineral Resource and Mineral reserve estimation without limitations on Mineral Resource confidence categories.

12.0 DATA VERIFICATION

12.1.1 AMEC (2002)

As a test of data integrity, the data used to estimate the January 2002 Donlin Creek Mineral Resources reported in the February and March 2002 Technical Reports (Juras, 2002, and Juras and Hodgson, 2002) were validated. AMEC concluded that the assay and survey database used for the Donlin Mineral Resource estimation at that time was sufficiently free of error to be adequate for support of Mineral Resource estimation.

12.1.2 NovaGold (2005)

NovaGold conducted a 100% check of 2005 drill hole gold assays within the 2005 resource area against electronic assay certificates. An error rate of less than 1.5% was noted.

NovaGold also checked the 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.

As a result of the verification, NovaGold in 2005 considered that the database at the time was adequate to support Mineral Resource estimation.

12.1.3 NovaGold (2008)

In support of preparation of a technical report on the Project in early 2008, NovaGold undertook a data review of the 2006 and 2007 drilling. Data reviewed included:

- Drill collar locations: The Ashtech output files and geologic logs were compared to 5 percent of the electronic collar surveys. There was one unexplained 20-cm discrepancy between the elevation file and the database. Although a number of errors were noted in the geological survey tables, and attributed to likely use of proposed, rather than final, collar co-ordinates, NovaGold concluded that the collar surveys from the Ashtech data files were sufficiently error free to be used for support of Mineral Resource estimation.
- Down hole surveys: 10% of the drill holes were checked and an error rate of 4.4% was measured. NovaGold recommended that DCLLC review their down-hole survey transcription protocols and complete a 100% check of the down-hole survey database. Despite the high error rate, the magnitude of the errors was small;

therefore, in NovaGold's opinion the impact on the estimation of grade was likely to be minimal

- Assay data: For 2006 drilling, 70% of the assays were compared and an acceptable discrepancy rate of 0.4% was measured. For 2007, 99% of the assays were compared to the electronic assay certificates and a discrepancy rate of 1% was measured. NovaGold recommended that the source of the discrepancies be identified. NovaGold believed that the assay database was sufficiently error free to be used for Mineral Resource estimation.

12.2 AMEC (2011)

AMEC reviewed the March 2008 through December 2010 QAQC information for the drill hole data used to construct the DC9 model from with the following results:

- Certified Reference Materials Results: AMEC reviewed the results for 691 CRMs from 2,078 samples and calculated that the relative bias for all CRMs is within the acceptable limits of $\pm$5% relative bias (mean/best value)/1).
- Blank Material Results: AMEC reviewed the results for 694 blank samples submitted for analysis. There were 4 samples which returned higher than allowed gold assays which may be due in part to mislabelled samples. This is an infrequent occurrence and will not affect project assay results, thus AMEC considers that there is no significant risk to the resource estimate.
- Coarse Reject Duplicate Results: The absolute value of relative differences (AVRD = |pair difference|/pair mean) of the crush duplicate pairs that have pair means greater than 1 ppm Au show that 90% of these pairs agree within 15 percent; this indicates acceptable precision is achieved thus the current sample preparation procedure (crushing and pulverizing).

12.3 Comments on Section 12

AMEC considers that a reasonable level of verification has been completed during the 2011 data review, and that, between this review, and reviews completed by NovaGold in 2007 and 2005, and AMEC in 2002, no material issues would have been left unidentified from the verification programs undertaken.

The QPs, who rely upon this work, have reviewed the appropriate reports, and are of the opinion that the data verification programs undertaken on the data collected from the Project adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposits, and adequately support the geological interpretations, the analytical and database quality.

No problems with the database, sampling protocols, flowsheets, check analysis program, or data storage were identified that were sufficient to preclude the use of the database for estimation purposes.

Drill data are typically verified prior to Mineral Resource estimation by comparing data in the Project database to data in original sources. For most of the data, the original sources are electronic data files; therefore, the majority of the comparisons were performed using software tools.

AMEC recommends that Donlin Gold performs a comparison between the trench samples and core/rotary samples to determine if there is any bias that may affect the resource estimation, as trench sampling is used to support the estimate.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Testwork

13.1.1 Domains

Lithological and Geological Domains

Mineralization at Donlin is temporally and spatially associated with rhyodacite dikes and sills intruded into sediments of the mid-Cretaceous Kuskokwim Group. The sediments are predominantly inter-bedded greywacke and shale. Six intrusive phases and two major sedimentary phases have been recognized. The lithological domains and their respective percentages within the Donlin deposits are summarized in Table 13-1. The listed intrusive lithologies are ranked in order of relative age.

Due to poor continuity and minor occurrence within the deposit, the Mafic Dike (MD) intrusive lithology is not separately defined within the geological model, and therefore its relative content within the ore is not accurately defined. Based upon lengths of ore intercepts within the exploration drilling, the content of MD in the ore is estimated to be 0.2% of the ore tonnes.

Two main pits have been identified within the current Donlin deposits, Lewis and ACMA, each subdivided into spatial zones typically separated by faults or other key geological structures.

The Lewis pit is dike dominated and contains more sedimentary ore than ACMA. The ACMA pit is sill dominated and typically contains less sedimentary ore and more intrusive ore than Lewis. With regard to the intrusives, the Lewis pit is dominated by RDX and RDXB compared to the ACMA pit area, which is dominated by the RDA, RDX, and RDXL intrusive lithologies.

For the purpose of defining discreet geological domains within the deposit, the Donlin geological team is using a multi-type separation of the deposit based on a combination of both lithology and spatial location. This is undertaken by independently recognizing the main pit spatial domains (Lewis, ACMA, Akivik, 400, Aurora, and Vortex) for the intrusives, but separating out the sedimentary lithologies from every spatial area, and differentiating into the two sedimentary global domains, greywacke (GWK) and shale (SHL). However, for metallurgical interpretation all different types of domain categorization are considered, used, and applied as appropriate. Table 13-2 provides an estimate of the ore tonnes and gold ounces distribution on the basis of geological domain break-up.

Table 13-1: Intrusive and Sedimentary Lithologies of the Donlin Gold Project

Lithology Description	Abbrev.	Relative Intrusive Age	~% Au Ounces LOM (%)	~% Ore Tonnes LOM (%)
Intrusive Phases				
Blue Porphyry	RDXB	Youngest	13.9	15.2
Aphanitic Flow-banded Porphyry	RDA		22.8	22.5
Lathe-rich Porphyry	RDXL		7.1	7.2
Crystalline (Crowded) Porphyry	RDX		28.7	27.6
Fine-grained Porphyry	RDF		0.4	1.4
Mafic Dikes	MD	Oldest	1.5*	0.2*
Sedimentary Phases				
Greywacke	GWK		22.7	20.4
Shale	SHL		4.3	5.4
Total			**100.0**	**100.0**

Note: MD component percentage is an estimate only – and is therefore not included within the total composition

Table 13-2: Major Geological Domains of the Donlin Gold Project

Description	~% Ounces LOM (%)	~% Ore Tonnes LOM (%)
Lewis Intrusive	23.1	26.7
AMCA Intrusive	21.5	16.8
Aurora Intrusive	5.2	4.4
Akivik Intrusive	4.0	4.8
Vortex Intrusive	10.0	12.0
400 Intrusive	6.3	5.9
Oxide	4.0	4.0
Greywacke (GWK)	20.5	20.0
Shale (SHL)	5.4	5.4
Total	**100.0**	**100.0**

Figure 13-1 is a graphical representation of the geological domains in the Donlin deposits.

Gold Mineralization

There are four primary vein types in the deposit area with variable mineralization as summarized in Table 13-3.

Sulphide Mineralization

A number of mineralogical investigations were undertaken in 2004 to 2007, including work carried out by Amtel, Hazen Research, G&T Metallurgy, and Barrick Technology Centre (BTC). Typical sulphide mineralization presence based on these investigations is summarized in Table 13-4.

Figure 13-1: Donlin Gold Project Geological Domains

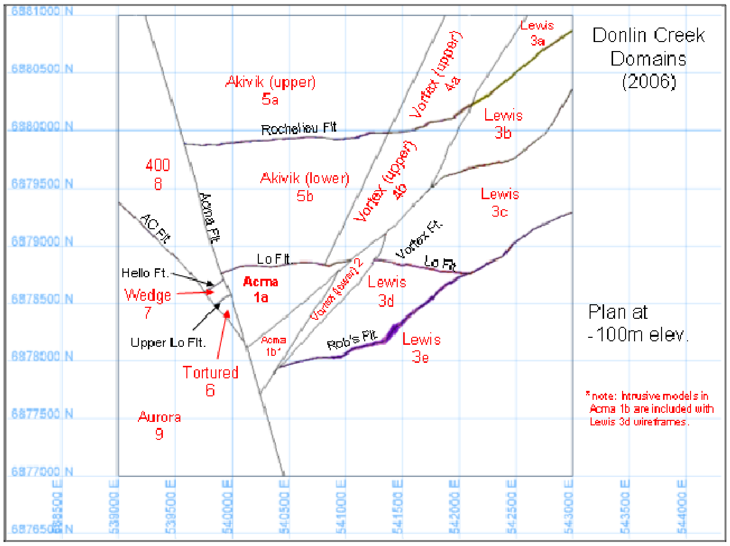


Table 13-3: Vein Types

Vein Type	Dominant Mineralogy	Grade ~(g/t)	Average Orientation	Relative Age
V1	Sulphide	2.7	020/67	Oldest
V2	Qtz-Sulphide	3.9	022/68	
V3	NA, St, Re	7.4	028/72	
V4	Carbonate	0.6	028/65	Youngest

Table 13-4: Typical Sulphide and Metals Mineralization in the Donlin Ores

Typical Occurrence	Sulphides	Chemical Formula
Major to Minor	Pyrite	FeS_2
	Marcasite	FeS_2
	Arsenopyrite	FeAsS
Minor to Trace	Native Arsenic	As
	Realgar	AsS
	Stibnite	Sb_2S_3
Trace	Chalcopyrite	$CuFeS_2$
	Sphalerite	ZnS
	Tetrahedrite	$(Cu,Fe)_{12}Sb_4S_{13}$
	Galena	PbS
	Pyrrhotite	FeS
	Molybdenite	MoS_2
	Bornite	Cu_5FeS_4
	Covellite	CuS
	Mercury Sulphide	HgS in pyrite
	Native Copper	Cu

Pyrite is the dominantly occurring sulphide within the deposit. Marcasite is also present, at an approximate ratio of 1:7 as marcasite to pyrite. Arsenopyrite is the main carrier of arsenic within the deposit. Stibnite is the main carrier of antimony.

13.1.2 Gold Deportment

The primary host mineral of gold within the Donlin ores is arsenopyrite, which carries 80% to 90% of the gold as "solid solution" gold atomically distributed within the arsenopyrite crystal. Fine arsenopyrite is typically highest in gold grade at 500 to 1,500 g/t (mineral gold content), compared to coarse arsenopyrite at ~100 to 500 g/t. A proportion of the arsenopyrite (particularly coarser-grained arsenopyrite) has been shown to have a preferential concentration of gold around the rim of the crystal.

Pyrite is the second most important gold carrier, hosting 10% to 20% of the contained gold, also in "solid solution" form. Typical gold grades of pyrite are 1 g/t to 50 g/t. Similarly, marcasite is also a gold carrier.

Free gold is a minor source of gold (rare occurrence) within the deposit at less than 1% of the contained gold being free liberated particles less than 20 μm in diameter. Native arsenic is an insignificant carrier of gold in terms of both grade and quantity.

13.1.3 Mercury, Chlorine, Carbonates and Organic Carbon Deportment

The Donlin ore hosts mercury at a grade of 1 g/t to 3 g/t. For occupational health and environmental considerations, mercury is an important species considered for the process plant design.

Detailed deportment of mercury was undertaken by Amtel in 2007, which indicated that, based upon the concentrate samples tested, pyrite is the principal carrier (66%) of mercury as HgS in "solid solution" within the sulphide mineral matrix, followed by the sulphide marcasite (18%). The Amtel study also indicated that fine-grained pyrite, marcasite, and stibnite are relatively enriched in mercury content. Arsenopyrite has relatively low mercury content and is an insignificant carrier. No specific mercury minerals have been found or identified in any of the samples examined by Amtel in the deportment work completed.

The Donlin ore contains chloride with an average concentration of approximately 22 ppm. Detailed testing of chloride deportment carried out by Amtel in 2007 indicated that the principal carrier of chloride in flotation concentrate was muscovite (white mica) as $KAl_2(Si_3Al)O_{10}(OH,F,Cl)_2$, followed by hydroxylapatite, $Ca_5(PO_4)_3(OH,F,Cl)$.

The form of carbonates within the ores at Donlin is an important consideration - carbonate within the flotation tails stream is used to neutralize the acidic liquor coming from the autoclave. The three forms of carbonate identified by Amtel in 2007 were calcite ($CaCO_3$), siderite ($FeCO_3$), and dolomite ($CaMg(CO_3)_2$). The predominant carbonate species remaining in flotation tails was dolomite, followed by ankerite. Calcite was only identified in two test samples and siderite in only one test sample. For the samples tested, the Aurora geological domain consistently contains calcite as the dominate carbonate.

The Donlin ores host organic carbon in both the intrusive and sedimentary lithologies. The sedimentary lithologies are relatively abundant in organic carbon. The RDF and RDXB intrusive lithologies are characteristically high in graphitic carbon content.

13.1.4 Samples

The samples used for the metallurgical testwork during 2006 to 2007 were obtained from a variety of sources. Samples for metallurgical testwork were selected to represent the various geological lithologies and geological domains.

13.1.5 Comminution

Grinding testwork for the Project has been conducted periodically since Project inception. The initial grinding testwork was undertaken in the 1990s and very little specific information is available on this work. Subsequently, in 2002-2003 additional work was completed at Hazen managed by NovaGold.

Placer Dome initiated some further work by SGS Lakefield in 2004, which tested ACMA material. During this program, JK and Bond testwork were completed in conjunction with testing the applicability of high pressure grinding rolls (HPGRs) to the deposit. During 2006 Barrick initiated three major testwork programs at SGS Lakefield:

- SGS Lakefield 2006 HQ half-core testwork
- SGS Lakefield 2006 PQ whole-core testwork
- SGS Lakefield 2007 HQ half-core testwork.

The work completed in 2007 for the feasibility study indicated that grinding testwork should preferably be carried out on freshly drilled core that has been protected from the Alaskan weather. It is apparent that the physical hardness properties of the drill core are affected upon exposure to the Alaskan environment while in storage (i.e., reduced in competency, believed to be through a cryogenic weathering effect) and therefore could lead to biased low hardness test results.

Consequently, the test results from the core recovered from fresh exploration drilling undertaken in 2006 were used preferentially as the basis for the FSU design of the grinding circuit.

Testwork Period 2002–2003

A summary of results of the testwork undertaken in 2002–2003 is shown in Table 13-5. The testwork summary indicates that the material tested at that time was moderately hard. These results align with more recent testwork.

Testwork Period 2004–2005

Placer Dome initiated a testing program at SGS Lakefield in 2004. This work was performed on two large samples from the Donlin deposits with the objective of comparing the power efficiency of using high-pressure grinding rolls (HPGR) as opposed to semi-autogenous grinding to prepare the ore ahead of ball milling. The results of the JK and Bond work are summarized in Table 13-6.

The following key items were identified from the testwork:

- The sedimentary sample was described as moderately hard with respect to resistance to impact, as measured by the impact work index (CWI) and drop-weight test (A x b).
- The intrusive sample measured as hard in terms of low-energy impact and medium in the drop-weight test.
- The sedimentary sample was moderately hard with respect to resistance to abrasion breakage (ta), while the intrusive sample was hard.
- Both samples can be categorized as medium in terms of Bond rod mill (RWI) and ball mill work indices (BWI).
- Both samples were only mildly abrasive (Ai).

Table 13-5: Grinding Testwork Results from Hazen Research

Pit	Locale	Ore Type	BWI (kWh/t)
ACMA	-	Intrusive	14.3
	-	Sediment	13.0
Lewis	Rochelieu	Intrusive	14.1
	Rochelieu	Sediment	13.3
Lewis	North	Intrusive	13.9
	North	Sediment	13.1
Lewis	South	Intrusive	15.1
	South	Sediment	12.1

Table 13-6: Summary of Grindability Testing

Sample Composite	CWI (kWh/t)	A x b	ta	Ore Density (g/cm^3)	RWI (kWh/t)	BWI (kWh/t)	Ai (g)
ACMA Sedimentary	9.9	38.7	0.39	2.76	14.7	14.0	0.205
ACMA Intrusive	11.3	52.8	0.31	2.69	13.5	14.7	0.181

- The two samples were also submitted to a series of bench-scale HPGR tests. The tests showed that the HPGR successively reduced the ½" material feed to ball mill feed size with an energy input of 2.07 and 1.94 kWh/t for the sedimentary and intrusive samples, respectively. The BWIs of the resultant samples were tested at 75 µm and produced values of 12.5 and 13.3 kWh/t, respectively, indicating that a reduction (in the BWI) was attributable to the HPGR processing.

SGS-Lakefield recommended further HPGR work at Polysius with the REGRO unit accompanied by ATWAL abrasion testing. The testwork by Polysius judged that a specific energy input in a range of 1.5 to 2.0 kWh/t was the optimum for HPGR comminution of the provided test samples. A specific throughput rate of approximately 250 ts/hm^3 was achieved with the grinding force selected.

The abrasion testwork indicated an ATWAL wear index of 8.3 g/t for sedimentary material and 24.8 g/t for intrusive material. This testwork was done at the standard feed moisture of 1%. This wear is medium abrasive compared with Polysius' database at the time.

SGS Lakefield 2006 HQ Core Testwork

In 2006, SGS Mineral Services (SGS) conducted an extensive test program (Appendix D3-7) to determine grinding parameters for the Donlin ores. The samples (Appendix D2-8) used were HQ drill core taken from the 1999 and 2002 drilling campaigns, which had been stored at the exploration site. Parameters were obtained from the following tests:

- 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), rod mill work index (RWI), ball mill work index (BWI), and abrasion index (Ai)
- High-pressure grinding roll energy test.

SGS Lakefield 2006 Whole PQ Core Testwork

HQ drill core was too small in diameter for full drop weight tests for JKSimMet modelling. Larger diameter PQ holes were drilled in a 2006 drilling campaign, targeting bulk mineralized areas of the deposit, and covering the full range of lithologies. The samples from these drill holes were processed to develop JK grinding parameters, in addition to conventional Bond ball mill and rod mill work index numbers. This information was important for use in checking the grinding parameters developed from the HQ testwork. It was seen by comparing work index properties within lithologies from the PQ core results and the 2006 HQ core results that the freshly drilled PQ core was consistently harder than the HQ core samples drilled in 1999 and 2001. At that point in the study, it was recommended that a second set of HQ samples be selected from freshly drilled core that became available during 2006.

SGS Lakefield 2007 HQ Core Testwork

With the availability of additional fresh HQ drill core from the 2006 exploration program, a second phase of variability testing was initiated in early 2007. A total of 149 additional samples were tested.

In parallel with this testwork AMEC undertook grinding circuit design trade-off studies in 2006, which indicated that a semi autogenous–ball mill-crushing circuit (SABC) circuit design was the preferred option. Therefore, the test program was designed to maximize generation of hardness properties relating to the required parameters for the SABC circuit. The parameters tested in the program were then restricted to:

- Minnovex SAG power index (SPI)
- Minnovex crusher index (Ci)
- Minnovex modified Bond ball mill work index

- Rod mill work index (RWI)
- Ball mill work index (BWI).

Once again, it became apparent that within lithology, the results from the 2007 test program indicated consistently harder comminution properties than the 2006 test program.

Therefore, it was recommended that comminution data obtained from core predating 2006 be normalized to fresh rock conditions for design calculations when it is known that the core has been exposed for an extended period to Alaskan climate. Such normalization was actually carried out on the 1999 and 2001 HQ core data.

Comparison of 2006 Drilled and 1999-2001 Drilled Half HQ Core

Table 13-7 directly compares the average results by lithology between the two crushing and grinding hardness data from the 2006 (core drilled 1999/2001) and 2007 (core drilled 2006) testwork programs.

It should be noted that for the test JKTech parameters Axb and Crushing Index, ore hardness is inversely proportional to the test result (i.e., lower numerical result is physically harder to process, requiring more power).

Given that the 2007 program used the freshly drilled 2006 HQ core, these results were regarded as being a more reliable predictor of the hardness of the deposit. However, the 2006 test results were preserved to provide more test samples to augment the variance analysis and subsequent population of the geological model. Minnovex adjusted the 2006 variability results to match the hardness distribution of the later (harder) test results.

Table 13-8 is a summary of the adjustments for each lithology, derived by splitting the data into separate lithological groups, comparing the frequency distribution of the 2006 and 2007 sample data, and then determining a simple adjustment to move the 2006 data to fit the form of the 2007 distribution.

After the adjustment to each of the lithologies within the 2006 test data, the two 2006 and 2007 data sets (Ci, SPI, BWI) were combined and forwarded to AMEC for geostatistical review and modelling, to provide a block-by-block mill feed hardness schedule for the parameters of Ci, SPI, and BWI. With these parameters, it is possible to use a Minnovex CEET (comminution economic evaluation tool) grinding model to predict the milling capacity and power requirements for each ore block in the designed Donlin circuit.

Table 13-7: Comparison of Average Results from 2006 and 2007 Test Programs

	JKMRC, A x b		SPI, Minutes		Crushing Index		RWI, kWh/t		BWI, kWh/t	
Lithology	2006	2007	2006	2007	2006	2007	2006	2007	2006	2007
RDX	46.0	42.7	47.6	90.5	17.2	14.9	13.3	15.1	14.6	15.6
RDA	41.0	41.1	53.8	90.1	17.3	14.6	14.0	15.0	12.4	13.8
RDXL	54.2	48.5	44.6	72.3	22.5	21.3	13.6	14.0	13.6	14.3
RDXB	50.0	49.4	53.7	91.9	18.8	15.3	13.7	15.5	14.6	16.4
GWK	41.8	34.4	55.9	94.7	16.6	12.5	14.5	15.2	13.3	14.7
SHL	50.2	42.2	48.4	58.1	21.7	16.0	13.4	16.7	13.5	14.4
RDF	37.0	30.1	61.9	100.8	18.3	14.2	13.9	14.8	15.1	15.3
Average	**45.7**	**41.2**	**52.3**	**85.5**	**18.9**	**15.5**	**13.8**	**15.2**	**13.9**	**14.9**

Table 13-8: Adjustments Made to the 2006 Test Program Data

Lithology	Ci 2006 Correction	SPI 2006 Correction	BWI 2006 Correction
RDA	x/1.2	1.8*x+6	1.09*x
RDX	x/1.1	1.35*x+18	1.07*x
RDXB	x/1.1	1.1*x+33	1.17*x
RDXL	x	1.2*x-18	1.07*x
RDF	x/1.3	1.2*x+27	1.05*x
GWK	x/1.3	1.8*x+2	1.12*x
SHL	x/1.5	1.3*x	1.07*x
MD	x/1.8	1.8*x	1.09*x

Note: where * = multiplication

Effect of Grind Size on BWI

The potential adoption of an MCF2 flowsheet would result in two different ball mill product size distributions. The products from the primary ball milling circuit would target P_{80} 120 to 150 µm, and the secondary ball mill would target P_{80} 50 µm. From previous work by others, it is known that in some circumstances the measured BWI of a test sample will vary according to the target P_{80} of the BWI test.

To determine if this is the case for Donlin, and to quantify that effect, a number of additional BWI tests at different final product sizes were undertaken by SGS Lakefield. For the feasibility grinding circuit modelling and design, an overall adjustment model was applied to the BWI number used, based on these results.

Using the blended pilot-plant feed sample, an additional set of BWI tests was undertaken at varying product sizes.

From the results obtained, it was seen that as the target product P_{80} size increased in fineness, the measured BWI of the test sample increases. On the blended pilot-plant sample a significant increase in BWI occurs between P_{80} 54 µm and 42 µm. Figure 13-2 is a graphical representation of the pilot-plant blend feed sample results.

Feasibility Comminution Circuit Selection

In 2006, Barrick contracted Orway Mineral Consultants (OMC), Australia, and SGS Lakefield (Minnovex), Canada, both of which have specialist groups in grinding circuit modelling and design, to perform appraisals of the various comminution options. The consultants were asked to examine the testwork information and provide capital and operating cost alternatives for four options:

- Option 1 – autogenous milling with ball milling and crushing of pebble reject (ABC)
- Option 2 – semi-autogenous milling with ball milling and pebble crushing (SABC)
- Option 3 – coarse crushing followed by HPGR with ball milling
- Option 4 – fine crushing followed by ball milling.

Both consultants created grinding models based on their internal databases and calculated capital and operating costs.

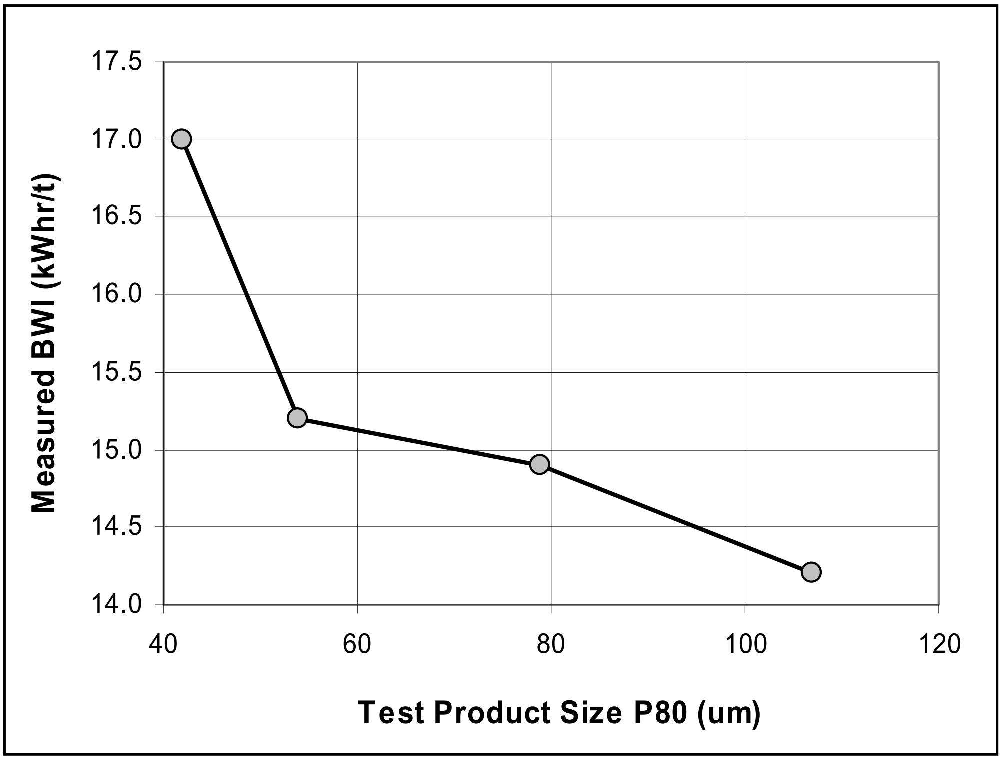
AMEC then performed a trade-off study based on the OMC recommendations to investigate the economics of various options as well as non-economic factors. AMEC also performed a check analysis on the various models to ensure that these were reasonable in their execution. It should also be noted that since the OMC and SGS work was done, it has become apparent that the samples from the 1999 and 2002 drill campaigns may have weathered sufficiently to affect the primary mill grindability testing. Consequently, AMEC also performed a separate analysis using only the results from the samples obtained in the 2006 drilling campaign.

The SABC circuit, Option 2, was selected as the comminution circuit for Donlin, 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.

Figure 13-2: Test P_{80} vs. Measured BWI Results on Blend Composite Sample



Although the lower operating costs ascribed to Options 3 and 4 indicate a long-term financial advantage, these options are considered unwarranted in use because of uncertainty with regard to their ability to achieve the operating costs and stated availabilities. In particular, the natural high clay content of the Donlin ores would hinder the performance of these two types of circuits, particularly as ore moisture increases.

Geostatistical Assessment

To better define the hardness characteristics of the scheduled mill feed, the compiled comminution data set for Ci, SPI, and BWI was provided to AMEC (U.S.) to build a metallurgical model, which served as the basis for the geostatistical assessment. The purpose of the metallurgical model was to develop the supportable relationships that might exist between the ore sample hardness, rock lithology, spatial location, ore grades (Au, S, As, Mg, Sb), and RQD (rock quality designation), and to use these relationships to populate the geological block model with the ore hardness properties

Ci, SPI, and BWI. The key results of the geotechnical assessment are summarized below.

No correlations could be determined between these three grinding hardness properties, and therefore each parameter had to be assessed individually.

- Variability of the crushing index data is quite high within and between lithologies, and is dominated by lithology as indicated in Table 13-9.
- The SPI results are dominated by lithology, with the differences between lithologies being significant. Based on analysis of the variance of the data, four separate categories were defined and used to populate the block model. These are summarized in Table 13-10.
- Lithology was determined to be the significant variable influencing ball work index (Table 13-11). Within a lithology, the variance in results was quite low, with the exception of the SHL and MD lithologies, due to a small number of test results.

MCF2 Grinding Circuit Design Method and Capacity Definition

During the first quarter of 2007, pilot flotation testing was carried out at SGS Lakefield to assess the potential of an mill chemical float, twice-style (MCF2-style) flowsheet to improve confidence in the overall gold recovery and economic value of the Project (Figure 13-3). The MCF2 flowsheet incorporates two separate stages of grinding and flotation. An economic evaluation was completed and it was decided to use the MCF2 flowsheet as the basis of the circuit design for the 2007 FS and FSU, utilizing the SABC configuration as the primary grinding step.

In 2007, a throughput confirmation study carried out by the mining team, indicated that a Project optimum ore production rate was in the order of 49,600 stpd (45,000 t/d). In 2008, an updated resource model based on additional drilling was completed and the mill throughput was increased to 59,000 stpd (53,500 t/d).

Mineralogical assessment, flotation bench-scale, and flotation pilot-scale testwork have indicated that the primary rougher feed particle 80% passing size distribution (P80) should be in the range 120 to150 µm, and that the secondary rougher feed P80 should be in the range 50 to 60 µm, to achieve high flotation gold recoveries. These assumptions were applied to the grinding circuit modelling and design.

Based on a common hardness data set and the conventional SABC circuit, three different grinding circuit modelling techniques were used (JKSimMet, CEET, and a power-based model referred to as DJB) to “calibrate” a CEET model to represent an agreed design model set-up.

Table 13-9: Orebody Estimation of Crushing Index (Ci)

Domain	Ci Mean	Std. Dev.	Sample Count
RDXL	21.9	8.2	33
Non-RDXL West (<541,000)	14.2	6.7	158
Non-RDXL East (>541,000)	14.8	6.9	81
All	**15.3**	**7.4**	**272**

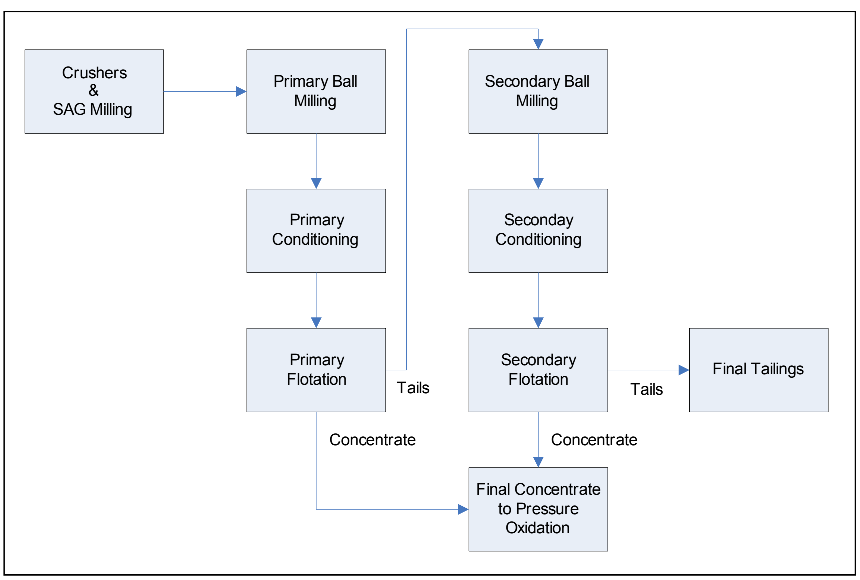
Table 13-10: Orebody Estimation of SAG Power Index (SPI)

Domain	SPI Mean	Std. Dev.	Sample Count
SHL	58	17	9
RDXL	71	14	33
RDX	85	22	79
Remaining Lithologies	95	21	152
All	**88**	**23**	**273**

Table 13-11: Orebody Estimation of Bond Ball Work Index (BWI)

Domain	BWI Mean	Std. Dev.	Sample Count
RDX + RDF + MD	15.5	1.3	107
GWK + SHL	14.6	1.5	48
RDA	13.8	1.3	46
RDXL	14.4	1.1	33
RDXB	16.4	1.2	39
All	**15.0**	**1.5**	**273**

Figure 13-3: Illustration of MCF2 Generic Flowsheet



At the 2008 target throughput of 59,000 stpd (53,500 t/d), the design parameters collated in the steps above suggested that large mills and power input would be required. The mill size was approaching the production capacity limitations of the largest existing and operating SAG mill (40 ft diameter) and ball mill (18 MW).

Based on the adjusted CEET grinding model and using the "calibration" dataset, a single 38 ft SAG mill (20 MW) followed by a single 18 MW ball mill could process ~59,000 stpd (53,500 t/d) to produce a primary flotation feed P_{80} of 120 to 150 µm. This could then be followed by another 18 MW ball mill to produce the required secondary rougher flotation feed of P_{80} of 50 µm.

In the design moving forward, therefore, a smaller 38 ft diameter SAG mill with a corresponding motor size of 20 MW was adopted, while still retaining two 18 MW ball mills.

Plant Ramp-Up

The Project ramp-up schedule was defined on the basis of operating data available to Barrick Gold Corporation from other sites. The sites used for the comparison were selected on the basis of being large SABC circuits with some additional downstream process sections, and where reasonable ramp-up data were available.

This ramp-up schedule provides an additional constraint to the CEET model for capacity scheduling. Figure 13-4 and Figure 13-5 show the assumed feasibility ramp-up schedules for the plant utilization and throughput, as a percentage of design.

Downstream Autoclave Productivity Limits

Given that the milling circuit provides feed to flotation, and that the flotation concentrate generated is then processed through a site-based pressure oxidation unit, CEET must consider downstream constraints to production. The pressure oxidation facility has a design capacity of 714 stpd (648 t/d) of sulphur.

Therefore, for example, during periods of high-grade sulphur content in the mill feed, the pressure oxidation units limit the upstream (grinding circuit) plant throughput. Sixty days of downtime for complete autoclave re-lining have been incorporated into the LOM schedule every six to seven years.

Figure 13-4: Plant Utilization Ramp-up Schedule for Donlin Feasibility Compared to Other Available Commissioned Sites

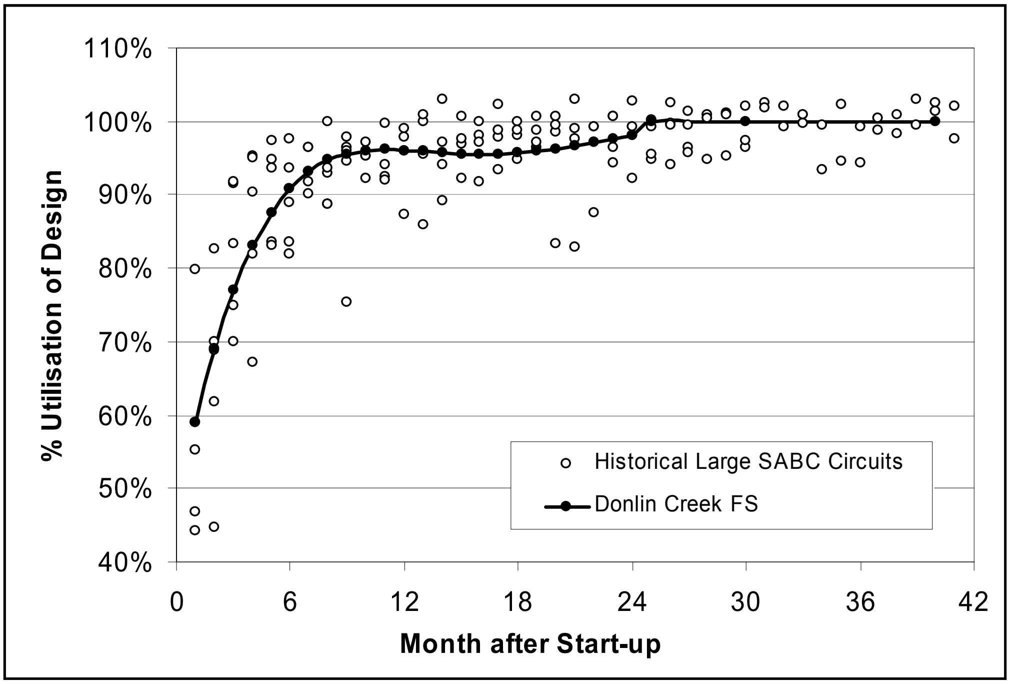
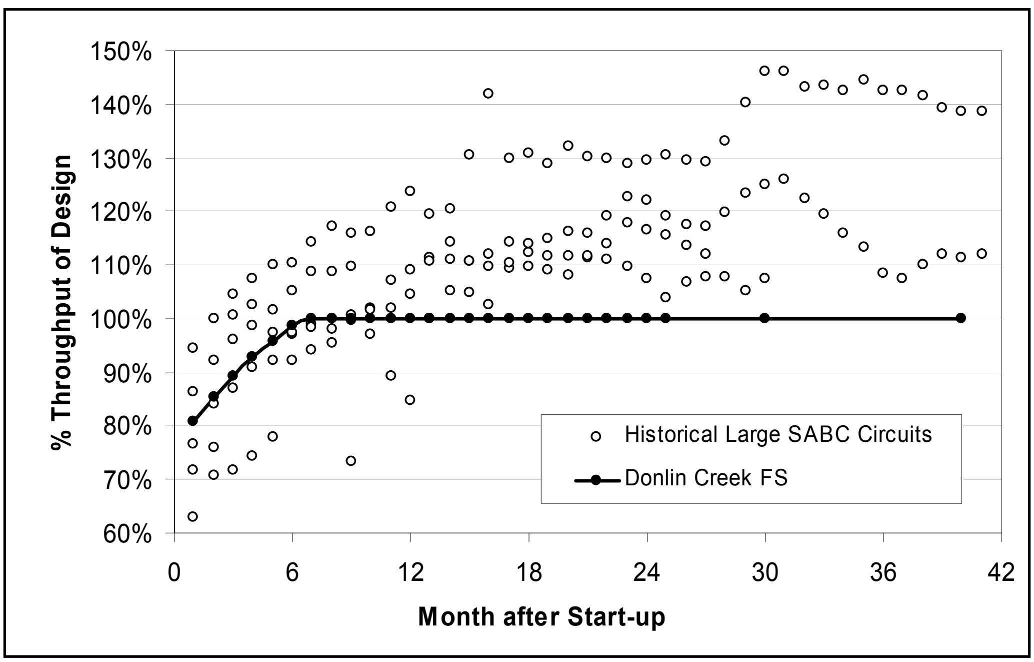


Figure 13-5: Plant Throughput Ramp-up Schedule for Donlin Feasibility Compared to Other Available Sites



Mill Feed Schedule Definition

With the CEET model set up with the recommended grinding equipment sizes, the sulphur productivity limits, and the ramp-up schedule, it is then possible to model the mill feed schedule from mining on a block-by-block basis to determine the operating capacity of the milling circuit to process the ore on a period-by-period basis, taking into account all the plant constraints.

Continuous Improvement Assumptions

Nominal continuous improvement targets have been defined over the life of mine to account for anticipated productivity gains earned through operating experience and promoting a continuous improvement culture at the mine. These increases are not significant in size and are considered to be within the existing plant design limitations. The fixed constraints of the designed and installed grinding circuit power and oxygen plant capacity are not exceeded within any given period regardless of these improvements.

Table 13-12 summarizes the productivity improvement assumptions used for the scheduling of FSU2.

13.1.6 Flotation

Introduction

Extensive bench and pilot flotation testwork has been carried out on samples of the Donlin ores from 1995 through to mid-2007.

The objective for the flotation circuit within the flowsheet is to provide high gold recovery (+90%) to a high-grade sulphur concentrate (greater than 6.5% total sulphur content) for pressure oxidation feed. Once the feed contains sufficient grade of sulphide sulphur as fuel to generate heat and achieve the required operating temperatures within the autoclave, there is little overall net benefit in increasing the sulphur grade of the concentrate any further. This is advantageous for Donlin because the mineralogy of the ores produces a gold recovery in flotation concentrate increases significantly as concentrate grade is reduced.

Table 13-12: Productivity Improvement Assumptions for the FSU

Production Period	Component	Assumption (% of Design)
Year 3 & Onwards	Milling Circuit Capacity	100.0
Year 6 & Onwards	Milling Circuit Utilization	101.5
Year 6 & Onwards	Autoclave Sulphur Treatment Rate	101.5
Year 6 & Onwards	Autoclave Circuit Utilization	100.0

For Donlin, a general target of 7% sulphur grade has been selected for the final flotation concentrate, noting that the autogenous grade is determined to be approximately 6.5% sulphur, varying slightly with changes in the solids/liquid ratio, the carbon, and arsenic grade of the autoclave feed; the gap between 6.5% autogenous grade and the target 7% grade is provided to allow for variability in feed and to minimize any requirement to add external heating to the autoclaves.

AMEC Review of Testwork from 1995 to 2006

The key conclusions and comments from the AMEC review of the 1995 to 2006 flotation testwork are as follows:

Bench Testing

- The testwork results indicated that producing a bulk concentrate was the optimum route to maximize gold recovery
- A variety of reagent schemes were attempted. The best reagent system was using plant acid, copper sulphate, Xanthate, and dispersant
- Nitrogen-based flotation technology was tested extensively throughout the Project but provided no overall benefit
- Adequate retention time was found to be very important. To achieve maximum recovery, a long flotation retention time, of 114 minutes, was found to be necessary
- The required particle grind size reflects the presence of the two different ore types in the feed. While the intrusive ores can tolerate coarser sizes in the range of 75 to 110 µm, the sedimentary ores perform best in the range of 60 to 80 µm for the conventional flotation flowsheet
- Mass pull from the rougher and scavenger circuits was dictated by entrainment of clays during the long residence time of the flotation. With the use of dispersants, lower flotation feed pulp densities and cleaning, the overall mass pull to final concentrate could be decreased to approximately 15%

- The process development testwork indicated that froth recovery was a critical factor. Froth recovery can be enhanced by the use of crowding cones in the flotation machine and through launder design
- Because of the different flotation response of the ores, testwork was performed to assess the outcome of blending the two main ore 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.

Flotation Pilot-Plant Testing

- Extensive testwork was performed in 2004 and 2006 leading to the demonstration that a recovery of 91% to 92% on a LOM lithology blend was possible, confirming the performance of the conventional rougher / scavenger flowsheet under continuous operation.

Mineralogy Summary

In summary, the various mineralogy studies undertaken on many different flotation test samples by a number of different investigators present a relatively consistent themes concerning the aim to achieve high gold recovery (+90%) to concentrate:

- Fine arsenopyrite must be recovered to final concentrate
- Many ores (particularly Lewis ores) must be ground finer than P_{80} of 75 µm to improve the liberation characteristics
- Pyrite hosts a significant portion of the gold as solid solution gold within the crystal matrix, and therefore must also be recovered to concentrate
- Over-grinding of the liberated sulphides to less than 10 µm diameter particles would be detrimental to flotation recovery
- The liberation properties of the sulphides within the Donlin ores are somewhat variable
- The expected concentrate sulphur grade from a Donlin flotation circuit producing high gold recovery (+90%) is going to be relatively low, less than 10% sulphur content, because of the presence of low-grade composite particles of sulphide with gangue, floatable gangue (such as carbon and carbon/clay binaries), and non-floatable gangue as entrainment.

Pilot-Plant Testing (G&T Metallurgy Q3 2006)

Testwork was performed at G&T in Q3, 2006 to confirm process parameters on a blend (50% Lewis Intrusive, 25% ACMA Intrusive, and 25% Sedimentary). This pilot flotation testing did not produce flotation gold recoveries (at required grade) that were comparable to those achieved from a bench test undertaken on the same sample. Pilot results showed recovery in the range of 83% to 85% in the blended sample, compared to the bench flotation test at 91% to 93%.

Subsequent testing was undertaken to explain this discrepancy through exhaustive re-testing of both the bench and pilot-plant flotation cells under different operating and test conditions. This work led to the conclusion that the froth conditions generated by the G&T Metallurgy pilot plant were hindering recovery of sulphide composite particles to the concentrate. As a consequence the importance of froth recovery was emphasized in design.

Pilot-Plant Testing (SGS Lakefield December 2006)

The SGS pilot run in December 2006 was initiated after the G&T Metallurgical work was completed in 2006, when it was reasonably understood which key items were affecting gold recovery under pilot conditions.

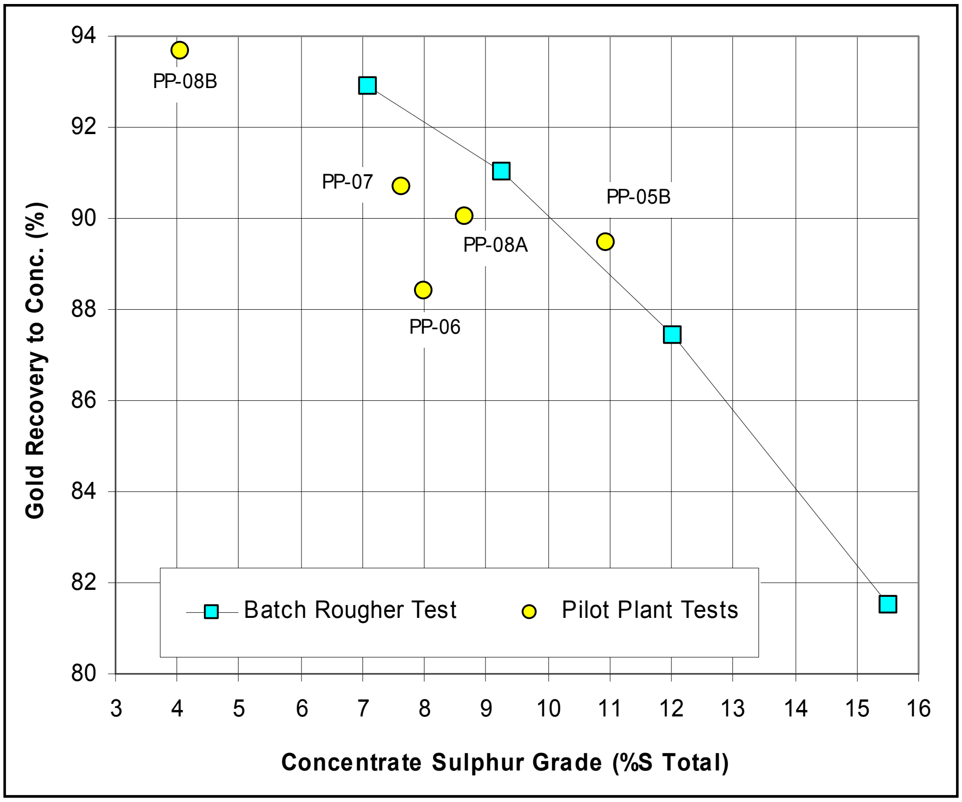
The purpose of this pilot run was to demonstrate the ability of closing the gap between bench and pilot performance and to understand the performance of the new proposed flowsheet. This pilot-plant testwork was performed on available material and later followed up by another pilot run using freshly drilled core to eliminate any issues related to sample weathering.

It should be noted that the generally poorer flotation results were attributed to partially geologically oxidized ore in some of the upper areas of the deposit, as evident from the 2007 variability testing. This was not understood at that time, and therefore the December 2006 composite sample included material that was geologically oxidized.

From the pilot-plant run approximately 91% gold recovery to a 7% sulphur concentrate was achieved compared to a bench test result of 93% recovery on a LOM lithology blend. This was a significant improvement from the earlier pilot work in 2006.

At this point in the study, to further improve the overall performance of the flotation circuit design, an alternative MCF2 flotation configuration (Figure 13-6) was considered, and some comparative bench testing of this option commenced.

Figure 13-6: Comparison of SGS Lakefield Dec 2006 Key Bench and Pilot-Plant Results



Bench Testing of MCF2 Alternative Flowsheet

In early 2007, a series of bench flotation tests were initiated to explore the potential benefit of the MCF2 circuit configuration.

Initial tests were conducted by SGS Lakefield at a nominal grind of P_{80} 40 µm for the second stage product and with varying primary stage grind sizes. The results showed that the alternative MCF2 grind/flotation configuration was realizing a measurable improvement in gold recovery of approximately 2% at the same final concentrate grade as the conventional grind/float configuration.

These tests also suggested that the primary grind size selection was not a key parameter and that there could be some flexibility in the final size selection for the stage of grinding.

Subsequent bench tests were then undertaken to attempt to quantify the effect of the secondary grind size on gold recovery. The results did not clearly define an optimal secondary grind target but a nominal 50 µm (P_{80}) was selected for the subsequent MCF2 pilot run based on trends identified through previous mineralogical work.

Pilot-Plant Testing, (SGS Lakefield February 2007)

A second SGS Lakefield pilot-plant campaign was undertaken February to March 2007, with the primary aim to confirm pilot-plant recovery of the conventional flotation circuit on a LOM lithological blend of freshly-drilled core.

The pilot runs produced on average 92.8% recovery to a 7% sulphur concentrate grade compared to the bench tests, which indicated ~94.0% recovery. The second series of pilot runs was undertaken and confirmed that a mild steel primary mill with high chrome media could be used in lieu of the original stainless steel mill with high chrome media. A third series of pilot tests was undertaken where the scavenger concentrate was reground before cleaning. A slight improvement in the grade/recovery profile was evident. A fourth series of pilot tests was conducted to evaluate the MCF2 configuration. A clear trend of improved recovery is evident. At a concentrate of 7% sulphur grade, the recovery difference between MCF2 and conventional flotation is ~1.8%. This compares reasonably well to the ~2% recovery improvement indicated by the initial MCF2 screening bench test results.

Based on the Phase 2 piloting at SGS Lakefield, it was recommended that the MCF2 option be selected evaluated as a potential base case for the Donlin feasibility grinding/flotation circuit design.

Variability Testing (SGS Lakefield, Q2, 2007)

The flotation variability testwork program consisted of a total of 149 flotation tests using 102 different test samples selected to cover a range of different lithologies and geological domains from core drilled throughout 2006. The number of samples selected for each lithology was based on the proportion of the orebody represented by that lithology

Of the 102 samples, 22 were characterized as having some form of partial geological oxidation, and the remaining 80 were considered unoxidized fresh rock. Two types of variability bench-scale flotation tests were carried out, a modified Minnovex Flotation Test (MFT), and a conventional bench flotation test (CFT).

Analysis of test data indicated:

- RDX, RDA, RDXL, and MD lithological domains appear to behave similarly in terms of average and standard deviation, and together have an average flotation recovery of ~96%.
- GWK and SHL recoveries are, on average, lower (91.5% and 89.8%, respectively), with a relatively large variation in performance.
- RDXB lithology is the worst-performing intrusive, with an average recovery of 94.8%. This is characteristic for RDXB, which has relatively high graphitic carbon content compared to the other intrusives.
- RDF is the best-performing intrusive, with an average gold recovery of 97.7% and relatively low variance in performance.
- The GWK domain dataset does exhibit a weak correlation of flotation recovery with both arsenic and gold grade, noting that there is a natural strong relationship between gold and arsenic head grades. Given the relatively poor correlation with the GWK dataset this type of relationship is not recommended to predict recovery for the GWK ores. Instead, a non-weighted average of the test data should be used.
- For the SHL test results, no obvious correlations are evident to improve recovery predictions, and a non-weighted recovery average should be used here as well.

Another methodology for characterizing the test samples is via geological domain, which is mainly based on physical location rather than rock type.

Examining the statistical representation of recovery data based on geological domain suggests:

- Samples from 400, ACMA, Akivik, and Aurora behave fairly similarly, with an overall average recovery of ~96.5%.
- GWK and SHL recoveries are, on average, lower (91.5%, 89.8%, respectively), with a relatively large variation in performance.
- Lewis is the worst-performing intrusive, with an average recovery of 94.3%.
- Vortex is the medium-performing intrusive, with an average recovery of 95.2%.

The Vortex geological domain is adjacent the Lewis domain; the Lewis geological domain has a high content of RDXB intrusive, and conversely, RDXB is concentrated mainly in the Lewis area of the deposit.

Effect of Geological Oxidation on Flotation Performance

The upper portion of the Donlin orebody contains ore that shows some sign of geological oxidation or weathering. To quantify the potential impact of this oxidation on flotation at Donlin, a series of 22 samples were selected. Results indicated that presence of some form of geological oxidation significantly affects the flotation performance. The average gold recovery is 72%, with a relatively high standard deviation of 22% recovery.

Where there was insufficient sulphur content in the test sample to generate a 7% concentrate, gold recovery was determined at the 15% mass-to-concentrate point instead.

Flotation Circuit Scale-up and Modelling for Feasibility Design

The conventional approach to designing flotation circuits focuses on the use of scale-up factors from bench-scale testwork to determine the residence time required for a full-scale plant. For Donlin, a modelling approach was adopted.

Two flotation simulators were tested. One is JKSimFloat developed by the AMIRA P9 Project, Australia, and the second is FLEET (Flotation Economic Evaluation Tool) developed by Minnovex (now SGS).

JKSimFloat was the simulator used for Donlin flotation circuit design because of its robustness based on rigorous validation at various operations, including those with refractory and PGM ore types relevant to Donlin. The JKSimFloat simulations predicted the Donlin pilot-plant results fairly accurately, especially with the addition of a cleaning circuit in the secondary rougher circuit.

Simulated results for a 59,000 stpd (53,500 t/d) flotation circuit using the floatability parameters suggested a gold recovery of about 94% at a concentrate grade of 7.2% S for a two-row circuit configuration treating a throughput with head grade of 1.12% S and 2.37 g/t of gold. Flotation cell sizes of 300 m^3 were selected based on lip loading data.

13.1.7 Pressure Oxidation

Chemistry of Pressure Oxidation and Hot Cure

Pressure oxidation 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 leaching by cyanidation.

Batch Autoclave Testing (Dynatec 2004)

During 2004, Dynatec carried out bench-scale autoclave testing of four composite samples, ACMA Intrusive, ACMA Sediment, Lewis Intrusive, and Lewis Sediment. The scope of the test program 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.

The concentrates were relatively fine, with P_{80} levels of 33 to 41 µm (82% to 89% minus 44 µm), and were tested without further size reduction for comparison purposes. Direct CIL cyanide leaching of the unoxidized feeds yielded gold extractions between 3% (ACMA Sedimentary) and 11% (Lewis Intrusive).

Higher autoclave oxidation kinetics were observed at 210°C and 220°C than at 200°C. Gold extractions were highest from the solids oxidized at 220°C. All subsequent pressure oxidation testwork on all four concentrates were, therefore, conducted at 220°C.

As shown in Figure 13-7, the sulphide sulphur oxidation kinetics was rapid, with more than 98% oxidation achieved within 30 minutes.

Gold extractions from the oxidized concentrates, as shown in Figure 13-8, were correspondingly high after 30 minutes of pressure oxidation and improved marginally to their maximum values of between 95.1% (ACMA Sedimentary) and 98.5% (ACMA Intrusive) after 45 minutes of oxidation. With extended pressure oxidation time, however, the gold extractions declined, most markedly for the sedimentary concentrates, which had relatively high organic carbon content.

A retention time of 45 minutes was selected for the pressure oxidation in the subsequent material balance and locked-cycle testwork.

In the locked-cycle testwork, the pressure oxidation tests were conducted at pulp densities approaching those anticipated for the discharge slurries in commercial autoclave operation. The extents of sulphide sulphur oxidation with the 45 minute pressure oxidation retention time at 220°C exceeded 98%, with more than half over 99%. There was no systematic change in the oxidation extent with increasing cycle number, indicating that the recycled solution did not affect the sulphide oxidation.

Figure 13-7: Sulphide Oxidation Pressure Oxidation Kinetics at 220°C

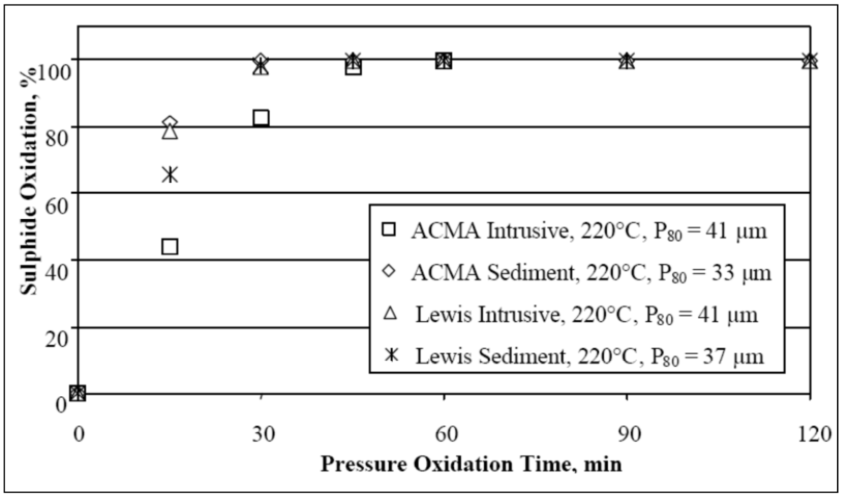
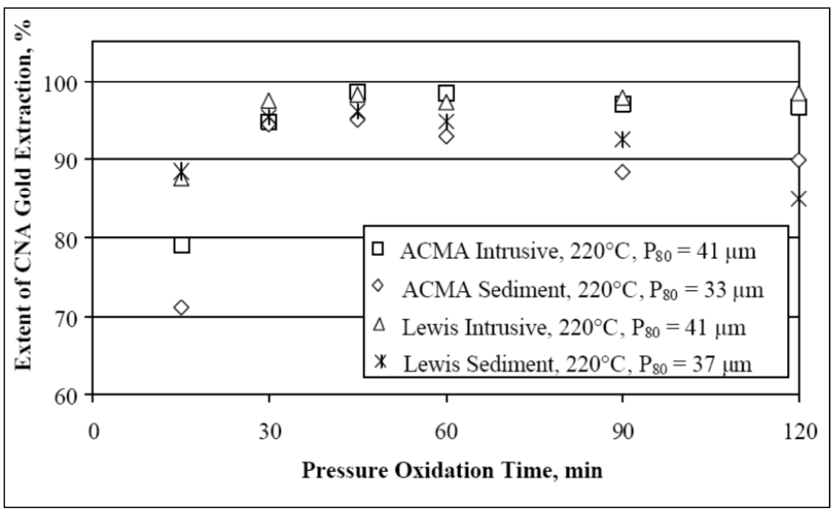


Figure 13-8: Gold Recovery Profiles from Pressure Oxidation at 220°C



Analysis of selected solution samples for gold and silver indicated that these were below their respective detection limits of 0.2 mg/L and 1 mg/L in the pressure oxidation discharge solutions.

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 strongly affected by the decision to pre-acidify the concentrate sample charge, or not, both at pilot scale and bench scale.

Batch Autoclave Testing (Barrick Technology Centre 2006)

A summary of the various tests undertaken on composite samples in 2006, indicating the key conclusions, is provided in the following subsections. This discussion is mainly limited to gold recovery performance.

Baseline Tests

A series of batch tests was conducted to understand the potential impact of autoclave temperature, oxidation time, pre-acidification, oxygen concentration, and thiocyanate concentration on autoclave performance. .

The bench autoclave test (BTAC) results for the various concentrate samples at a temperature of 220°C. Recoveries in the order of ~95% to 99% were achieved at the 45 minute pressure oxidation time for all ores tested.

The results indicate that oxidation rate increases with POX temperature and that of the temperatures tested, the optimum in terms of gold recovery profile is 230°C, being marginally better in performance than at 220°C. No definitive explanation can be provided to explain the unusual decrease in performance of the 240°C test result.

In conclusion, autoclave temperatures of more than 192°C are required to achieve >92% gold recovery within a nominal 1 hour autoclave residence time, and temperatures of 220°C to 230°C provide maximum recovery values.

Some preliminary BTAC 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

From the autoclave test data, it was noted that gold recovery improved with increasing POX residence time. Therefore, experiments were undertaken to test the potential application of higher-temperature POX (230°C to 240°C). It was seen that gold recovery does improve with extended autoclave time and that this improvement is indeed accelerated at 240°C.

BTAC Testing on 2007 Phase 1 Composite Concentrate Sample

During January 2007, the Barrick Technology Centre carried out a series of BTAC tests 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. The best result was at 220°C with 45 minutes of residence time.

Pilot-Plant Testing (Barrick Technology Centre 2006)

Four phases of autoclaving pilot tests were carried out during the latter half of 2006. In discussing the gold recovery aspects of the pilot-plant results, all testwork phases undertaken in 2006 are considered as one, for clarity.

Pilot Campaigns at 220°C and 225°C

A series of 220°C and one 225°C pilot campaigns were also carried out during the latter part of 2006. The three tests undertaken that generated final CIL gold recoveries of less than 95% were due to oxidation rate performance issues. Based on earlier results, 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.

Pilot Campaigns at 240°C

Previous batch tests carried out at 240°C indicated the potential for operating at this temperature and achieving high gold recovery. As for the 240°C test runs, high gold recoveries were achieved quickly, albeit with faster oxidation kinetics than at 220°C. An improvement in recovery is evident, but the level of improvement is insufficient to achieve high gold recoveries under practical design constraints.

Pilot-Plant Testing (2007 Phase 1)

During February 2007, an additional autoclave pilot campaign was undertaken at the Barrick Technology Centre (BTC) to verify the autoclave design parameters and potential gold recovery results for the 2007 FS. This pilot run attempted to explore the three different operating temperatures, 200°C, 210°C, and 220°C. Based on 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 unit operation were purposely set to be higher than design to try to achieve more fully oxidized conditions at the discharge than had been realized in previous campaigns, and to provide a set of data covering a large range of operating residence times. This coverage of residence time is possible by sampling each of the pilot-plant compartments and undertaking CIL tests on each sample.

A maximum gold recovery of ~96.9% was reached in compartments 3 and 4 for the 225°C test run, representing a residence time of 50 to 55 minutes. For the 220°C run, a maximum recovery of 95% to 96% was achieved in compartment 5 at a residence time of ~55 minutes. It can be seen that CIL gold recovery drops significantly towards the discharge of the autoclave, after peaking at 96.0% to 96.4% near compartments 3 and 4. As the operating temperature of the autoclave is lowered; the required residence time for reaching maximum gold recovery increases.

Pilot vs. BTAC Testing

It is well known that the BTAC test has inherent limitations in its ability to replicate a continuous autoclave operation. The continuous introduction of fresh feed into a continuously operating facility, the continuous transfer of new feed to each stage (residence time distribution), and the need to provide the BTAC unit with acid solution in the feed to allow the oxidation reaction to "kick-start" all result in some important differences in the chemistry, and the timing of that chemistry occurring, between a BTAC test and a pilot test.

Pilot testing is considered the more appropriate test method to replicate the chemistry and kinetics of a production-scale autoclave and is regarded as best practice for definitive autoclave testing.

Summary

The main results from 2007 Phase 1 pilot testing indicated that:

- 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 either too short or 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 to 55 minutes.

Considering 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 into Phase 2.

Pilot-Plant Testing (2007 Phase 2)

During June 2007, Phase 2 pilot autoclave testing was carried out at the Barrick Technology Centre. The aim of the Phase 2 testwork was several-fold:

- To operate the pilot unit more closely to the design optimum autoclave residence time as determined from Phase 1
- To demonstrate that target CIL gold recoveries can be achieved on final products from the autoclave, not just on compartmental sub-samples
- To confirm potential downstream CIL gold recoveries
- To confirm the selected design criteria for the pressure oxidation circuit for the 2007 FS
- To provide more autoclave profile data for oxidation rates and gold recoveries.

The concentrate feed sample was obtained from material generated from the SGS Lakefield pilot flotation test program undertaken in January 2007.

Pre-Acidification

Concentrate pre-acidification reduced the carbonate content (inorganic carbon) of the concentrate reduced from 0.35% to 0.05%, representing approximate 86% dissolution of the contained carbonates.

Sulphide Oxidation Performance

The pilot run investigated two different operating temperatures, 220°C and 225°C.

For test runs W and X, the oxidation rates were reasonably fast and consistent, and that measurable sulphide oxidation was essentially completed by 37 to 42 minutes' residence time.

Autoclave Discharge Gold Recovery Performance

Before the first set of discharge samples was collected, the autoclave was operated for approximately two hours, or approximately 2.5 turnovers, to allow it to come to near steady-state before sampling commenced. The autoclave then continued to operate for another 4 to 5 hours, with regular sampling.

The pilot autoclave operated at periods of high gold recovery, 96.1% to 97.3%, at times, but then shifted and operated at 93% to 94%, seemingly moving from one "recovery regime" to the other very quickly. It was found that if the autoclave residence time is permitted to exceed 50 minutes, then CIL gold recovery of the AC discharge drops to 93% to 94%. Similarly, if autoclave residence time is too short, then recovery is also lower than optimum at 96% to 97%. Optimum gold recovery, exceeding 97%, is recorded at around 45 to 49 minutes.

Autoclave Profiles Recovery Performance

During the operation of pilot campaign runs W and X, four profile samples were collected through the autoclave, approximately every 30 minutes. The same trend of the effect of residence time on CIL gold recovery as seen for the discharge samples is clearly evident. Based on the results from these samples, the optimum operating residence time is confirmed in the range 37 to 47 minutes. Optimum gold recovery from the autoclave appears to correspond with the "just completed" extent of sulphide sulphur oxidation. Excess oxidation after that point is detrimental to gold recovery.

Carbon Dissolution in Autoclave

It was seen that the organic carbon content (by assay) has decreased from 0.78% to an average of 0.57%, meaning that ~25% of the organic carbon has oxidized within the autoclave. Further, the graphitic carbon (by assay) shows negligible reduction in assay grade in the autoclave. Inorganic carbon in the feed is reduced to below assay detection limits by the pre-acidification process ahead of the autoclave, and so no apparent trends are discernible.

Autoclave Vent Gas Sampling

During the Phase 2 pilot autoclave run, standard gas testing was performed on the main vent gases from the autoclave, ahead of the water scrubber, with the purpose of quantifying the oxygen, carbon dioxide (CO_2), carbon monoxide (CO), and total hydrocarbon (THC) generation rates from the autoclave that would be fed to the gas scrubbing system.

Hot Cure

An additional testwork series on hot curing optimization was undertaken on the Phase 2 pilot autoclave products to confirm the proposed hot cure section design for the 2007 FS and FSU1, and to provide the latest information for future optimization.

The analyses of the hot cured solids and liquor components show that sulphur in the solids is indeed dissolving, with a corresponding decrease in solids mass and liquor sulphuric acid concentration and a consequential increase in liquor iron content. It can be seen that a decrease in lime consumption is required for pH adjustment to 11 and that CIL gold recovery improves slightly at the six-hour residence time point. The FSU2 hot cure circuit is designed with a residence time of six hours. Lime consumption of well-washed hot cure solids to pH 9 was relatively constant at around 3 to 4 kg/t of hydrated lime.

Hot Cured Product Counter-Current Decant Washing

A large sample of autoclave hot-cured product was collected from the Phase 2 pilot run and subjected to a series of washing tests followed by neutralization to pH 7, 9, and 11. Significant reductions in lime consumption occur up to 98% washing efficiency, the feasibility design target, and the addition of magnesium sulphate substantially increases lime consumption from 8 to 22 kg/t when adjusting pH to 11 but does not greatly affect lime consumption for adjustment to pH ~8.8 to 9.0; in this case, lime consumption increases from ~5.7 to 6.5 kg/t. Note that lime consumption is specified as kilograms of $Ca(OH)_2$.

The benefit of operating a CIL circuit at pH ~9, in terms of lime consumption, is also demonstrated from this work.

Summary

The main results of the 2007 phase 2 pilot autoclave testing program were as follows:

- Product CIL gold recoveries of 96.6% can readily be achieved and recoveries of 97% are possible under optimum operating conditions, as indicated by the tests on the discharge samples as well as the autoclave profile samples.
- CIL gold recovery 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 because of incomplete sulphide sulphur oxidation. Recovery can also be lower than optimum if autoclave residence time is too long and oxidation is excessive.
- Autoclave operating temperatures of 220°C and 225°C provided good results with optimum residence times of 45 to 49 minutes for CIL gold recovery, based on the autoclave discharge samples.
- Measurable sulphide sulphur oxidation is essentially completed by 37 to 42 minutes residence time, as indicated by analysis of the autoclave profile samples.

- The selected hot curing time of six hours as per the feasibility design provides good lime consumption results and CIL gold recovery performance, but dissolution of arsenic is evident, subsequently requiring precipitation in the following neutralization stage.

Autoclave Mercury Emission Testwork

During a pilot run, two gas streams were sampled for mercury content in the flash pot vent stream and the autoclave vent stream.

The amounts of mercury vented through the flash system and the autoclave vent were 0.21% and 0.026%, respectively, of the calculated mercury head in the concentrate.

The results from a second series of emissions testing indicated the presence of very little mercury emission in the combined gas streams (0.003% of feed mercury content), with the gas and scrubber mercury contents being close to assay detection limit.

Despite the low measurements, a mercury abatement system has been designed to comply with the December 2010 US EPA National Emissions Standard for Hazardous Air Pollutants for gold ore processing and production facilities.

13.1.8 Neutralization

Introduction

The oxidation of pyrite and other naturally occurring sulphides generates sulphuric acid and other metal sulphates within the autoclave and hot curing circuits. These species need to be neutralized and precipitated into a stable form to ensure that the final tails from the plant have a low soluble metals content, and is also at approximately neutral pH.

Typically limestone (calcium carbonate) and hydrated lime (calcium hydroxide) are utilized for this neutralization duty. Limestone is used for the first part of the pH adjustment while the final pH adjustment to 7 (for acidic liquor neutralization) and 9-11 (for CIL feed) is carried out with hydrated lime as $Ca(OH)_2$, which is a more reactive neutralizing reagent.

Metallurgical studies were undertaken to determine the potential neutralization capacity of the flotation tails stream and also of an identified local natural source of low-grade carbonates known as calcareous sandstone (CSS). This local material was composed of ferroan dolomite, ankerite, calcite and siderite in decreasing levels.

The orebody itself has a relatively high carbonate content of 2.33% reported as CO_2 (or 3.18% reported as CO_3) over the life of mine compared to a sulphur content of 1.13% S. This represents a stoichiometric ratio of carbonate to sulphur of 1.49. Therefore, there is sufficient, or rather, excess alkali in the ore available to neutralize the sulphates generated from the oxidation of all the contained sulphur. However, this requires effective utilization (reactivity) of the measured neutralization content of the ore for the excess to be valid.

It has been shown through the testwork carried out, that with the provision of sufficient neutralization residence time, and the elevation of the slurry temperature in neutralization, good utilization of the contained carbonates in the ore is possible, resulting in the minimal amounts of lime needing to be consumed by the plant.

Further, it has been determined that the use of CSS is not required for neutralization of the acidic autoclave acid, due to effective use of the ore itself, as flotation tails, and at high pH (>5.0), CSS does not compete economically with imported lime. The local CSS resource instead serves as a back-up alkali source for the Donlin Gold Project if, and as, required.

Dynatec – November 2004

During 2004 Dynatec tested the neutralization properties of the acidic liquors generated from bench autoclave tests, using flotation tails, limestone, and lime. The neutralization tests conducted with limestone and lime performed well, and at pH 8.0 all metals, with the exception of Mn and Mg, precipitated virtually completely from solution at generally below detection limit grade. The results however were not favourable from a lime consumption perspective.

Placer Dome Technical Services 2005

During late 2005 and into 2006, Placer Dome Technical Services (PDTS) investigated the neutralization capacities of CSS and flotation tails. The results show that CSS with high CaO/MgO ratios provided higher carbonate utilization compared to low ratio composites. The grind size did not have a large effect on the neutralization capacity of the CSS composites.

Testwork on flotation tailings showed higher carbonate usage from the intrusive materials compared to the sedimentary materials. Overall though, the utilization of the carbonate within the tailings was very low.

Preliminary Batch Neutralization Testing, 2006

A limited batch neutralization testwork program was initiated in mid-2006, aiming to improve quantification of the neutralization options for Donlin. The results indicated that the flotation tails / lime neutralization option, with extended neutralization residence time, was the most economic with the lowest total cost.

Pilot Neutralization 2006

During October 2006 a pilot neutralization testwork program was undertaken. The acid solution used for the pilot test was produced during the pressure oxidation pilot run conducted on 26 Sept 2006, with the autoclave operating at 220°C followed by hot curing of the slurry for at least 12 hours at 95°C.

Flotation tailings were used as produced from G&T Metallurgical being blended in the ratio of 25% ACMA intrusive, 50% Lewis Intrusive and 25% Sedimentary.

Profile samples were routinely collected from the pilot-plant run, and then lime added to achieve a final pH of 7.0. At the given ratio of flotation tailings to concentrate (5.135 kg tailings per kg concentrate), neutralization of the dilute acidic pressure oxidation solution using flotation tailings with a carbonate grade of about 1.8% CO_2 reached a pH of 4.0 to 4.5 after 12 hours. There was no significant increase in pH at longer retention times. Under the conditions tested, lime consumption after flotation tailings neutralization was approximately 4.5 g quicklime per litre of dilute acid solution, or 24 g of quicklime per kilogram of concentrate. It was also possible to use CSS at the rate of 1.6 kg of CSS per kg of concentrate.

Neutralization Phase 1 Bench Testing, 2007

The acidic solution used for this testwork was generated from Donlin concentrate during the continuous pressure oxidation (POX) pilot run on 6 February 2007. The concentrate was pre-acidified to pH of about two with sulphuric acid prior to the oxidation process. The campaign run was at 225°C with a 70 minutes retention time. The discharge slurry was then hot cured for about 24 h immediately following the POX process.

The hot cure slurry was washed with gypsum saturated water in a pilot counter-current decant (CCD) circuit at a ratio of 2:1. The overflow acidic filtrate was collected and used for batch and continuous tests. Initially, the pH of the diluted POX solution was approximately 1.0.

The flotation tailings used was a blend from a flotation piloting campaign in December 2006 obtained from SGS Lakefield. The flotation tails used had a carbonate grade of 2.0% CO_3 and was sourced from the same pilot float feed sample that provided the concentrate used to generate the acidic liquor, via the pilot autoclave.

Bench tests at varying temperatures were undertaken to determine the effect of temperature on neutralization rate and utilization. It was seen that a significant improvement in performance occurs as temperature increases. Temperatures of 55°C or greater allowed a pH of 6.0 to 6.5 to be reached, consequently resulting in the reduction of lime consumption to about 1 to 2 g/L of diluted acidic liquor.

Neutralization Phase 1 Pilot Testing, 2007

Based on the bench tests, a pilot neutralization campaign was initiated in early 2007, using 70°C and two residence time selections of eight and 12 hours, and using the same acidic liquor and flotation tails as used for the bench testing.

The average pH profiles of both the eight and 12-hour residence time test campaigns are shown in Figure 13-9. It can be seen that final pH levels of greater than 6.0 were reached with the flotation tails, for both the eight hour and 12 hour runs.

Figure 13-10 shows that lime consumption tests of profile samples taken from the pilot-plant runs. It can be seen that lime addition required is less than 1 g/L of diluted acidic liquor, at the end of the neutralization circuit.

Neutralization Phase 2 Pilot Testing, 2007

With the undertaking of the Phase 2 pilot autoclave test program in June 2007, an opportunity arose to confirm the expected performance from the final feasibility neutralization circuit using a suitably designed pilot-plant set-up. In addition, this provided an opportunity to test the neutralization performance of the MCF2 pilot flotation tails. The design of the neutralization pilot circuit closely followed that of the actual feasibility circuit design, with the following key parameters:

- Four float tails neutralization tanks
- One lime neutralization tank
- Six hours' residence time ~ optimization based on testwork
- Operating temperature of 55°C ~ based on heat balance
- Addition of CIL tails into tank one

Figure 13-9: 2007 Phase 1 Neutralization Pilot pH Profiles

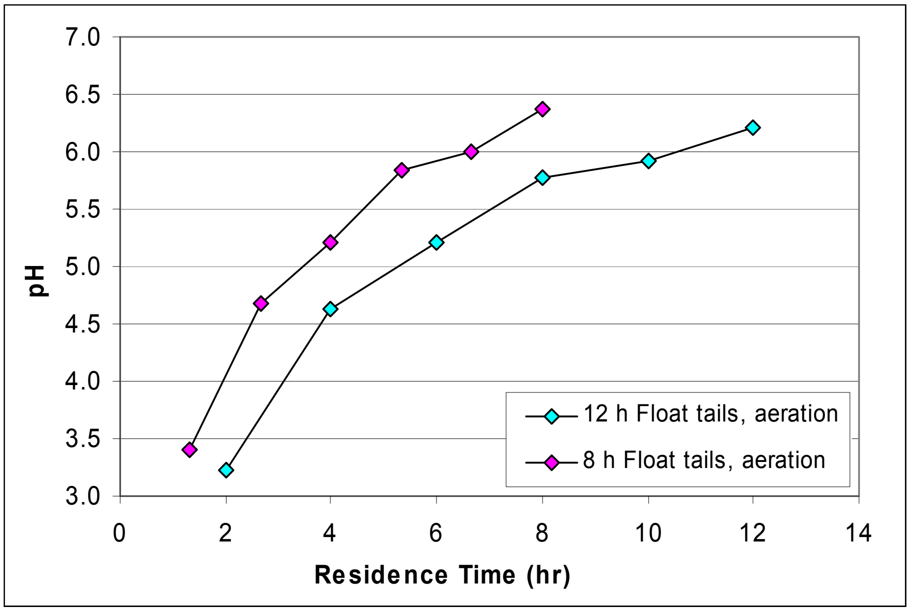
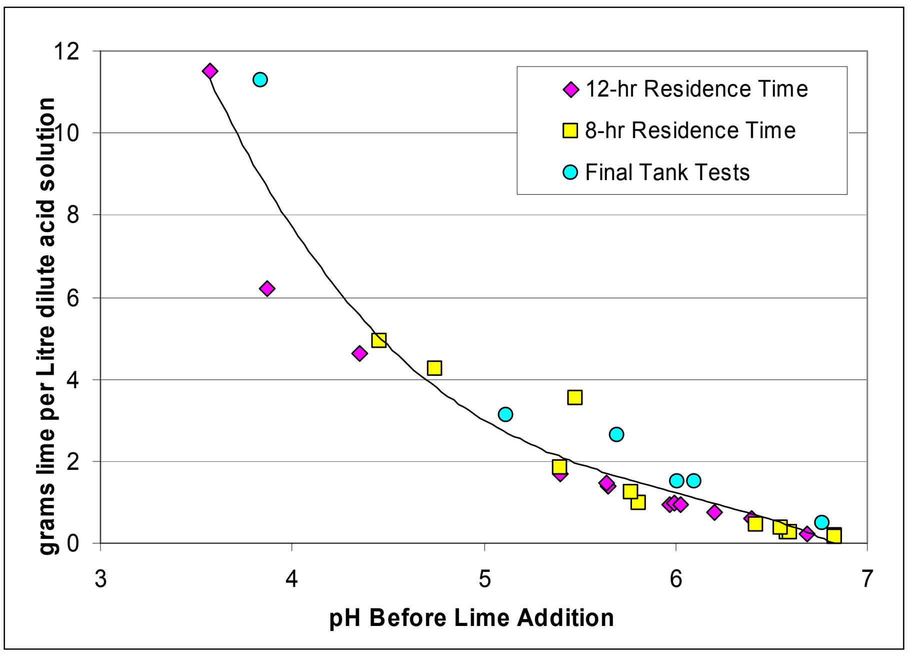


Figure 13-10: Lime Demand Test Results of 2007 Phase 1 Pilot Samples, Plotted against Initial pH



- Mixing ratios of diluted acidic liquor and flotation as per the feasibility mass balance.

In addition to the six-hour residence time run, a second three-hour residence time campaign was undertaken to investigate the potential to reduce the size of the circuit for the detailed design phase.

The source of the dilute acidic liquor was the overflow stream from the operation of the pilot CCD plant, washing Run W hot cured product. To be conservative, no calcium carbonate was added to this acidic liquor to correct for the pre-acidification process.

The source of the flotation tails was SGS Lakefield flotation pilot PP-11, which incorporated the MCF2 grinding/flotation flowsheet. The sample used for the pilot test had a carbonate grade of 1.65% (as CO_2), compared to the LOM predicted carbonate grade of 2.33% (as CO_2).

The average pH profiles for the two test campaigns are shown in Figure 13-11. It can be seen that with the six-hour test campaign (i.e., matching the feasibility circuit design), a final pH of 6.75 was reached prior to the lime addition step. With the reduction to three hours' residence time, the final pH was 6.50.

Figure 13-12 shows the results of the lime demand tests undertaken on profiles from the pilot plant. Lime consumption at the end of the neutralization circuit is less than 0.2 g/L of diluted autoclave acid solution, noting that wash water flows have been increased in the feasibility design (more dilution of the acidic liquor) to improve washing efficiency of the autoclave discharge hot cure product.

The results of the three-hour test campaign are encouraging, suggesting the potential to decrease the size of the neutralization circuit further. However, for the purposes of managing the potential variability of carbonate content in the ore, and also to provide time for the operations personnel to respond to unplanned grinding or flotation circuit shutdowns, the longer six-hour residence time circuit continues to be the recommended design.

Figure 13-11:2007 Phase 2 Neutralization Pilot pH Profiles

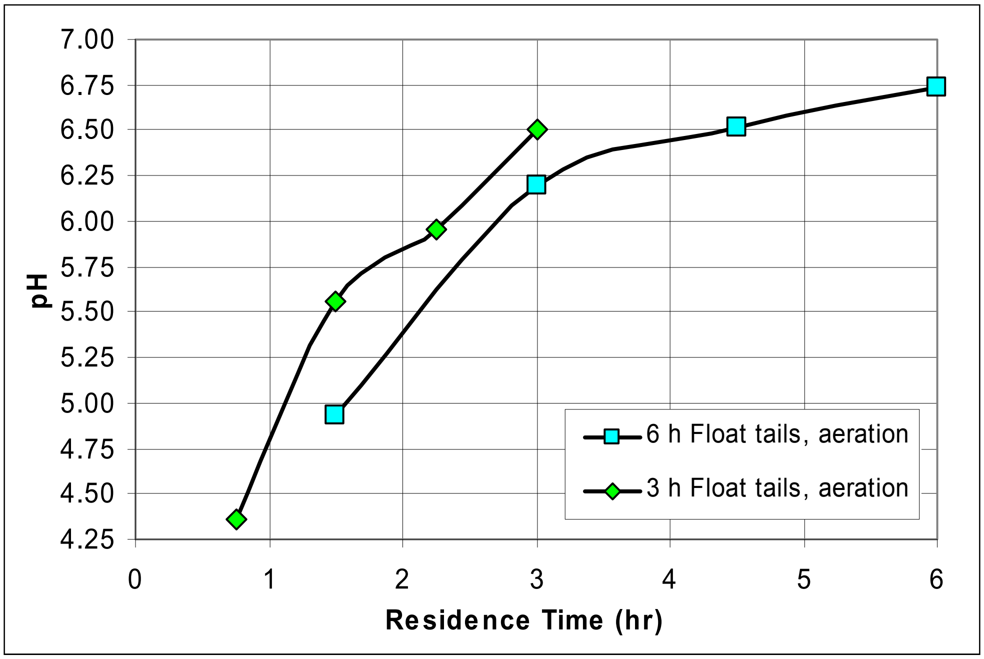
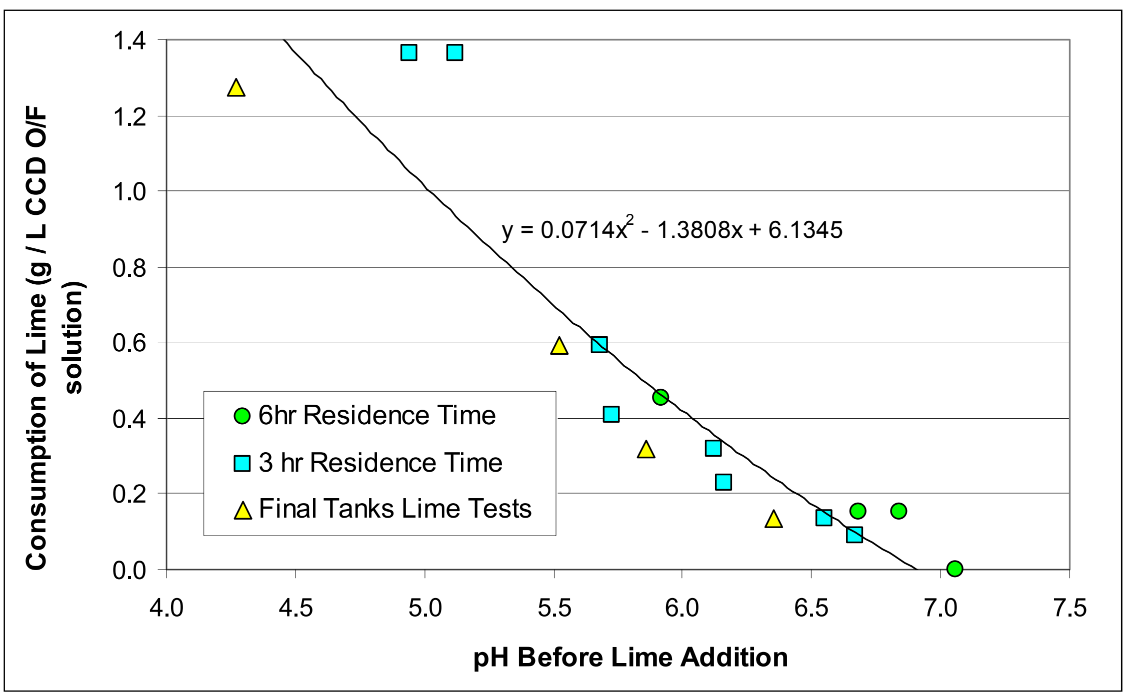


Figure 13-12: Lime Demand Test Results of 2007 Phase 2 Pilot Samples, Plotted against Initial pH



Neutralization Variability Testing

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

SGS Lakefield generated a synthetic dilute autoclave acid based on the prediction of key species content from the near-final MetSim model's prediction of that stream. Flotation tails samples from the recently completed flotation variability program were used as the source of neutralizing solids.

Figure 13-13 is a summary chart of the results of the tests completed.

Prediction of lime consumption for acidic liquor neutralization for the FSU2 operating cost estimate is based upon the relationship between lime demand and flotation feed carbonate grade developed from this test data.

13.1.9 Carbon-in-Leach (CIL)

The following subsections summarize the processes of cyanidation and carbon adsorption, and then describe the results of the various testwork programs completed on the Donlin ores.

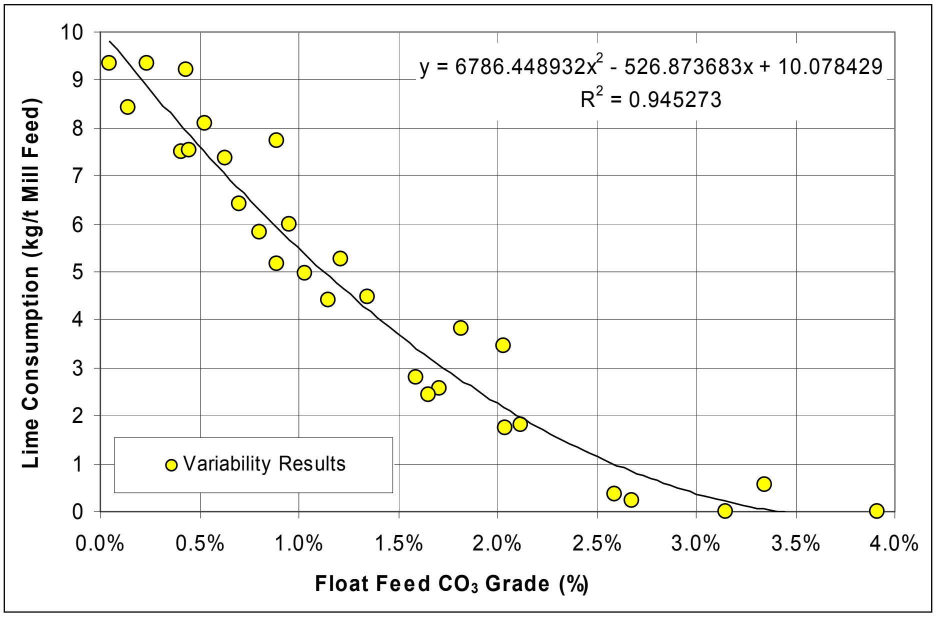
Testwork Introduction

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

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

The bulk of the cyanidation tests carried out to date have largely been on autoclave compartmental and discharge samples, where large numbers of relatively small samples are leached with high concentrations of carbon and cyanide. This is diagnostic tool that enables the performance of various autoclave tests to be established without the added complication of the constraints that could be imposed by attempting to optimize leaching kinetics.

Figure 13-13: Plot of Neutralization Variability Testing Lime Demand Results at 6 Hours' Residence Time



CIL gold recovery has generally shown to be more sensitive to the autoclave operating conditions (residence time, temperature) than to the operating conditions and methods applied in the CIL circuit. The target of the metallurgical design of the CIL circuit is rather 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 Metallurgical Aspects of Planned Donlin CIL Circuit

The key metallurgical aspects of the final selected CIL circuit design are summarized as an introduction to the detailed discussions of the specific testwork results presented subsequently. The aim of the testwork programs was to define the operating characteristics and circuit design of the CIL circuit for treatment of CCD-washed autoclave product at Donlin.

Leach Circuit pH

The MetSim modelling and metallurgical testing have shown that the Donlin CIL circuit could operate well at a relatively low pH of 9.

Increasing CIL pH above 9 in the CIL circuit will consume additional lime through the precipitation of magnesium sulphate in solution to magnesium hydroxide. The lime then re-dissolves back into solution when mixed back into the tailings and returns to the plant through tailings water recycle. To achieve the traditional CIL circuit pH levels of 10 to 11, all of the magnesium in the feed solution would need to precipitate completely. In the FSU2, reclaim water used in CIL is treated to remove magnesium and enable operation at conventional pH in CIL.

Assuming a CIL pH of ~9, lime addition is estimated to be in the order of 5 to 7 kg/t of concentrate. The key component affecting both is the washing efficiency of the autoclave product CCD wash circuit. To maximize washing efficiency, a four-stage CCD circuit with a high wash ratio of 4:1 is used.

CIL of Flotation Tails

During late 2006, BTC undertook CIL tests on the flotation tails from the G&T pilot-plant campaigns. Table 13-13 summarizes the results. Based on the low head grade and low gold recovery, it is not economically viable to leach the flotation tails.

CIL Optimization Testwork – 2006

A composite of autoclave discharge material from the 2006 autoclave pilot test program was collected and leached under varying conditions. The gold recoveries achieved from this work were limited due to the nature of the product from the particular pilot run. The following conclusions are drawn from the CIL optimization work undertaken on the pilot autoclave product:

- Cyanidation in the absence of carbon is detrimental to final leach gold recovery.
- Using higher carbon loadings (pre-loaded with gold) does not adversely affect gold recovery.
- Leaching at pH 9.2 does not negatively affect CIL gold recovery.
- Increasing the wash efficiency of the CIL feed slurry can significantly reduce reagents consumption rates.
- Leaching at too high a slurry density can negatively affect CIL gold recovery through unsuitable rheological properties of the slurry.

Samples of the detoxified CIL products were sent to the University of British Columbia for rheological testing, where it was seen that the slurry solids content (% solids) had a significant impact on the viscosity of the slurry. Densities above 35% are considered potentially problematic.

Table 13-13: CIL Results from Pilot Flotation Tails

Ore Type	Head Grade (Au g/t)	NaCN Consumption (kg/t)	Lime Consumption (kg/t)	Float Tails Gold Recovery (%)
Sediment	0.75	0.88	0.7	16.0
Lewis Intrusive	0.51	0.56	0.6	33.3
ACMA Intrusive	0.47	0.61	0.7	27.7
Blend	0.59	0.79	0.7	30.5

CIL Pilot-Plant Testing 2007

In early 2007, the Barrick Technology Centre carried out a pilot CIL test run using CCD-washed pilot autoclave product from the 2007 Phase 1 autoclave pilot testwork program.

Gold and Silver Recoveries

The carbon in the pilot CIL circuit was successfully loaded up to 4,000 g/t gold, and two carbon transfers were undertaken.

A gold balance for the period of the pilot-plant run returned a calculated gold recovery of 93.6% with a tailings grade of 1.39 g/t. At the tank 5 position, ahead of the adverse influence of the higher carbon loadings in tanks 6 and 7, the solids assay was lower, at 1.2 to 1.3 g/t, representing a gold recovery of ~94.0% to 94.5%.

A bottle roll test of the CIL feed conducted at pH 11 yielded a comparative gold recovery of 93.9%, indicating that the pilot operation provides equivalent recovery performance to that achieved in a batch bottle roll test, even with the pilot plant operating at a relatively low pH of 9 and using profile-loaded carbon.

Silver recovery from the pilot plant was low, at 29.7%, which is typical of the leaching characteristics of silver from autoclave solids product due to its dissolution and subsequent precipitation as cyanide insoluble Ag-jarosite within the autoclave.

Leaching residence time of 20 to 24 hours over six tanks is an appropriate design for the Donlin continuous CIL circuit.

Cyanide Consumption and Addition

The Donlin CIL feed is relatively free of cyanide consumers. This is characteristic of concentrate that has been subject to pressure oxidation, where sulphur is oxidized completely to sulphate and base metals are dissolved into solution, and then CCD washing, which removes the dissolved metals from the autoclave product ahead of CIL.

Cyanide addition to the pilot circuit was 1.5 to 1.6 kg/t, with a consumption of 1.1 to 1.3 kg/t. Most of the consumption is likely to be from losses through HCN from the pilot CIL circuit, rather than consumption by species within the ore. HCN losses of this magnitude will not be experienced at full scale, and the HCN that evolves will be recovered via the ventilation and scrubbing system and returned to the CIL circuit feed. Cyanide addition to the pH 11 bottle roll tests on the pilot-plant feed was 1.2 kg/t, with a consumption of 0.05 kg/t.

Assuming a CIL pH of ~9, cyanide addition of 0.7 to 0.9 kg/t of concentrate is estimated. Consumption of cyanide will be lower at a higher CIL pH of 11.

CIL Optimization Testwork – 2007

A series of bench-scale tests was undertaken on products from the 2007 Phase 2 Pilot autoclave test program to attempt to optimize the CIL process.

Cyanide Addition Optimization

A series of 24-hour bottle roll CIL tests were conducted at varying cyanide concentrations to determine the relationship between cyanide addition rate and gold recovery. Due to the limitations associated with undertaking low pH CIL tests at laboratory scale, a higher pH of 11 was used. CIL gold recovery was found to be not significantly affected at low cyanide concentration, and that the process is more economically favourable at low CIL cyanide concentration levels.

Rheology

Additional rheology testing was undertaken on a detoxified CIL product from the 2007 Phase 2 pilot-plant work. Beyond a level of 35% solids the viscosity of the material was found to climb rapidly.

Cyanide Detoxification Testwork

Introduction

The SO_2/Air (sulphur dioxide cyanide destruction) process will be used for cyanide detoxification of the Donlin CIL tailings before this stream is transferred into the neutralization circuit.

Testwork Summary – 2006

Cyanide detoxification testwork has been completed on CIL tails slurry generated from the 2006 pilot autoclave test program. Three types of cyanide detoxification methods were tested and found effective:

- Prussian blue (iron sulphate precipitation) – consists of adding autoclave discharge acid to detoxify the cyanide complexes
- AVR (Acidification, Volatilization, Recycle) – consists of acid addition to drive the cyanide off as HCN and capturing the HCN for reuse in the circuit
- SO_2/Air testing – uses a combination of SO_2 and air to detoxify the cyanide.

All three test methods were successful in reducing the cyanide levels to expected permit requirements. The SO_2/Air method was selected over the Prussian blue and AVR methods for cyanide detoxification at Donlin.

For the SO_2/Air testing, the CIL tailings slurry was effectively treated in a single stage operating with approximately 60 minutes of retention and an SO_2 dosage of 4 g/g CNWAD. A pH of 8.5 was used for all tests since acid addition was required to lower pH levels. The addition of $CuSO_4$ at 10 mg/L Cu^{2+} was required for effective removal of cyanide that was present in the feed.

It was found that the content of arsenic in the liquor phase increased after SO_2/Air cyanide detoxification. This solubilized arsenic will be re-precipitated upon mixing the CIL tails into the neutralization circuit as a result of the presence of high levels of dissolved iron in this circuit.

13.1.10 Thickening and Counter-Current Decantation (CCD) Washing

Introduction

The feasibility flowsheet includes the following thickening/solids settling operations:

- 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.

Earlier flowsheets presented for Donlin also included a final tailings thickener to dewater the combined carbon-in-leach (CIL) tailing and neutralization residue prior to discharge to the tailing storage facility. This thickener has been removed from the FSU2 flowsheet.

The flotation tailings, which are a combination of the secondary rougher and the cleaner scavenger tailings, are de-watered before being directed to the pressure oxidation to provide cooling.

13.1.11 Environmental Testwork

To provide samples that are reasonably representative of both the complete metallurgical processes, and also the ore, the testing of combined pilot-plant tailings was selected as the preferred testing method.

The final tailings from Donlin consist of a blend of detoxified CIL tails (cyanide leached autoclave and hot cure product) and neutralized autoclave acidic liquor using the flotation tails stream.

The metallurgical process adopted for Donlin is favourable for the establishment of tailings that are not acid producing as a result of near-complete sulphide sulphur oxidation.

The average mill feed grade is ~1.12% sulphur, with no significant sulphate sulphur present. The mill feed averages 2.51% carbonate as CO_2, which is a molar excess of 37% to the contained sulphur in the mill feed after Year 2, meaning that the ore has excess carbonate content to sulphur content.

Mineralogy undertaken by SGS Lakefield indicates that up to 23% of the sulphate sulphur in the 2006 pilot final tails sample is in the form of jarosite, with 7% in the 2007 Phase 1 pilot-plant final tails and 8% in the 2007 Phase 2 pilot-plant final tails. Modifying the calculated ABA parameters, assuming that jarosite is an acid-forming component of the sulphate, indicates that the tailings will still contain an excess of neutralization capacity.

Pressure oxidation of arsenopyrite in the presence of excess iron is generally considered a best-practice process for generation of stable arsenic precipitates, in forms such as scorodite, for disposal into a tailings storage facility. Promoting the formation of stable precipitates is particularly favoured when molecular ratio of iron to arsenic ratio in the applicable process solutions exceeds 4. Within the plant feed for Donlin, there is sufficient iron to provide the recommended molar ratio of 4 of iron to arsenic. It should be noted that the actual assay grade of iron typically is double the iron content that is accounted for by arsenopyrite and pyrite alone and is more typically at grades of 15,000 to 40,000 ppm.

The cyanide within the CIL circuit dissolves a portion of the mercury in the solids feed to the circuit. A portion of this dissolved mercury in the CIL circuit is adsorbed onto the circuit carbon, and is then recovered from the carbon via stripping and carbon regeneration. However, the capacity of the circuit carbon to completely adsorb the mercury is limited and therefore a component of the soluble mercury remains in the CIL tails solution. This remaining soluble mercury is then blended with the detoxified CIL tails into the neutralization circuit, which then reports to the tailings storage facility.

Reductions in soluble mercury content in recirculating plant waters can be achieved by addition of mercury precipitation reagents, which convert soluble mercury to a stable mercury sulphide product. This is currently practised using the Cherokee Chemical UNR reagent suite at operating mine sites in the U.S.

Based on the testwork completed, it is recommended that the process plant design includes 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. Doing so will eliminate potential build-up of mercury in the process water circuit.

13.2 Recovery Estimates

There are two components to defining the final recovery of gold to bullion from the proposed Donlin processing facility:

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

Due to the refractory nature of the Donlin ores 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, along with the CIL tail residue post cyanide destruction.

The following sections will discuss both these aspects of the definition of final gold recovery from the proposed Donlin processing facility.

Please note for the purposes of this document the terminology MCF2 and MF2 are interchangeable as they pertain to the key equipment within the Donlin flowsheet.

13.2.1 Flotation

The MCF2 flowsheet is assumed to be the basis of the flotation circuit design and for all recovery figures referenced in this section, both as bench testing and pilot testing.

In late 2006, sections of the Donlin deposit were identified as being geologically weathered (altered). While the extent of the geological oxidation is relatively low, there is still potential for significant effects on flotation recovery. Good flotation performance relies on the existence of un-oxidized sulphide surfaces for flotation collector adherence. Minor surface oxidation of sulphide particles can strongly affect recovery of gold to flotation concentrate performance. This is particularly the case for Donlin, where the fine nature of the arsenopyrite mineralization means that there is a potentially greater exposure of the sulphides to minor particle surface oxidation.

In anticipation of this potential issue, flotation pilot and variability testwork was undertaken separately on fresh ores or oxidized (partially) ores. The pilot flotation testing was undertaken exclusively on non-oxidized ores, as this represents the majority of the Donlin orebody (93%). All partially oxidized ores were excluded from the pilot composite sample.

Variability bench flotation testing was then used to define the relative performance of the (partially) oxidized ores so that the overall deposit flotation recovery could be corrected to account for this smaller oxidized component of the orebody.

Sulphide Ore MCF2 Pilot-Plant Results

The MCF2 pilot-plant testwork campaign is recommended as the basis for selecting the overall recovery for flotation. The MCF2 pilot flotation test program simulates the entire FSU MCF2 flotation flowsheet, incorporating the cleaning circuit, cleaner scavenger, and recirculation of the cleaner scavenger tails back to the secondary mill. This scope exceeds the work done in the batch flotation variability tests, which for practicality consist of roughing stages only.

The design of the flotation circuit is directly scaled from, and based upon, the pilot-plant flowsheet, incorporating an identical cleaner and cleaner-scavenger recycle circuit configuration.

The sample selected for the MCF2 pilot campaign was sourced from newly recovered HQ core drilled in 2006 and was compiled as a blend to represent the known LOM composition at the time of compositing based on rock lithology. Notably the sample compositing was not based on geological domain categorization, and known geologically oxidized affected core samples were purposely excluded from this composite sample.

The establishment of gold recovery from the MCF2 pilot program is achieved by means of fitting a linear regression line through all the MCF2 pilot survey results.

No MCF2 test results are excluded from the data set used for the linear regression fit (Figure 13-14), and the 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.

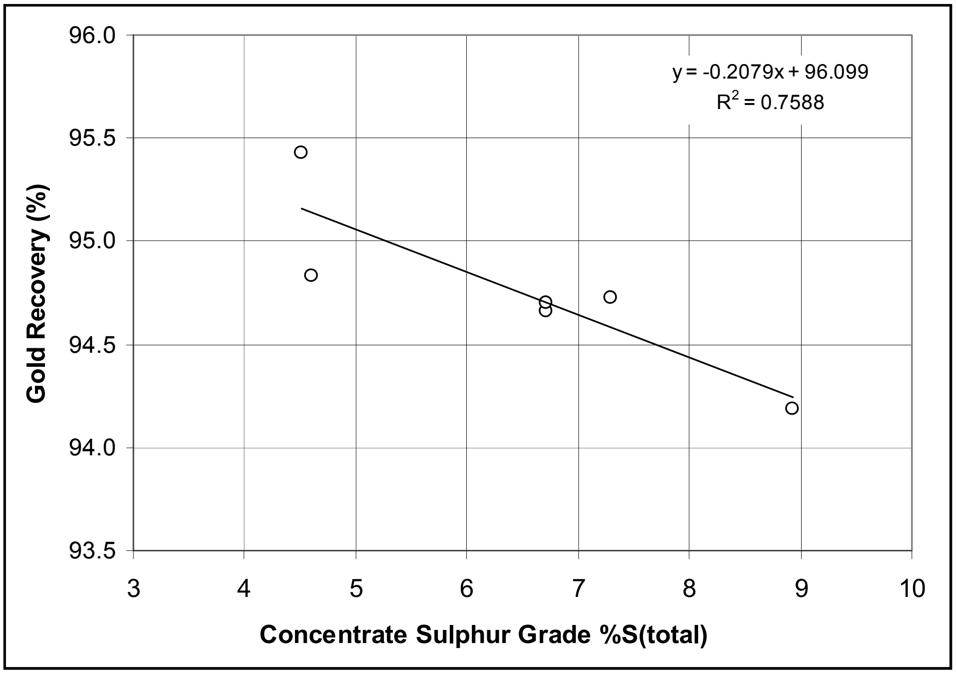
At the design concentrate target of 7% (total) sulphur, based upon linear regression fit to the MCF2 pilot-plant results, gold recovery of the blended composite sample tested is 94.64%. This recovery forms the basis of the flotation gold recovery estimate but must be adjusted to account for effects of geological domain and alteration (oxidation extent).

Variability Testing – Unoxidized Ores

Two different methods of compiling and assessing the non-oxidized/intrusive ore variability flotation recoveries are discussed in this section, being based upon either lithology or geological domain. It is recommended that the intrusive variability samples be grouped on the basis of geological domain, for the following reasons:

- Statistical variance of the variability test flotation recovery results grouped by geological domain is lower than sets of results grouped by lithology. Average statistical variance by lithology is 2.48%R, compared to lithology groupings at 2.64%R.

Figure 13-14:MCF2 Pilot-Plant Campaign Survey Results



- Geological domain assignment is spatially based, so that sample grouping by geological domain would better correct the orebody recovery predictions on a spatial basis to overcome any potential bias through variability sample selection being higher in density in better performing areas of the deposit (i.e., the AMCA deposit area).
- Calculation of overall flotation recovery by geological domain results in a more conservative estimate of the orebody average recovery compared to that based on lithological grouping.

The recommended recoveries per geological domain are summarized in Table 13-14.

Adjusting the MCF2 pilot-plant recovery based on the geological domain variability flotation performance results in a minor upward adjustment of 0.16% for the unaltered ore types. The gold and sulphur head grades of the MCF2 pilot-plant composite are very close to the orebody average.

Table 13-14: Summary of Average Flotation Recovery in Variability Testwork Program, by Geological Domain

Geological Domain	% Tonnes in Orebody (%)	Average Variability Flotation Recovery (%)
AKIVIK	4.6	97.61
400	5.7	96.97
ACMA	16.2	96.45
AURORA	4.2	96.22
VORTEX	11.5	95.19
LEWIS	25.7	94.87
GWK	19.2	91.45
SHL	5.2	89.99
OXIDE	7.7	81.45
Overall	**100.00**	**93.65**

Variability Testing – Oxidized Ores

There is a large variation in flotation test results (i.e., gold recoveries to target concentrate grades) of the ores affected by oxidation. Therefore, to improve the accuracy of the application of the test data to the deposit characteristics, it is recommended that the relationship between sulphur grade and flotation recovery be used as the basis of recovery estimation.

Oxidation Wireframe Model and Mine Block Model

To better clarify the tonnes and grades of oxidation-affected ores within the deposit, a geological wireframe was developed by the Donlin geological team. They then used this model to categorize each mining block as either oxidized-affected ore or non-oxidized ore. This was undertaken on the 6 m x 6 m x 6 m block size, where a block was flagged as oxidized if 50% of the block or greater was located within the oxidation wireframe model.

With the mine blocks defined as such, it was then possible to allocate tonnes and grade of oxidized ore into a mill feed schedule. The oxidized-affected tonnage portion of the deposit is estimated by wireframe modelling to be 7.7.

Mine Model and Stockpile Oxidation Allowance

Ores will be stockpiled and subsequently reclaimed for mill feed occur during the course of the mine life. A sulphur degradation and flotation recovery factor of -5% was applied to reclaimed material that was stockpiled for longer than one year.

Final Flotation Recovery Model Definition

As discussed previously, it is recommended that the adjusted MCF2 pilot-plant performance results be used for the un-altered (un-oxidized) areas of the deposit. The performance of the altered oxides is not changed. Because the pilot-plant sample was originally composited on the basis of lithological domain, rather than geological domain, the results have been adjusted slightly to account for the variation in content between the pilot-plant sample and the latest estimate for the orebody.

The results from the variability testwork program can be used to assign different geological domains within the mine plan to improve the estimation of time-based cash flow from the mine. However, a slight adjustment is again required to match the MCF2 pilot result.

The proportionally adjusted flotation recoveries by geological domain are summarized in Table 13-15; these numbers have been adjusted slightly from FSU1 to reflect the ore release sequence in the FSU2 mine plan. It is recommended that these recovery values be used within the mine plan where the geological domains are separately defined on a period-by-period basis.

No clear relationships between gold, arsenic, or sulphur head grades or in flotation recovery were able to be identified in the variability testwork.

Considering the flotation recoveries of non-oxidized ores by geological domain, of the separately defined oxidized ores, and the impact of stockpiling, the entire flotation circuit can be determined. The overall LOM flotation recovery, based on the ore feed delivery schedule also taking into account feed grade, is calculated to be 93.81%.

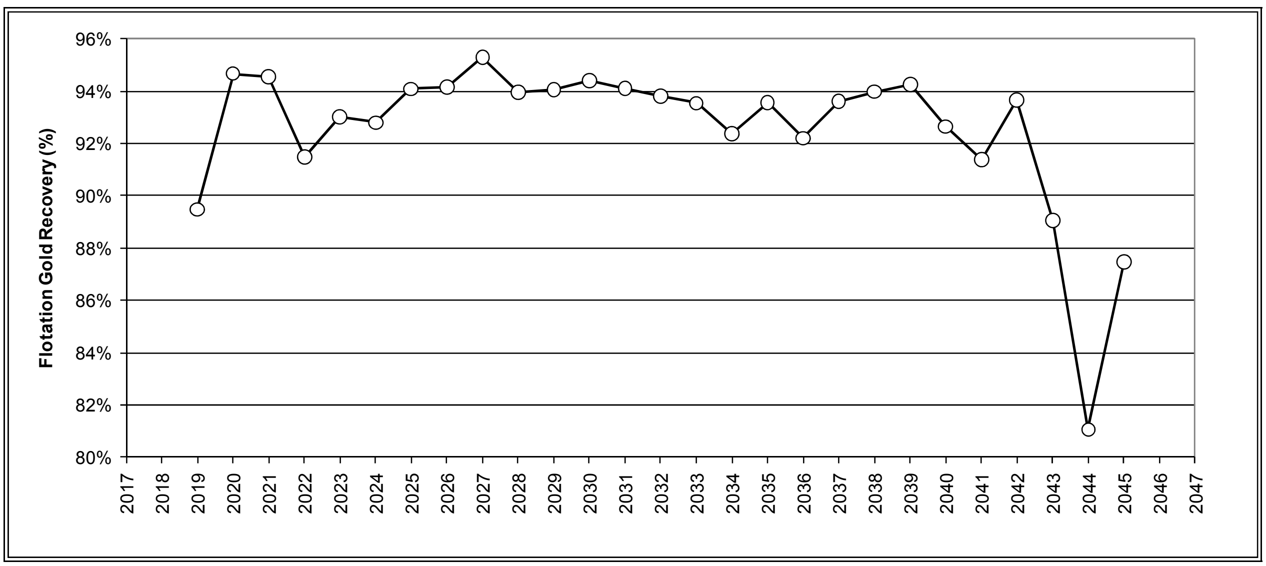
The production plan has been optimized to maximize the feed grade and cash flow in the early years of processing. This has been accomplished through the use of sequential mining of both the Lewis and ACMA pits along with the use of stockpiles and associated ore rehandling,

Figure 13-15 shows the flotation recovery trend across the mine life. Recovery drops in 2043 due to an increase in oxide content in addition to the Lewis intrusive component in the mill feed. Recoveries steadily improve from 2019 through to 2027 due to decreasing oxide content in the mill feed. Flotation recoveries trend downwards from 2028 to the end of the mine life in 2045 due to increased content of oxide and Lewis intrusive material and a decrease in ACMA intrusive content. In 2022 a dip in flotation recovery occurs due to a significant increase in processing of oxidized stockpiled material.

Table 13-15: Summary of Flotation Recovery in Variability Testwork Program by Geological Domain and Adjusted to MCF2 Pilot Result

Geological Domain	% Tonnes in Orebody (%)	Adjusted Recovery to MCF2 Pilot Result (%)
AKIVIK	4.6	97.77
400	5.7	97.13
ACMA	16.2	96.61
AURORA	4.2	96.38
VORTEX	11.5	95.34
LEWIS	25.7	95.03
GWK	19.2	91.61
SHL	5.2	89.99
OXIDE	7.7	81.45
Overall	**100.0**	**93.81**

Figure 13-15: Flotation Recovery Trend throughout Mine Life



13.2.2 Pressure Oxidation

Introduction

Considering the varying nature of test results from the different test methodologies (bench, semi-continuous, or continuous), it is recommended that the continuous testing method be used as the basis of estimating gold recovery performances from the Donlin ores. Continuous pilot testing is considered to best represent a real continuous autoclave operation and is the basis that Hatch has used for evaluating the expected gold recovery from the pressure oxidation/CIL circuit. The Hatch evaluation predicts that an overall 96.6% recovery can be achieved from the POX/CIL circuit under the testwork conditions.

To undertake variability testing on individual ore types, a test method that consumes less sample, such as bench-scale testing (BTAC) or semi-continuous testing (SCAC), must be used.

It is the degree of oxidation, the residence time required to achieve full oxidation, and the ability to control the autoclave oxidation level that influence the chemistry in the test autoclave vessel and therefore the final gold recovery.

Pilot-Plant Testing and Design Considerations

Hatch has reviewed the pilot autoclave testwork completed to date on the Donlin Gold Project and has concluded that, 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. This evaluation assumes that the concentrate sample used for piloting during the 2007 Phase 2 test program is representative of the overall orebody composition and that the conditions of the testwork are maintained in the final design. The result of this review has been carried forward to FSU2.

The deleterious components in the feed to pressure oxidation, based on the historical BTAC testwork, are the sedimentary ores themselves (shale and greywacke rock lithologies).

The proposed autoclave design for Donlin incorporates a level control system on the slurry content of the pressure vessel to permit direct control of operating residence time in the autoclave. Therefore, based on the pilot testwork and the flowsheet design, Hatch recommends that an overall POX and CIL recovery of target of 96.6% be adopted and be applied equally to all ore types fed to the autoclave.

The nature of the orebody within the deposit is such that sedimentary ores will always be blended into the mill feed. The actual lithologies of the intrusives present within the blend may change on a macroscopic basis, but the sedimentary content from greywacke and shale will remain an ongoing part of the blend. The inventory within concentrate storage tanks ahead of the autoclave feed tank will provide a mechanism to smooth short-term variability of sedimentary content.

13.2.3 Overall Plant Gold Recovery

To determine the overall plant recovery, both pressure oxidation and flotation need to be considered together. The overall plant recovery averages 89.83% over the LOM.

13.3 Metallurgical Variability

Variability testing undertaken for the Project is discussed in Sections 13.1 and 13.2 under the various testwork programs.

13.4 Deleterious Elements

The likely deleterious elements identified in the metallurgical testwork programs are discussed in Sections 13.1 and 13.2 under the various testwork program headings.

13.5 Comments on Section 13

In the opinion of the AMEC QPs, the following conclusions are appropriate:

- The metallurgical test results and process design described in this report are essentially the same as those presented in the 2008 Technical Report prepared on the Project.
- Metallurgical testwork and associated analytical procedures were performed by recognized testing facilities, and the tests performed were appropriate to the mineralization type
- Samples selected for testing were representative of the various types and styles of mineralization at Donlin. Samples were selected from a range of depths within the deposit. Sufficient samples were taken so that tests were performed on sufficient sample mass
- Mineralogical studies have shown that the gold is not visible. Testwork analysis indicates a high level of association of gold with arsenopyrite. Other sulphides such as pyrite and marcasite are also present, with reduced tenors of gold.
- Organic carbon, a potential preg robber, is present in the sedimentary ore. It is also present at lower levels in the intrusive ores, believed to be in the form of well-ordered graphite. This form of organic carbon is possibly less likely to preg-rob
- Testwork completed by SGS-Lakefield Research, Hazen Research, and G&T Metallurgical Services (G&T) under Barrick's supervision has shown that the Donlin ore requires pre-treatment prior to cyanidation to recover the gold. Process development work has determined that pressure oxidation is the preferred method

of pre-treatment. Extensive testwork on composites has shown that acceptable gold recoveries can be produced through a combination of flotation pre-concentration, POX, and CIL cyanidation.

- Alternative flowsheets to flotation-POX-CIL were considered, including whole ore pressure oxidation, roasting a flotation concentrate, and bio-oxidation (BIOX). None of these proved to be a viable economic alternative to the flotation-POX-CIL route
- No new metallurgical testwork programs have been carried out.
- The average Bond work index for the ore is in the range of 15 kWh/t. Flotation work has shown that kinetics are initially rapid, but to achieve high recoveries, a combined primary and secondary rougher residence time over 100 minutes, together with a high reagent loading in the system, is required. Clay-like minerals will affect slurry viscosity and settling. Slurry density in the underflow will be less than 50% solids for the concentrate thickeners.
- Partially geologically oxidized (altered) ore in the deposit, up to 7% of the mill feed, is the key non-performing ore type in the flotation circuit. Degradation of the sulphide ore via oxidation in the stockpile will also affect the flotation recovery, applied as 5% recovery loss within flotation on all ores stockpiled for longer than one year.
- Pressure oxidation (POX) has been shown to be successful in releasing the valuable constituents, under certain conditions. To optimize oxidation conditions, the water systems design has been modified to use the highest-quality water in the oxidation circuit. The autoclave design incorporates variable level control to provide better control over operating residence time.
- Areas of design modification from FSU1 include detailing the mercury abatement systems in the gold cyanidation, elution, and refining circuits, and also the treatment of off-gases from the pressure oxidation (POX) process to meet more stringent air emissions legislation
- Metal production and recoveries from the flotation process have been adjusted upward slightly to account for changes in the new mine plan related to ore-type sequencing, and the Prussian blue process has been removed as a backup system to the SO_2/Air method for cyanide detoxification
- Air flotation using the MCF2 flowsheet provides an estimated life-of-mine (LOM) average of 93.0% recovery, with CIL recoveries after POX at approximately 96.6% for an estimated combined plant total gold recovery of 89.8%. The concentrate pull will vary from 15% to 17% and that will result in a concentrate grade of 13.0 to 12.7 g/t Au.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Key Assumptions/Basis of Estimate

The geologic model and resource model (DC9) are based on all drilling through the 2009 drilling campaign. The cut-off date for the DC9 model was 1 November 2009, and no new information was added after that time. Note that the geologic model update includes only the felsic dikes and sills and the overburden model. Shale wireframes were not updated. Also of note is that additional assays for arsenic were provided 18 July 2008, with an updated block model estimate for arsenic only provided on the basis of that data.

The mineral estimate was prepared by Barrick and audited by AMEC. Composites and 3D solid models were constructed utilizing Vulcan® commercial mine modelling software. The models extend a total of 13,200 ft (4,020 m) in the north–south direction, 13,200 ft (4,020 m) in the east–west direction and a total of 3,150 ft (960 m) in the vertical direction. The block model was created with a constant block size of 20 ft x 20 ft x 20 ft (6 m x 6 m x 6 m).

The coordinate system used for resource modelling is NAD83. Resource estimation uses a topographic surface derived from a 2004 survey by Aero-metric. The survey has an accuracy of ±6.6 ft (±2 m).

The Mineral Resource estimates were prepared by Mr. Chris Valorose of Barrick with reference to the Canadian Institute of Mining Metallurgy and Petroleum (CIM) Definition Standards (2010) and CIM Best Practice Guidelines.

14.2 Geological Models

Three-dimensional solids for the geological model were constructed from polygons resulting from geologic interpretation of cross-section and level plans. Tools available in Vulcan® commercial mine design software were used to create the polygons representing the geometry of the intrusive sills and dikes. These were digitized directly on a computer screen snapping to drill holes in section.

Once digitized, the polygons were used to develop 3D wireframe solids to incorporate geologic control into the grade model for the intrusive rocks. The solids were validated and checked for crossing errors, consistency, and closure prior to use. The solids were used to assign the corresponding geological code to the 3D block model.

To limit the size of the model, blocks were assigned a default code of greywacke (ROCK = 93) and were then overprinted with rock values according to the established priorities. Rocks assigned a greywacke code had the lowest priority value.

Rock codes are held as three variable types in the block model. The 'ROCK' variable is assigned with values that include codes for lithology, overburden, and air. The 'ROCK_EST' variable, although similar to 'ROCK' does not include overburden and topography in order to allow unrestricted estimation of blocks at or near the topographic surface. The 'ROCK_MINE' variable holds a simplified rock code nomenclature.

Nine mineral and geological domains were assigned to the database as indicated in Figure 14-1.

The geotechnical domain zone codes were input into the resource model, as required for the LG pit optimization, using domain solids provided by BGC on 27 June 2008.

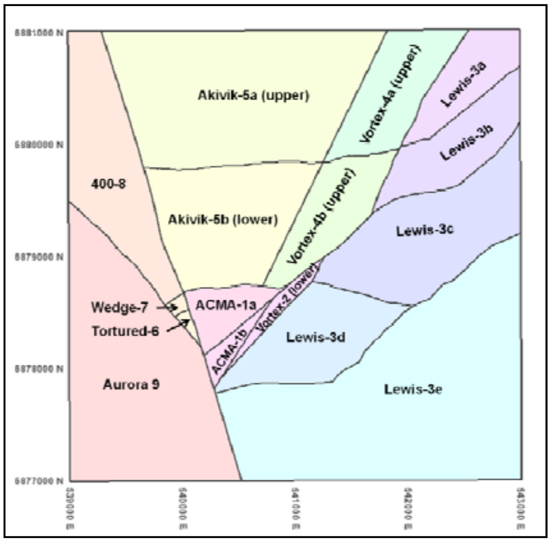
A waste rock management category (WRMC) model was coded to identify overburden from the other WRMC codes.

14.3 Exploratory Data Analysis

To better understand the deposit, AMEC performed several additional EDA studies with the following conclusions:

- Gold mineralization appears to reflect a single population that approximates a log normal distribution and includes a high percentage of sub-economic grades.
- Arsenic histograms suggest two separate near-log normal populations of mineralization, one less than 300 ppm and one greater than 300 ppm.
- Mercury shows a near-log normal distribution with two distinct kinks that probably reflect different detection limits.
- Arsenic shows a near-log normal distribution with two distinct kinks that probably reflect different detection limits.
- Sulfur exhibits two distinct populations with a break at approximately 0.5% which supports the use of using 0.5% sulphur value as the discriminator in the indicator models.

Figure 14-1: Donlin Geology and Mineral Domains



Note: Figure courtesy Donlin Gold

- The major host for the gold is in the felsic dikes and sills which all have similar gold grades which supports combining all the different intrusive lithologies into a single intrusive domain for estimation. Locally, all lithologies may host economic mineralization.
- Arsenic boxplots are similar to the gold boxplots which is expected since the gold was precipitated with arsenopyrite.
- Mercury and antimony are elevated in the intrusives, but to a lesser extent than the gold.
- Sulphur is elevated in the intrusives, but occurs in all lithologies.
- AMEC calculated the Pearson product-moment correlation coefficients and found very good correlations for all modeled elements with gold which is easily explained by the observed gold-arsenopyrite-stibnite-pyrite association.

To investigate changes in grade across the domain boundaries, AMEC created contact profiles or plots of average grades at increasing distances for each estimation boundary with the following observations:

- None of the contacts show a distinct transitional contact profile except for mercury.
- All mineralized to unmineralized contacts (as defined by the discriminator models) are hard for all elements regardless of lithology.

- The intrusive mineralized to shale/greywacke mineralized contacts are hard for gold but soft for all other elements.
- All unmineralized to unmineralized contacts are soft for all elements regardless of lithology although higher grades occur along shale- unmineralized to greywacke unmineralized contact.
- In the DC9 resource model, all contacts are treated as hard which is reasonable for gold and all mineralized to unmineralized domain boundaries based on analyses of the contact profiles.
- The shale mineralized to greywacke mineralized contact is different than the other contacts in that higher gold grades tend to occur along the contact, and the use of hard contacts may lead to local overestimation.
- In general, the use of hard domain boundaries for all estimations appears reasonable.

14.4 Density Assignment

As discussed in Section 11-3, two specific gravity values were used:

- Intrusive Rocks: 2.65
- Greywacke and shale: 2.71.

14.5 Grade Capping/Outlier Restrictions

14.5.1 Gold, Sulphur, Arsenic, Mercury, and Antimony Grade Caps

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. Total sulphur, arsenic, mercury, and antimony assays were not capped.

Assay top cuts were selected from cumulative frequency plots for each major rock type. The distribution for all gold assays shows a well-defined lognormal population with no obvious breaks in the higher-grade trend. However, the grade–frequency trend becomes erratic above 30 g/t Au, deviating from the lognormal approximation line. This is therefore the position at which the raw assay would be capped and represents a metal loss of 1.8%.

Individual frequency distribution plots were generated to determine the appropriate grade cap for each rock type. The grade caps were applied to all raw assays prior to compositing. Capping grades used for each rock type are summarized in Table 14-1. Values represent all assays greater than 0.1 g/t Au.

14.5.2 Neutralization Potential Grade Caps

Values for neutralization potential were also capped, as indicated in Table 14-2. 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 the domains established in the estimation of calcium, and capping values for NP were determined for each domain. High NP populations were noticed in the sediment domains and determined to be from the mafic dikes unit. Because the block model geology does not break out this unit, it was decided to cap the NP value in the sediment domain values below the mafic dikes population to prevent blow-outs of high NP values.

14.6 Composites

14.6.1 Gold, Sulphur, Arsenic, Mercury, and Antimony Composites

Composites were created down each hole at 20 ft (6 m) intervals. The composites were not broken at intrusive or sedimentary rock contact boundaries.

A composite database was generated for Au values where non-assayed (missing) intervals are set to zero. The non-assayed (missing) intervals include primarily trench data, RC data outside the resource area, or tops of holes within overburden. These account for approximately 1% (1,958 out of a total of 193,237 Au assay intervals) of the entire assay database.

Two additional composite databases were generated; one for sulphur; one for arsenic, antimony, and mercury (multi-elements).

14.6.2 Neutralization Potential Composites

A separate composite database was generated for magnesium, calcium, CO_2, and neutralization potential (NP). Composites were also created down each hole at 20 ft (6 m) intervals. The composites were not broken at intrusive or sedimentary rock contact boundaries.

Table 14-1: Summary of Capping Grades for Major Rock Types

Rock Type	Capping Grade in FSU1 (g/t Au)	Capping Grade in FSU2 (g/t Au)	Percentile	No. of Samples Capped	Metal Loss (%)
GWK	25	25	99.07	127	3.61
SHL/ARG	30	30	99.58	15	6.97
SLT	20	20	99.14	17	10.04
MD	30	30	98.04	26	13.05
RDA	20	20	99.50	51	1.23
RDF	16	16	98.91	26	3.35
RDX	26	30	99.84	34	1.22
RDXB	28	28	99.71	22	0.99
RDXL	10	10	98.87	40	2.07

Table 14-2: Summary of Capping Values for Neutralization Potential, with COV and GT Lost

Ca_RKTYP Domain Code	Description	Capping Grade NP	COṼ Uncapped	COṼ Capped	GT Lost (%)
1	RDX, Other, RDF	235	0.68	0.68	0.00
2	RDA	103	0.77	0.77	0.00
3	RDXB	130	0.73	0.59	2.36
4	RDXL	75	0.97	0.75	6.20
5	SHL – NW, SW, SE	125	1.03	0.56	11.66
6	GWK – NW,SW,SE	225	0.89	0.72	3.92
7	SHL – NE	250	0.59	0.55	1.20
8	GWK – NE	240	0.59	0.52	2.47
9	OVB	1.5	0.20	0.85	85.29
All Rock Types		*400*	*0.81*	*0.8*	*0.11*

Note: COV = co-efficient of variation, g/t lost= grams per tonne removed

14.7 Gold and Sulphur Indicator Models

A gold indicator model was used to estimate gold, arsenic, antimony, and mercury grades based on gold composite data. A separate sulphur indicator model was used to estimate sulphur. An indicator value of 0 or 1 was assigned to each 6 m composite interval based on nominal cut-off grades of 0.25 g/t for Au and 0.50% for S.

Intrusive and sedimentary (shale and greywacke) rocks were modelled separately. All 6 m gold composites with an assigned rock code between 1 and 89 (all intrusive rocks) that were below 0.25 g/t Au were assigned an intrusive indicator value of 0. Data with an assigned rock code between 1 and 89 and an assay grade equal to or greater than 0.25 g/t Au were assigned an intrusive indicator value of 1.

All data with an assigned rock code of 92 (shale) or 93 (greywacke) and a grade below 0.25 g/t Au were assigned a sedimentary indicator value of 0.

Data with an assigned rock code of 92 or 93 and an assay grade equal to or greater than 0.25 g/t Au were assigned a sedimentary indicator value of 1.

Similarly, indicator values were assigned to the 6 m composite database for sulphur such that all data with an assigned rock code between 1 and 89 that were below 0.50% S were assigned an intrusive indicator value of 0. Data with an assigned rock code between 1 and 89 and an assay of greater than or equal to 0.50% S were assigned an intrusive indicator value of 1. All data with an assigned rock code of 92 (shale) or 93 (greywacke) and a grade below 0.50% S were assigned a sedimentary indicator value of 0. Data with an assigned rock code of 92 or 93 and an assay grade equal to or greater than 0.50% S were assigned a sedimentary indicator value of 1.

A block discriminator model was then developed by interpolating the assigned indicator values in the composite database in two passes for each major rock type (intrusive, shale, and greywacke) into the block model. The search distances for all passes were 175 m (Major axis –z), 175 m (Semi-major axis –y), and 100 m (Minor axis –x). The orientation of the search ellipse was set to 024° in X, 0° in Y, and -68° in Z. The inverse power of distance method (power = 2) was utilized for block indicator assignment.

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.

As a result of the restriction on sample selection, some areas in the indicator model in each of the major rock types did not receive an indicator value in the first pass. A second indicator pass was performed with the same search and selection criteria as in 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.

The resulting block estimates are values between 0 and 1, which are best considered as the probability that the given block contains grades above the threshold indicator value. For the gold indicator values, a 50% threshold was used to separate blocks that have a high confidence of containing grades greater than the threshold value of 0.25 g/t Au and those that do not. An indicator block value of 0.5 equates to a 50% probability of a block having a gold grade equal to or greater than 0.25 g/t Au.

The same search parameters and sample constraints used for the gold indicator were used to create the sulphur indicator model.

14.7.1 Overburden

The gold grade values in blocks that were coded as overburden were removed. This procedure was completed to prevent overburden blocks from having inappropriate gold values, or at least gold values that will not be recovered. Eliminating gold values in overburden prevents not only erroneous block valuations in the LG pit optimization, but also inconsistent volumetric resource reports.

14.8 Variography Performed in Support of PAG Model

The 6 m composites were used to develop indicator, correlogram, and relative pairwise variograms. The variograms were generated for all sample data and by domain using orientations along the average strike and dip of the mineralized zones identified both geologically and through stereonet 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 semi-variograms generated at 0.25 g/t Au for the 6 m composites were fitted with a spherical model. Ranges of 98.4 ft (30 m) and 147.6 ft (45 m) were observed at 80% and 90% of the total sill variance.

14.9 Estimation/Interpolation Methods

14.9.1 Gold

Gold grades were estimated into the block model using an inverse distance to the third power methodology for the two populations:

- Internal to the mineralized envelope, defined as blocks with indicator values greater than or equal to 50%
- 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 threshold. Intrusive indicator values were back flagged to the appropriate intrusive variable in the composite database. Indicator values in the sedimentary units were back flagged to the sedimentary variable in the composite database.

Interpolation of grade into the blocks was broken into five passes based upon increasing search distances. Gold grades were estimated separately for intrusive rocks, shales, and greywackes, and further sub-divided based upon whether blocks were internal or external to the mineralized envelope.

The initial grade estimation pass used a “box search” with a search range having the same dimensions as a single block. The range was increased for each successive estimation pass, out to a maximum of 125 m using an elliptical search. Search ellipse distances and sample weights were adjusted based on an anisotropic model. Once estimated, blocks could not be overwritten by subsequent estimation passes.

14.9.2 Sulphur

Sulphur grades were estimated using the same methods and parameters as for the gold grade estimation. A series of five passes was used for blocks inside and outside the 0.50% sulphur grade indicator populations. Separate estimation runs were generated for intrusive rocks, shale, and greywacke using search parameters with the same extent, orientation, and constraints as those for gold.

Sulphur data are less extensive than gold data; therefore, sulphur was not estimated for a number of blocks during the inverse distance estimation runs due to a lack of support. Regression curves were derived from the relationship between gold and sulphur for each of the major rock types. The regression formulae were then used to assign sulphur values to non-estimated 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.

14.9.3 Arsenic, Mercury and Antimony

Arsenic, mercury, and antimony grades were estimated using methods and parameters similar to those for the gold grade estimation. Multi-element, 6 m long composites 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 threshold. A series of five passes was made 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.

Data available for As, Hg, and Sb are much less extensive than the data availability for Au and S. Regression curves 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 non-estimated 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.

14.9.4 Calcium, Magnesium and Carbon Di-oxide

Values for CO_2, calcium, and magnesium were estimated into the block model based on an ordinary kriging method within nine estimation domains. The domains consisted of the four major intrusive rock types, two domains of shale, two domains of greywacke, and one domain of overburden. The CO_2 and calcium estimates were used for waste rock management and environmental assessment and the magnesium estimate for metallurgical models.

Data for these elements are limited; therefore, blocks that could not be estimated were assigned the average value within each of the estimation domains.

14.9.5 Neutralization Potential

Neutralization potential (NP) was estimated into the block model for use in the classification of WRMC. The NP database consists of 3,571 ABA samples within the block model limits. These sample intervals were composited into 6 m sample lengths prior to grade estimation. The block geology was back-flagged into the database from the block model to assign geology to each interval.

Estimates were generated using ordinary kriging methodology in eight passes for each of the identified domains. No new variogram models were generated; rather, the same models used in the calcium estimation were used for NP estimation. Block discretization was undertaken with a 4 x 4 x 2 pattern. A minimum of two and maximum of six composites were used for the estimation. The search parameters are identical to those used for calcium estimation.

Blocks that could not be estimated were assigned a value based on regression analysis of NP against Ca. Regression curves were derived from the relationship between NP and calcium for each of the identified domains. The regression formulae were then used to assign NP values to non-estimated blocks based on the estimated Ca grade in each block.

14.9.6 Classification of Waste Rock Management Categories

Several variables were included in the block model to aid with the geochemical classification of waste rock at Donlin, based on ongoing waste rock characterization studies.

Acid Potential (AP) was calculated from the estimated total sulphur concentration (S_T) where:

Acid Potential = 31.25 x Estimated S_T (%)

Neutralization Potential (NP_{CO3}) from carbonate minerals was estimated from:

NP_{CO_3} = 0.85 x NP + 3.4

To avoid a slight bias at low NP below 22.7 kg $CaCO_3$/t resulting from the regression equation, the calculated NP_{CO_3} should not exceed analytical NP when NP is below 22.7 kg $CaCO_3$/t. Therefore, the following rules were applied to the calculation:

If NP ≤ 22.7 kg $CaCO_3$/t: NP_{CO_3} = NP
If NP > 22.7 kg $CaCO_3$/t: NP_{CO_3} = 0.85 x NP + 3.4

The variables NP_{CO3} and AP were estimated for each block separately and were then used to calculate ARD potential (ARD). ARD was modelled using the ratio NP_{CO3}/AP. Recommended ARD categories were assigned to each block according to calculated ARD potential.

The block model estimates for arsenic and sulphur values were used to calculate the ratio of arsenic to sulphur (As/S) for each block. During the process, it was recognised that arsenic leaching was ubiquitous throughout the deposit.

Blocks were classified into seven waste rock management categories (WRMC) subdivided into potentially acid generating (PAG) and non-PAG (NAG) groups, based on ARD potential. Some reclassification of PAG material into the NAG categories resulted during the process.

14.10 Block Model Validation

The gold block model grades were validated 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 6 m x 6 m x 6 m block model.

Grade profile plots were generated for the 6 m x 6 m x 6 m Measured and Indicated resource model as a further validation check.

No estimation biases were noted from the validation reviews.

14.11 Dilution

Grade dilution will be an operational consideration given the nature of the narrow, steeply dipping mineralized zones that characterize the Donlin 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 of mineralized material were determined using a Barrick in-house program referred to as SMUman. Dilution is discussed in detail in Section 15.2.

14.12 Classification of Mineral Resources

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. The classification methodology is summarized in Table 14-3.

14.13 Reasonable Prospects of Economic Extraction

The extent of the classified material that might have reasonable expectation for economic extraction was assessed by applying a Lerchs–Grossmann (LG) pit outline using Whittle software to the Mineral Resources. This conceptual pit used the economic parameters summarized in Table 14-4. Mill recoveries vary by rock type, domain, and degree of oxidation. Recoveries used for calculation of net sales return (NSR) are summarized in Table 14-5.

14.13.1 NSR Calculations for Marginal Cut-off Application

The 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.

Table 14-3: Donlin Gold Project 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	-	3	Block pierced by drill hole	≥0.0	≥0.0
Indicated	-	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

Table 14-4: Assumptions used for Calculation of NSR Values for Mineral Resources

Economic Parameters	Assumptions
Au selling price (resources)	$US1,200/oz
Grams per troy ounce	31.10348
Process cost ($/t)	2.1874 * (sulphur grade) + 10.6485
Rehandle cost ($/t)	$0.20/t
Administrative cost ($/t)	$2.29/t
Refining, freight & marketing (selling costs)	$1.85/oz recovered
Royalty	4.5% * (Au price – Selling cost)

Table 14-5: 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%	$((8.7361 \cdot S^3 - 49.806 \cdot S^2 + 95.233 \cdot S + 30.004) \times 0.966)$

For those blocks with a resource classification of Measured or Indicated, the NSR per tonne value was calculated with the following equations:

General: *NSR = [Au grade] * [Recovery] * [Price Gold less Refining and Royalty Costs] – [Processing Costs + General and Administrative Costs + Rehandling Costs] US$/tonne*

For Mineral Resources: *NSR = [Au grade] * [Recovery] * [US$1200 – (1.85 + ((US$1200 – 1.85) * 0.045))] – [(10.65 + (2.1874 * S%)) + 2.29 + 0.20] US$/tonne*

For Mineral Reserves: *NSR = [Au grade] * [Recovery] * [US$975 – (1.78 + ((US$975 – 1.78) * 0.045))] – [(10.65 + (2.1874 * S%)) + 2.27 + 0.19] US$/tonne*

Therefore, for Mineral Resource reporting, the NSR for the purposes of evaluation of reasonable prospects is the gross revenue, consisting of the total revenue minus production costs. For Donlin, any material that had a positive NSR was treated as having potential for reasonable prospects of economic extraction as a Mineral Resource. The applied cut-off was $0.001/t milled, which represents the net sales return marginal cut off strategy.

14.14 AMEC Review

Gold, arsenic, antimony, mercury and sulphur grades were estimated using a combination of lithologic and indicator domains. AMEC considers the estimation procedure to be appropriate with the following comments:

- The block size of 6 m is good balance between the blocks being small enough to reflect the geometry of the lithologic domains, but large enough to be make construction of the model manageable. The 6 m blocks size is also a subunit of the SMU size used in the mine plan.
- Using intrusive, shale and greywacke domains is a key factor in the design of the modeling methodology, and appropriate as these domains match the different styles of mineralization. The intrusives 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 a more favourable host than the shale.
- Using a discriminator or indicator model to separate mineralized from unmineralized material is appropriate as this design prevents higher-grade assays inside the mineralized domains from smearing into unmineralized domains, and restricts the lower-grade assays in the unmineralized domains from diluting the grades in the mineralized zones.
- The selection of the 0.25 g/t Au discriminator between mineralized and non-mineralized material is lower than the cutoff grade, which adds an unquantified amount of dilution into the mineralized domains.
- Search orientations were the same for all estimations (strike = N24E and dipping 68 to the southeast). This trend correlates the trend of the intrusives in Lewis, but

the mineralization in ACMA appears to be more disseminated and less structurally controlled. Since the ACMA geology and mineralization appears to be different than Lewis, AMEC recommends that the search orientations be reviewed for ACMA.

AMEC reviewed the correlation, regressions, and implementation of the values and found the procedure to be reasonable given the strong correlations between the elements:

- The power regressions used to estimate sulphur, arsenic, mercury, and antimony where the data is too sparse to estimate directly is reasonable if the populations approximate a log normal distribution, and there is a good correlation between the elements and gold. AMEC calculated the Pearson product-moment correlation coefficients and found very good correlations for all elements with gold.

AMEC reviewed the specific gravity data with the following comments:

- The SG data were imported and viewed in relation to the pit design. The geometry of the SG data is more concentrated in the Lewis than the ACMA area, but the entire resource area is adequately covered.
- AMEC performed statistical studies which showed that the SG values are very similar between the intrusive rocks, and between the sedimentary rocks. Low coefficients of variations (CV) demonstrate that there is little variability with 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 resource model was classified using the distance to the nearest composite as stored in the block model during the nearest-neighbor grade estimation. Blocks were classified as Measured only if they were pierced by a drill hole. Indicated and Inferred classifications were based on a combination of distance to the nearest drill hole, the number of drill holes used to estimate a block, and the indicator values. Distance thresholds were based on the lag distances at 80% (30 m) and 90% (45 m) of the variance from an omni-directional indicator correlogram model generated using 6 m composites and a 0.25 g/t Au discriminator.

AMEC reviewed the classification methodology and found the classification to be acceptable with the following observations:

- AMEC calculated gold correlograms in different direction and geologic domains, and compared the lag distances at 80% and 90% sill in the gold correlograms to the distances used for classification. Although there is considerable variability in the lag distances with the different directions, overall the average lag distances are similar the lag distances used for classification.

- Using the criteria that a block must be pierced by a drill hole to be classified as Measured is an outdated classification method and should be revised. Classification is designed to reflect the continuity of the mineralization, and to establishing continuity requires at least two points. Visual inspection of the blocks classified as Measured shows that the Measured material consists of isolated spots around the drill holes which 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 is geologically sensible with coherent zones that reflect a realistic level of geological and grade estimation confidence taking into account the amount, distribution and quality of data.
- 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 spacings. According to industry-recognized guidelines, Indicated Resources should be known within ± 15 percent with 90 percent confidence on an annual basis, and Measured Resources should be known within ± 15 percent with 90 percent confidence on a quarterly basis. 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 Donlin Gold's criteria. Measured would require a 25 m x 25 m drill hole spacing indicating that the current criteria for classifying material as Measured is conservative.

Donlin Gold estimated grades using an inverse distance cubed method, with an increasing five-pass stepped search selection. This creates a model with very little smoothing that is not appropriate for mine planning without post processing or the addition of planned dilution.

- AMEC checked the smoothness of the resource models using the discrete gaussian or hermitian polynomial change-of-support method (Herco). The Herco analysis shows that for an SMU of 12 m x 12 m x 6 m the model is not smooth enough, and is expected to underestimate the tonnes by approximately 8% and overestimate the grade by approximately 5%. Similarly for a for an SMU of 12 m x 12 m x 12 m the model is expected to underestimate tonnes by approximately 10% while overestimating the grade by 7% if no adjustments are made to the model.
- The model, however, was post-processed using an in-house software program (referred to as SMUman) which accounts for dilution, mineralization loss, and helps

identify areas where selective mining using the smaller SMU size has a positive economic benefit. Results from SMUman optimization which include a mixture of both the 12 m x 12 m x 6 m, and 12 m x 12 m x 12 m SMU resulted in the addition of 7.1% waste tonnes which is in agreement with the Herco change-of-support analyses.

Overall, the data and documentation is well organized and in good order. Estimation parameter files were checked for errors by AMEC and found to be created as intended.

14.15 Mineral Resource Statement

Mineral Resources take into account geologic, mining, processing and economic constraints, and have been confined within appropriate LG pit shells, and therefore are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves.

The Qualified Person for the Mineral Resource estimate is Gordon Seibel, R.M. SME Registered Member, an employee of AMEC.

Mineral Resources are reported at a commodity price of US$1,200/oz gold, and have an effective date of 11 July 2011. Mineral Resources are stated in Table 14-6 using an NSR cut-off grade of US$0.001/t milled. Mineral Resources are reported inclusive of Mineral Reserves.

Table 14-6: Mineral Resources Summary Table, (Inclusive of Mineral Reserves) Effective Date 11 July 2011, Gordon Seibel, R.M. SME Registered Member

Category	Tonnage (kt)	Au (g/t)	Contained Au (koz)	S (%)
Measured	7,731	2.52	626	1.15
Indicated	533,607	2.24	38,380	1.08
Total Measured and Indicated	541,337	2.24	39,007	1.08
Inferred	92,216	2.02	5,993	1.08

Notes to Accompany Mineral Resources Table

1. Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
2. Mineral Resources are contained within a conceptual Measured, Indicated and Inferred optimized pit shell using the following assumptions: gold price of US$1,200/oz; variable process cost based on 2.1874 * (sulphur grade) + 10.65; administration cost of US$2.29/t; refining, freight & marketing (selling costs) of US$1.85/oz recovered; stockpile rehandle costs of US$0.20/t processed assuming that 45% of mill feed is rehandled; variable royalty rate, based on royalty of 4.5% * (Au price – selling cost)
3. Mineral Resources have been estimated using a constant net sales return (NSR) cut-off of US$0.001/t milled. The NSR was calculated using the formula: NSR = Au grade * Recovery * (US$1,200 – (1.85 + (US$1,200 – 1.85) * 0.045)) – (10.65 + 2.1874 * (S%) + 2.29 + 0.20) and reported in US$/tonne Assuming an average recovery of 89.54% and an average S% grade of 1.07%, the marginal gold cutoff grade would be approximately 0.57 g/t, or the gold grade that would equate to a $0.001 NSR cutoff at these same values.
4. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
5. Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

14.16 Comments on Section 14

The QPs are of the opinion that the Mineral Resources for the Project, which have been estimated using core drill data, have been performed to industry best practices, and conform to the requirements of CIM (2010).

Factors which may affect the geological models or the conceptual pit shells used to constrain the mineral resources, and therefore the Mineral Resource estimates include:

- Gold price
- Pit slope angles
- Changes to the assumptions used to generate the NSR cut-off
- Changes to the 0.25 g/t threshold for defining the indicator mineralized shells
- Changes in interpretations of fault geometry, in particular the Vortex and Lo faults
- Changes to the search orientations used for grade estimation in the ACMA area
- Review of the Measured classification criteria.

15.0 MINERAL RESERVE ESTIMATES

15.1 Key Assumptions/Basis of Estimate

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—stockpiles, waste dumps, offices, maintenance shops, fuel storage, permanent camp, power-generating facilities, mineral processing facilities, tailings pond, water storage ponds—will be external to the ultimate pit design and its area of influence. The pit shell generation is constrained in the northwestern part of the ACMA mining area, however, to prevent it from encroaching on Crooked Creek, which is a salmon-bearing stream.

Mineral reserves were optimized for all Measured and Indicated blocks assuming a gold selling price of $975/oz. The reserves included dilution based on the block model, which identified blocks amenable to bulk mining (12 m high benches) and selective mining (6 m high benches). NSR assumptions for Mineral Reserves are summarized in Table 15-1.

The ore considered for processing in the optimization was based on a marginal NSR cut-off that varied from block to block. Material was considered to be ore if the revenue of the block exceeded the processing and G&A cost (marginal NSR). 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 varied block by block. Neither a minimum cut-off grade nor raised cut-off metal grade was applied.

15.2 Dilution and Mining Losses

During FSU2, a software application referred to as SMUman was developed collaboratively between NCL Ingenieria Y Construccion S.A (NCL) and Barrick staff. The objective of the software was to assist in the identification of areas within the Donlin resource where selective mining would 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 20 ft (6 m) bench and as part of a 40 ft (12 m) bench. Element grades for a 40 ft (12 m) bench block are derived by calculating a tonnage-based weighted average of the upper and lower 6 m blocks.

Table 15-1: Assumptions used for Calculation of NSR Values for Mineral Reserves

Economic Parameters	Assumptions
Au selling price (reserves)	$975/oz
Grams per troy ounce	31.10348
Process cost ($/t)	2.1874* (sulphur grade) + 10.65
Administrative cost ($/t)	$2.27/t
Rehandle cost ($/t)	$0.19/t
Refining, freight & marketing (selling costs)	$1.78/oz recovered
Royalty	4.5% * (Au price – Selling cost)

The SMUman software replicates the block NSR $/t calculation detailed in Section 14.13.1. In addition, 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 greater requirements for ancillary equipment. A typical example of the differential cost calculation was obtained from Kalgoorlie Consolidated gold Mines (KCGM). A total additional selective mining cost penalty of US$0.52/t was applied to the SMUman study.

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 are then simulated for each scenario based on the assumed degree of mining selectivity.

The FSU2 assumes that the minimum SMU size for calculation of ore loss and dilution is approximately 5,000 tonnes when mining a full 12 m bench. This figure was based on operating experience at the KCGM joint venture in Western Australia, which uses an equipment fleet similar to the one proposed for Donlin. As applied to the raw 6 x 6 x 6 m resource model:

- On a 6 m mining bench, a minimum SMU size of four contiguous 6 x 6 x 6 m blocks (in plan view) was required to form an ore pod. Note that this can include diluting waste as long as the overall pod grade is above cut-off. This equates to a minimum mining unit size of approximately 2,330 tonnes.
- On a 12 m mining bench, a minimum SMU size of four contiguous 6 x 6 x 12 m blocks (in plan view) was required to form an ore pod. Note that this can include diluting waste as long as the overall pod grade is above cut-off. This equates to a minimum mining unit size of approximately 4,660 tonnes.
- 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.

The approach was somewhat subjective. Selective mining of ore pods (using the assumed selective mining penalty of $0.52/t mined) generally resulted in all ore pods benefiting from selective mining. This was considered impractical given the large (+400 kt/d) scale of the operation, and because the selective mining penalty does not fully reflect additional (intangible) operational complexities associated with split-bench mining. An alternative approach was adopted, whereby 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.

The mine plan is based on Measured and Indicated resources. All selective mining areas (on 6 m benches) based on the 6 x 6 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 FSU2, the AMEC QP and the Barrick mining teams agreed that a pragmatic approach would be taken, such that the 12 m block was assigned the higher confidence category, for example:

- If one of the 6 m blocks was Inferred and one was Indicated, then the entire 12 m block was treated as Indicated. The grade of all Inferred blocks was set to zero at the start of the process. Therefore the combined grade of the 12 m block is derived from the Measured or Indicated metal grades only.

The following net adjustments were applied to the resource model for pit optimization:

- A loss of 1.26% on tonnes above cut-off (0.76% on oz) due to the removal of pods smaller than the minimum four-block SMU size
- Added diluting waste of 7.1% on tonnes (1.0% on oz) because of the inclusion of waste pods smaller than the minimum four-block SMU size when surrounded by blocks above cut-off
- A net adjustment of 5.8% additional rock tonnes (0.24% on oz).

As a result of the chosen method of treating Inferred material during the 12 m re-blocking process, 2.6% of the total ore tonnage (0% of the oz) is Inferred blocks that have been reclassified as either Measured or Indicated (Table 15-2).

15.3 Conversion Factors from Mineral Resources to Mineral Reserves

15.3.1 Mining Costs

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, and ore control

The reference mining cost used in the pit shell generation was $1.51/st ($1.668/t) of material mined. Ore and waste mining costs were assumed to be equal. An incremental increase in cost with depth of $0.00091/st/ft ($0.0031/t/m) was applied to blocks below a reference elevation of 722 ft (220 m) to represent increased haulage cost with pit depth.

The processing cost did not include any incremental drilling, blasting, loading, hauling, or ore control costs.

Table 15-2: Net Model Adjustments (within pit design)

Item	Ore (kt)	Au (g/t)	S (%)	Au (koz)
Undiluted ore	477,083	2.20	1.07	33,767
Net adjustments	27,728	0.09	0.88	82
Total M&I with adjustments	504,811	2.09	1.06	33,849
Adjustments				
Isolated ore blocks	-6,016	1.33	0.91	-257
Planned dilution waste	33,745	0.31	0.88	339
Sum of adjustments	27,728	0.09	0.88	82
Ore loss (%)	-1.26	-	-	-0.76
Dilution (%)	7.07	-	-	1.01
Net (%)	5.81	-	-	0.24
Inferred included	13,193	-	0.89	0
Inferred (%)	2.6	-	-	0

The following consumable costs were used in the calculations:

- Oil: $80/barrel
- Delivered diesel: $2.72/gal ($0.72/L)
- Bulk explosives (70:30 blend of emulsion:ANFO): $490/t.

Barrick provided the prices for oil and major consumables, plus labour rates, as applicable at the time of the study.

Mining cost assumptions were based on those derived in the throughput rationalization study and subsequent updates to reflect updated cost drivers. The average total mining cost was $2.14/t.

15.3.2 Processing Costs

The processing cost was based on first-principle calculations for a 59.0 kst/d (53.5 kt/d) processing facility. The processing cost (PC) was expressed as a function of the sulphur grade, as follows:

$$PC = 2.1874 * (S\ \%) + 10.6485$$

Costs were not included for any expansion of the tailings storage facility or other sustaining capital.

15.3.3 Recovery

Gold recovery values were based on work completed for the Donlin Gold Project. Recoveries for non-oxide ores are quoted as a constant for each rock type, whereas recoveries for oxide ores vary with sulphur grade. The recoveries applied in pit optimization are listed in Table 15-3.

15.3.4 Overhead Costs

G&A costs were based on first-principle calculations for a remote open pit mine supported by a fly-in operation and a camp. G&A costs, inclusive of aviation, catering, camps, clinic, health and safety, and logistics, are $2.06/st ($2.27/t) of material processed and were added to the processing cost in the LG analysis, as is typically assumed for cases limited by processing rate.

15.3.5 Refining, Freight, and Royalties

Refining, freight, and royalty values were provided by Barrick. Based on actual costs at Barrick operations, the combined refining and freight cost was $1.78/oz of gold.

An average royalty charge of 4.5% of the net gold value was added after the refining and freight cost had been applied. The 4.5% royalty was applied through-out the life of the Project based on an assumed 25-year mine life.

The refining, freight, and royalty costs were applied to the selling cost per ounce in the LG pit optimization using the following equation:

Selling Cost = 1.78 + ((975 – 1.78) x 0.045) and reported in $/oz

15.3.6 Metal Prices

A gold price of $975/oz was used for the pit optimization.

Table 15-3: Pit Optimization Process Recoveries

Deposit Area	Recovery (%)
Non-Oxide Ores	
Akivik	94.17
400	93.55
ACMA	93.05
Aurora	93.61
Vortex	91.82
Lewis	91.52
Gwk	88.22
Shl	86.66
Oxide/Weathered Ores	
Sulphur grade (S) >1.8%	87.90
Otherwise	= (8.7361*(S)^3 - 49.806*(S)^2 + 95.233*(S) + 30.004)*0.966

15.3.7 Pit Slopes

Geotechnical domains, design sectors, slope angles, and associated assumptions were provided by BGC Engineering Inc. (BGC), an applied earth sciences company. 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 LG 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 slope angles used in the pit optimizations are shown in Table 15-4.

15.3.8 Sensitivity of Optimized Pit

A sensitivity study was performed on the pit optimization to assess the sensitivity of the Measured and Indicated Mineral Resources to changes in gold price, recovery, and operating costs. Factors that favourably affected the revenue stream and cut-off NSR resulted in larger pits with higher recovered ounces at lower average grades. The process recovery and gold price had more influence on the ore tonnes and recovered ounces than did the process and G&A costs and mining costs. However, the mining cost had more influence on the total size of the pit than did the process and G&A costs.

Table 15-4: Pit Optimization Slopes

Domain	BGC Design Sector	Whittle™ Bearing	Slope Angle	No. of Ramps	Reduction	Reduced Slope Angle	Whittle™ Transition Adjusted
1	219	039	038	001	2.3	035.0	035.0
1	263	083	032	003	6.2	026.0	026.0
1	314	134	042	001	4.7	037.0	036.0
1	329	149	046	002	9.8	036.0	036.0
1	015	195	038	001	3.1	034.0	034.0
1	068	248	042	001	2.7	039.0	039.0
1	110	290	035	001	1.8	033.0	033.0
1	171	351	033	002	4.7	028.0	028.0
2	194	014	030	003	3.6	026.0	027.0
2	217	037	034	003	4.5	030.0	027.0
2	232	052	030	003	3.6	026.0	027.0
2	265	085	027	003	3.0	024.0	024.0
2	299	119	028	003	3.2	025.0	025.0
2	344	164	026	003	2.8	023.0	023.0
2	36	216	036	003	5.0	031.0	031.0
2	115	295	027	003	3.0	024.0	024.0
3	199	019	031	001	1.5	030.0	030.0
3	254	074	028	001	1.3	027.0	027.0
3	308	128	032	001	1.6	030.0	032.0
3	330	150	037	001	2.1	035.0	033.0
3	12	192	046	003	7.7	038.0	038.0
3	60	240	038	003	5.7	032.0	032.0
3	105	285	034	003	4.7	029.0	029.0
3	156	336	037	001	2.1	034.0	034.0
4	190	010	031	001	1.5	029.0	029.0
4	246	066	036	001	2.0	034.0	035.0
4	272	092	045	001	2.9	042.0	037.0
4	341	161	045	001	2.9	042.0	042.0
4	55	235	045	001	2.9	042.0	042.0
4	85	265	044	001	2.8	041.0	040.5
4	118	298	040	001	2.4	038.0	036.0
4	140	320	036	001	2.0	034.0	034.0

15.4 Mineral Reserves Statement

Mineral Reserves have been modified from Mineral Resources by taking into account geologic, mining, processing, and economic parameters and therefore are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves.

The Qualified Person for the Mineral Reserve estimate is Kirk Hanson, P.E., an AMEC employee.

Mineral Reserves are reported at a gold price of $975/oz gold, and have an effective date of 11 July 2011.

Mineral Reserves are summarized in Table 15-5.

Table 15-5: Proven and Probable Mineral Reserves, Effective Date 11 July 2011, K.Hanson, P.E.

Category	Tonnage (kt)	Au (g/t)	Contained Au (koz)	S (%)
Proven	7,683	2.32	573	1.12
Probable	497,128	2.08	33,276	1.06
Total Proven and Probable	*504,811*	*2.09*	*33,849*	*1.06*

Notes to Accompany Mineral Reserves Table

1. Mineral Reserves are contained within Measured and Indicated pit designs, and supported by a mine plan, featuring variable throughput rates, stockpiling and cut-off optimization. The pit designs and mine plan were optimized on diluted grades using the following economic and technical parameters: Metal price for gold of US$975/oz; reference mining cost of $1.67/t incremented $0.0031/t/m with depth from the 220m elevation (equates to an average mining cost of $2.14/t), variable processing cost based on the formula 2.1874 x (S%) + 10.65 for each $/t processed; general and administrative cost of US$2.27/t processed; stockpile rehandle costs of 0.19/t processed assuming that 45% of mill feed is rehandled; variable recoveries by rocktype, ranging from 86.66% in shale to 94.17% in intrusive rocks in the Akivik domain; refining and freight charges of US$1.78/oz gold; royalty considerations of 4.5%; and variable pit slope angles, ranging from 23º to 43º.
2. Mineral Reserves are reported using an optimized net sales return (NSR) value based on the following equation: NSR = Au grade * Recovery * ($975 – (1.78 + ($975 – 1.78) * 0.045)) – (10.64 + 2.1874 * (S%) + 2.27 + 0.19) and reported in $/tonne. Assuming an average recovery of 89.54% and an average S% grade of 1.07%, the marginal gold cutoff grade would be approximately 0.57 g/t, or the gold grade that would equate to a $0.001 NSR cutoff at these same values.
3. The life of mine strip ratio is 5.48. The assumed life-of-mine throughput rate is 53.5 kt/d
4. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
5. Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

15.5 Comments on Section 15

The AMEC QPs are of the opinion that the Mineral Reserves for the Project, which have been estimated using core drill data, appropriately consider modifying factors, have been estimated using industry best practices, and conform to the requirements of CIM (2010).

Factors which may affect the Mineral Reserve estimates include: effectiveness of the dilution model, gold price, metallurgical recoveries, geotechnical characteristics of the rock mass, ability of the mining operation to meet the planned annual throughput rate assumptions for the process plant, capital and operating cost estimates, effectiveness of surface and ground water management, and likelihood of obtaining required permits and social licenses. The AMEC QPs are of the opinion that these potential modifying factors have been adequately accounted for using the assumptions in this Report, at this feasibility level of study, and therefore the Mineral Resources within the mine plan may be converted to Mineral Reserves using the appropriate confidence categories.

Factors which may affect the assumptions in this Report include:

- Commodity price
- Unrecognized structural complications in areas with relatively low drill hole density could introduce unfavourable pit slope stability conditions.
- The stability of the south walls of the ultimate ACMA and Lewis pits is sensitive to the interpreted orientations of the Vortex and Lo faults, respectively. The current overall slope designs of these two walls may need to be modified if the interpretations of these faults are updated.
- Given the natural variations in the bedding dips, parts of footwall slopes may need to be excavated with flatter bench face angles than estimated for the feasibility design. This could potentially decrease the overall slope angle for the footwall slope(s).
- The residual friction angles of the ash beds are estimated to be 14°. If these low-strength layers are identified within footwall slopes, then, combined with bedding thicknesses of less than 2 m, artificial support may be required for the footwall bench faces
- Trench and other surface exposures show extremely complex intrusive rock contacts on an ore polygon scale that cannot be accounted for in wireframe interpretations. The ore control plan, which includes angled drill hole delineation of ore zones, should mitigate these uncertainties.
- If the mining rate results in the pit walls encountering the water table, producing extensive seepage into the pit, then additional horizontal drains and pumping wells will be required. Similarly, if the bulk bedrock hydraulic conductivity is lower in some areas of the pit than assumed for the base case, then the density of vertical wells/horizontal drains will need to be increased
- Stream flows in the vicinity of the open pit should be monitored continuously as the mine develops to minimize impacts to the creek. Additional hydrogeological investigations have been proposed for the next design phase to assess the potential for this concern.
- An estimation of the likely duration required to obtain the permits and social licenses to construct the gas pipeline and operate the planned mine is incorporated in this study; any changes to that time-frame may impact the assumptions used in the financial analysis, and therefore the declaration of Mineral Reserves.

AMEC notes that NovaGold's partner, Barrick, treats mineralization at Donlin as Measured and Indicated Mineral Resources, rather than Proven and Probable Mineral Reserves for securities reporting, accounting, and other public disclosure purposes, based on an assessment of qualitative, non-technical factors.

AMEC also notes that these same qualitative non-technical factors could affect the Project development timeline discussed in Section 24 of this Report if they are used by Barrick as a basis for development decisions.

16.0 MINING METHODS

16.1 Throughput Considerations

The preferred development is for a 55 kst/d (50 kt/d) process facility with on-site power; and a mine capacity of 485 kst/d (440 kt/d) with an elevated cut-off policy applied in the initial part of the mine life. The processing rate was upgraded to 58 kst/d (53.5 kt/d) during the FSU2 design phase to take into account processing design constraints and rationalization of the proposed pressure oxidization circuit to be installed.

16.2 Pit Design

Parameters considered in the pit optimization process are discussed in Section 15.

16.3 Geotechnical Considerations

BGC provided feasibility-level slope design criteria for the proposed Donlin open pit. The design incorporates data collected and evaluations conducted by BGC from 2004 to 2007. BGC collected additional data in 2010 to confirm that the FS slope design parameters remained applicable to the expanded FSU2 open pit. The pit shell developed for the FSU2 conforms to the feasibility-level geotechnical slope design parameters. Final slope parameters are those indicated in Table 15-4.

16.3.1 Rock Mass Model

The rock mass model is divided into geotechnical units, which for Donlin pit slope design are equivalent to the rock units as defined by the geological model. The sedimentary rocks of the mine-scale geology model are locally divided into eight units. The complex inter-bedding of the sedimentary rocks makes it impractical to delineate geotechnical units comprised entirely of shale, siltstone, or greywacke. The five intrusive rock types of the mine-scale geological model were combined into a single geotechnical unit (INT) for the feasibility slope design. Based on testing and logging completed to date, variations in the texture and mineralogy of these five units are expected to have a minor effect on their geotechnical behaviour.

The dominant structural fabric is bedding and bedding parallel discontinuities. Joints forming the minor structural fabric are best developed in the greywacke beds and intrusive units. Faults within the mine-scale geological model that have no in-fill have an estimated residual friction angle of 26.5°. Fault in-fill dominated by silt/clay gouge has an estimated residual friction angle of 26°; in-fill dominated by disintegrated/decomposed rock has an estimated residual friction angle of 29°. The FS-level slope design assumes a design friction angle of 26° for all faults and shears in the mine-scale geological model. Based on the BGC geotechnical database, this represents the most likely fault character to be encountered.

16.3.2 Open Pit Slope Design

For the Donlin FSU2-level design, the inter-ramp scale slope kinematic analyses were completed first to determine achievable slope angles and define the design sectors for the pit design. Conventional deterministic analyses were undertaken for the inter-ramp slopes assuming that designs satisfy a minimum static factor of safety of 1.2.

The open pit design was completed by Barrick using the slope design parameters provided by BGC. BGC checked selected cross-sections of the final pit design to confirm that the pit wall angles met the slope design criteria developed for the FS, FSU1, and FSU2. Overall slope stability analyses for depressurized conditions predicted by the 3D numerical groundwater modelling and for fully saturated groundwater conditions were completed for nine cross-sections developed from the feasibility-level pit slopes and the feasibility geotechnical model. Two critical areas of the proposed pit were highlighted during the slope design process:

Lewis Pit – Northeast Wall (Design Sector I-219)

Given the monoclinal dip of the sediments in this area of the pit, where the bedding dips at an average of 43°, a standard 65° to 70° bench face would undercut bedding and likely result in instability. As a result, a footwall mining configuration has been recommended and will achieve the maximum slope angle in this area.

Stability analyses to assess standard potential footwall instability modes indicate that a 160 ft (48 m) high unbenched slope will be stable against buckling, bi-linear slab, and ploughing failures. To practically dewater this area and maintain adequate protection against rock and ice fall, a rockfall catchment berm should be placed every 160 ft (48 m) vertically. The berm will need to be a minimum of 26 ft (8 m) wide, increasing to 43.7 ft (13 m) wide at every second bench to accommodate water wells, horizontal drain hole collection systems, a rockfall catchment ditch/net, and a roadway with a 16.5 ft (5 m) wide running surface for service vehicles. This results in a 38° inter-ramp slope angle for this wall.

ACMA Pit – South Wall (Design Sector IV-340)

Stability analyses suggest that kinematically possible inter-ramp failures in this part of the wall are unlikely and consequently that relatively steep inter-ramp slopes (up to 50°) are possible in the south and southwest wall. The wall angles are currently limited by the 79 ft (24 m) maximum bench height being used for the assessments.

However, rock mass failure analyses of the south wall (dip direction towards 360°) indicate that this area of the pit will have to be dewatered and the overall slope angle cannot exceed 45°. The presence of a flat (20° dip) fault set dipping to the north could also play a significant role in the stability of this wall. The full extent and continuity of the flat set cannot be established from the drilling conducted to date; however, it may be prudent to avoid pit wall orientations that the flat-dipping faults would affect.

16.3.3 Recommended Design Parameters

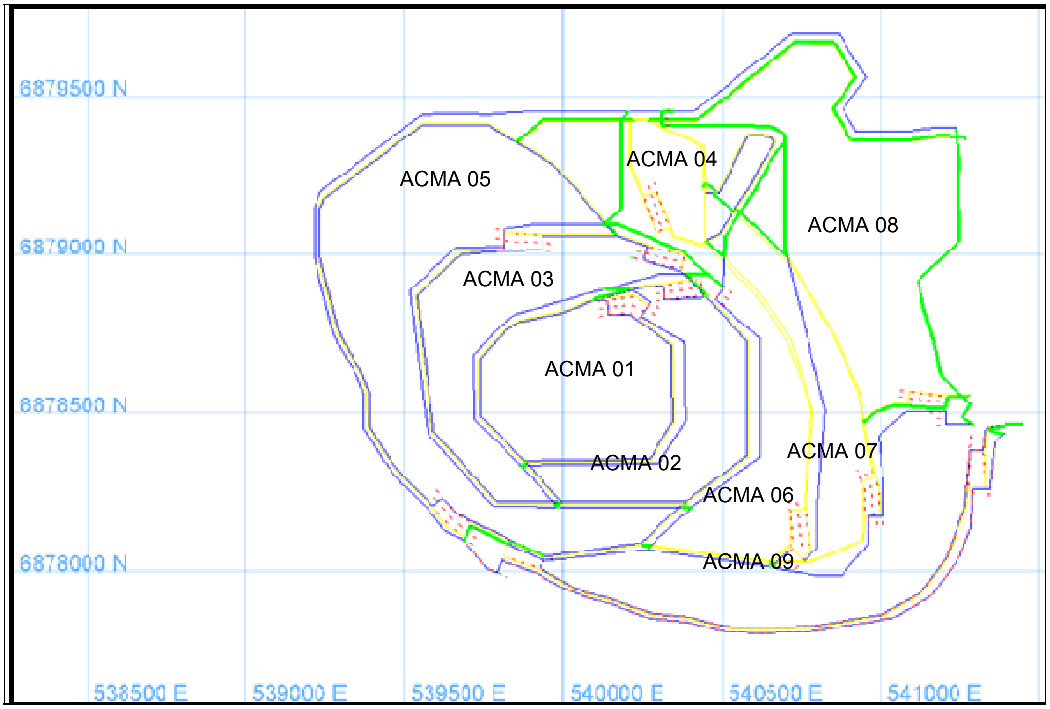
The general design parameters used in the detailed pit design are as follows:

- Bench height, single-bench mining: 20 and 40 ft (6 and 12 m)
- Height between catch benches: 79 or 157 ft (24 or 48 m)
- Bench face angle: : 43° to 65° (variable)
- Berm width (variable): 26 to 125 ft (8 to 38 m)
- Total width allowance, temporary roads: 148 ft (45 m)
- Total width allowance, final roads: 131 ft (40 m)
- Running surface on final two-way roads: 95 ft (29 m)
- Minimum road inside radius on corners: 66 ft (20m)
- Berms and ditches: 13 to 20 ft (4 to 6 m)
- Maximum grade uphill loaded: 10%
- Maximum grade downhill loaded: 8%.

16.4 Pit Phases

The ACMA ultimate pit has been divided into nine phases based on optimized nested pit shell guidance, gold grade, strip ratio, the ability to access the pit, and locations for waste backfill. The planned ACMA pit has a top elevation of 879 ft (268 m) above sea level (asl) and a bottom elevation of 1,129 ft (344 m) below sea level (bsl). The nine phases are delineated in Figure 16-1.

Figure 16-1: ACMA Phases in Plan at 94 m Elevation



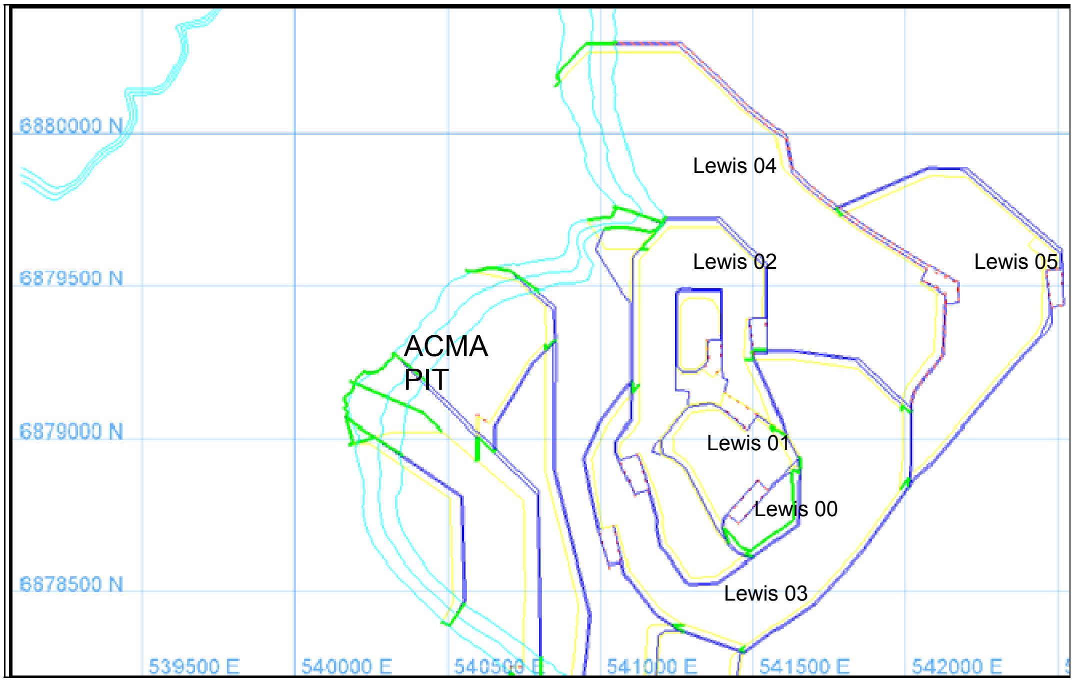
The Lewis pit will be on a hill directly above and to the northeast of the ACMA pit, at an elevation ranging from 1,430 ft asl to 223 ft bsl (436 m asl to 68 m bsl). The Lewis ultimate pit has been divided into six phases based on optimized pit shell guidance, gold grade, and ramp access. The five phases are delineated in Figure 16-2.

16.5 Haul Roads

Haul roads are required between the pit phases and the ore crusher, waste dumps, overburden stockpiles, construction areas, and truckshop. The roads have generally been laid out with a cut-and-fill balance. Roads within the ultimate waste dump are all fill construction. The road design is based on the following design parameters:

- Total width allowance temporary roads: 148 ft (45 m)
- Total width allowance on final two-way roads: 131 ft (40 m)
- Running surface on final two-way roads: 95 ft (29 m)
- Minimum road inside radius on corners: 66 ft (20 m)
- Berms and ditches: 29 to 36 ft (9 to 11 m)
- Maximum grade in pit, uphill loaded: 10%

Figure 16-2: Lewis Phases in Plan at 178 m Elevation



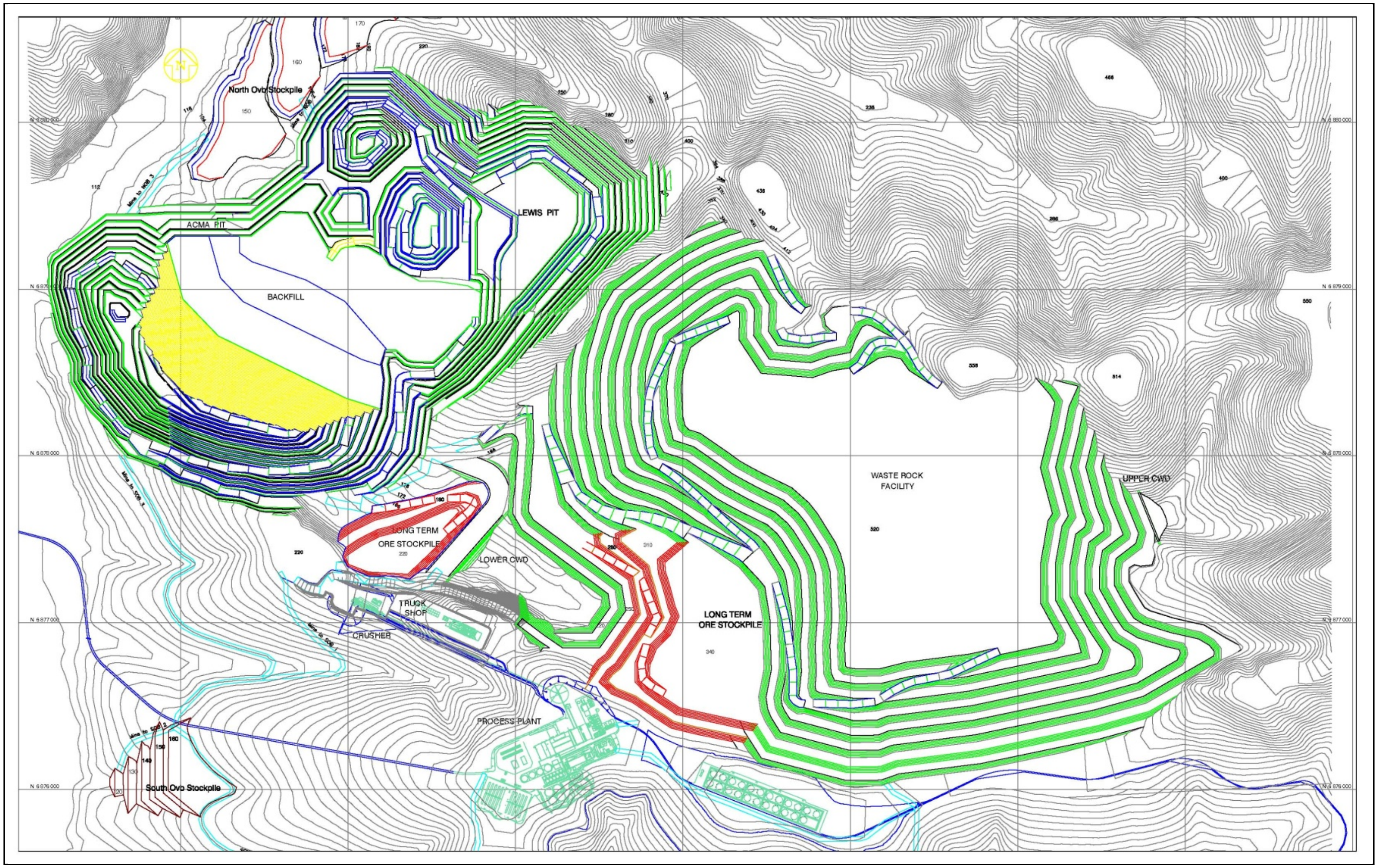
The initial phases of the two pits are independent, but they partially merge later in the mine life. The final overall pit layout plan is included as Figure 16-3.

- Maximum grade ex pit, uphill loaded: 8%
- Maximum grade downhill loaded: 8%.

16.6 Production Schedule

Years indicated in this section are for mine planning purposes only. Formal approval of any mining operation is contingent on approval from the Boards of Donlin Gold, Barrick, and NovaGold, and receipt of appropriate regulatory and social permits.

Figure 16-3: End-of Mine Plan Layout of Open Pit



16.6.1 Planned Production Schedule

Preproduction has been defined as starting in April 2018 and finishing at the end of December 2018, when the main orebody is exposed. Mill production starts in July 2019. The operating mine life is estimated to be 25 years based on a nominal processing rate of 58,900 stpd (53,500 t/d).

In accordance with the convention, the year 2018 (preproduction) was scheduled on a quarterly basis, while schedules for the first year of production, 2009, were developed by month. Years 2 and 3 (2020/21) are scheduled quarterly, and all remaining periods are scheduled annually.

All scheduling was carried out on 6 m benches. Given the geometry of the orebody and the distribution of the ore, almost all of the waste material, and those ore zones that can be mined without significant loss or dilution, will be mined on 12 m benches.

The schedule incorporates long-term and short-term ore stockpiles. The long-term stockpile will hold all ore produced at the mine in excess of plant feed, separated into three sections according to sulphur grade for blending purposes, as follows:

- High S grade (HS) S ≥1.4% and S <1.8%
- Medium S grade (MS) S ≥0.9% and S <1.4%
- Low S grade (LS) S ≥0.6% and S <0.9%.

Long-term reclaim is based only on sulphur category at the average Au grade of the stockpile; however, the first 21 Mt (higher grade) ore is stored separately in front of the Lower contact water dam. Other high-grade stockpiles (of ore to be processed during mine life) can be made in the AMCA 8/9 area to further separate preferential mill feed. Opportunities to apply value segregation and reclaim higher grades first may become evident in further studies. The short-term stockpile was established to cope with daily variations in plant capacity and to accommodate fluctuations in the average daily mill feed.

16.6.2 Pit–Phase Mining Rates

The key features of the planned mine schedule are as follows:

- The production plan is developed on a quarterly basis for the first four years from Year -1 to Year 3 and annually thereafter, except for Year 1, where a monthly schedule was done

- The mine will operate 355 d/a, with 10 days allowed for delays due to winter conditions
- The plant is scheduled to operate 365 d/a
- The average productivity of the main loading units is 46 to 53 kt/d (hydraulic shovel) for ore and 54 to 63 kt/d for waste material
- Pre-stripping from inside the pits totals 20 Mt
- The mine can sustain maximum material movement of 437 kt/d (355 d/a basis)
- ACMA phase 8 is completed at the beginning of Year 15, allowing in-pit waste dumping to begin
- ACMA phase 9 is mined out in Year 21, providing most of the space for the in-pit waste dump
- ACMA phases 1, 2, and 3 and Lewis phases 0, 1, and 2 are essentially one-year ore production phases. Accelerated and simultaneous mining is driven by the requirements for ore blending
- The highest rate of annual vertical advance is 96 m in Lewis phase 3 in 2030 and 2031. During this period approximately 95% of material is mined on 12 m benches
- Pit floor elevations (and therefore vertical advance rates) show the rate at which the mine must be dewatered to allow this development
- The initial phases are generally mined at a lower rate with the intent of keeping as many alternative areas open as possible in each phase
- In the production years, three to four phases will be active in any given period, with four active phases per year from Years 1 to 15. This is driven by the requirements for ore blending and to make ACMA available for the pit waste dump
- The maximum mining rate of 437 kt/d (355 day basis) is achieved in Year 6. The average rate increases progressively from 350 kt/d (Years 1 to 3) to 417 kt/d (Years 4 to 11)
- The mining rate peaks at approximately 206 kt/d (355 day basis) in ACMA phase 2 in Year 1 and at more than 203 kt/d in two periods of ACMA phase 5 (Years 7 and 8). The permanent double-access strategy and wider phases allow for high mining rates

Table 16-1 summarizes the proposed life-of-mine production schedule. Figure 16-4 graphically summarizes the planned mining rate per pit phase and Figure 16-5 provides the breakdown between ore and waste on an annual basis.

Table 16-1: Summary Projected Mine Production Plan by Year

		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 OVB (kt)	Total Rock (kt)
0	2018	1,724	30.42	1.97	0.98	12,022	463	1,485	172	4,133	20,000
1	2019	16,202	40.41	2.18	0.95	66,814	3,591	8,722	224	4,020	99,574
2	2020	21,064	50.25	2.46	1.10	80,766	3,625	8,069	104	6,372	120,000
3	2021	26,757	44.73	2.28	1.09	83,267	4,674	8,473	334	245	123,750
4	2022	33,119	34.44	2.03	0.94	79,933	3,555	7,390	753	7,750	132,500
5	2023	32,596	38.55	2.10	0.99	93,621	3,289	5,801	181	7,013	142,500
6	2024	30,508	39.12	2.07	1.09	112,976	3,661	4,698	55	602	152,500
7	2025	36,217	38.85	2.06	1.12	109,241	3,612	5,587	17	325	155,000
8	2026	23,321	38.55	2.02	1.02	115,684	3,900	7,124	91	1,131	151,250
9	2027	21,522	45.57	2.26	1.01	119,776	3,080	5,345	39	237	150,000
10	2028	28,193	43.72	2.24	1.09	113,272	3,211	4,422	25	876	150,000
11	2029	25,996	34.75	1.91	1.08	111,763	4,487	5,540	6	2,208	150,000
12	2030	26,251	41.86	2.15	1.08	111,446	4,202	6,433	15	1,652	150,000
13	2031	19,971	36.79	1.96	1.01	119,363	4,599	5,314	25	729	150,000
14	2032	15,351	33.99	1.85	1.02	128,263	2,952	2,889	-	545	150,000
15	2033	14,188	33.14	1.83	0.99	129,078	2,458	3,519	169	590	150,000
16	2034	11,848	36.68	1.97	1.06	129,060	3,267	5,415	9	401	150,000
17	2035	15,063	35.12	1.90	1.00	127,844	2,303	4,172	9	609	150,000
18	2036	11,619	32.90	1.84	1.02	113,314	2,447	4,265	7	1,348	133,000
19	2037	14,559	32.98	1.81	0.98	104,059	2,279	3,776	32	1,294	126,000
20	2038	15,475	46.46	2.31	1.02	127,479	3,025	3,956	46	20	150,000
21	2039	18,407	52.76	2.56	1.20	124,831	3,159	3,578	4	21	150,000
22	2040	15,489	39.44	2.10	1.36	98,742	3,145	2,123	-	-	119,500
23	2041	8,725	31.63	1.78	1.11	63,562	1,221	740	-	-	74,247
24	2042	16,336	35.25	1.92	1.10	35,727	2,105	2,832	1	-	57,000
25	2043	4,308	29.56	1.70	1.15	7,096	717	863	2	-	12,986
Total		**504,811**	**39.44**	**2.09**	**1.06**	**2,518,999**	**79,028**	**122,529**	**2,318**	**42,122**	**3,269,807**

Figure 16-4: Proposed Mining Rate per Pit Phase

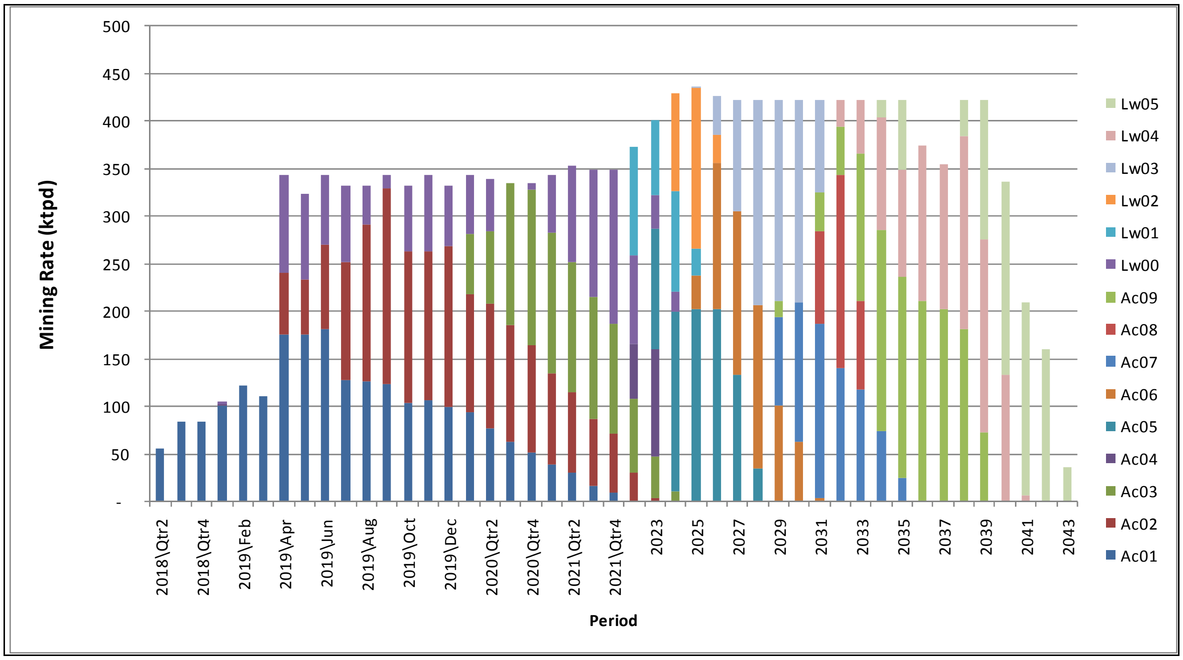
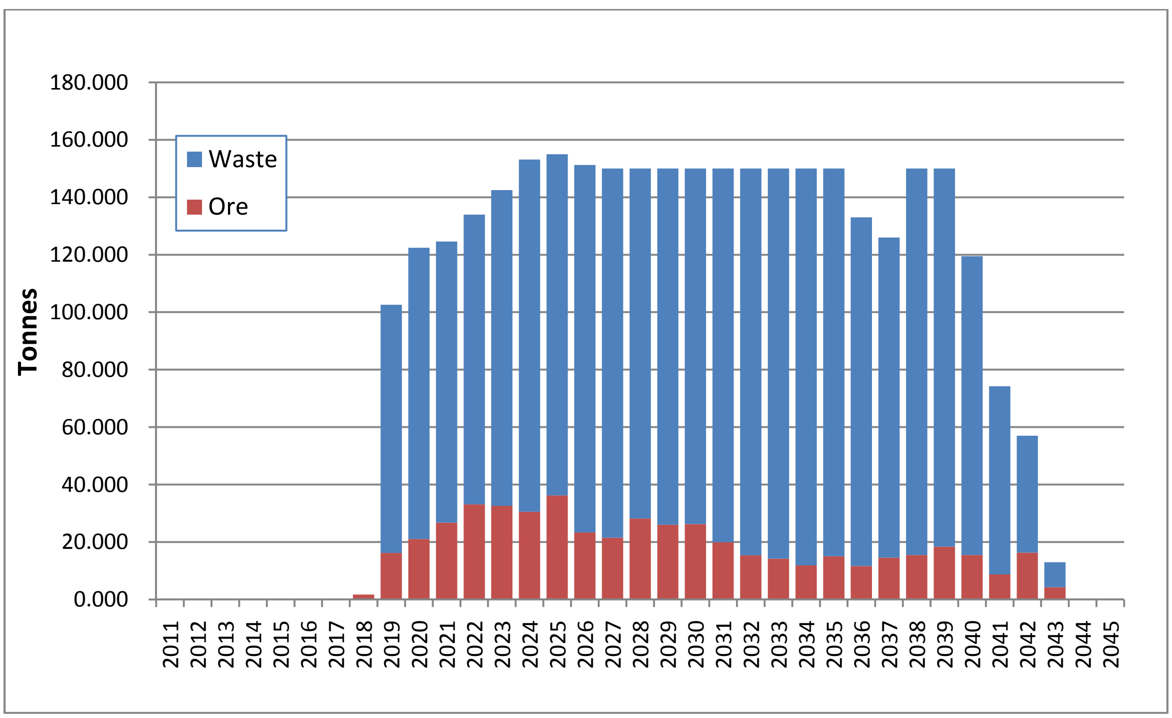


Figure 16-5: Projected Ore and Waste Production Schedule



16.6.3 Mill Feed Plan

The mill feed will constitute ore transported directly from the mine plus ore reclaimed from five stockpiles. The following issues are considered:

- Variable metallurgical recovery by metallurgical domain
- Variable mill throughput by rock type
- Ore degradation applied to metallurgical gold recovery and sulphur grade by guidance of Barrick metallurgical staff. When ore is stockpiled for more than one year:
  - Au recovery drops by 5% (RMAU = RMAU – 5%)
  - S grade drops by 5% (S% = Su% *0.95).

After plant ramp-up, mill feed averages 52.7 kt/d and reaches a maximum of 54.4 kt/d in 2030 (Year 12). Contained gold in the mill feed averages approximately 1.3 Moz per year, while gold production averages 1.6 Moz per year for the first five years, with a maximum of 1.731 Moz in 2024 (Year 6).

Details of the mill schedule, gold recoveries, and projected ounce production on an annualized basis are included in Section 17.6.

In the opinion of Donlin Gold, the proposed plant feed supports that the amount of sulphur in the feed can be controlled through a blending strategy combining ore feed directly from the mine and from stockpiles.

16.7 Ore Stockpiles

The maximum long-term stockpile volume is 104.8 Mt at the end of 2031. This includes 18.5 Mt of high sulphur-grade material, 31.9 Mt of medium sulphur-grade material, and 53.9 Mt of low sulphur-grade material.

A short-term stockpile was assumed with an average 45% annual re-handle.

Locations and planned usages for the stockpiles are included as Figure 16-6 and Figure 16-7 respectively.

Figure 16-6: Proposed Locations, Ore Stockpiles

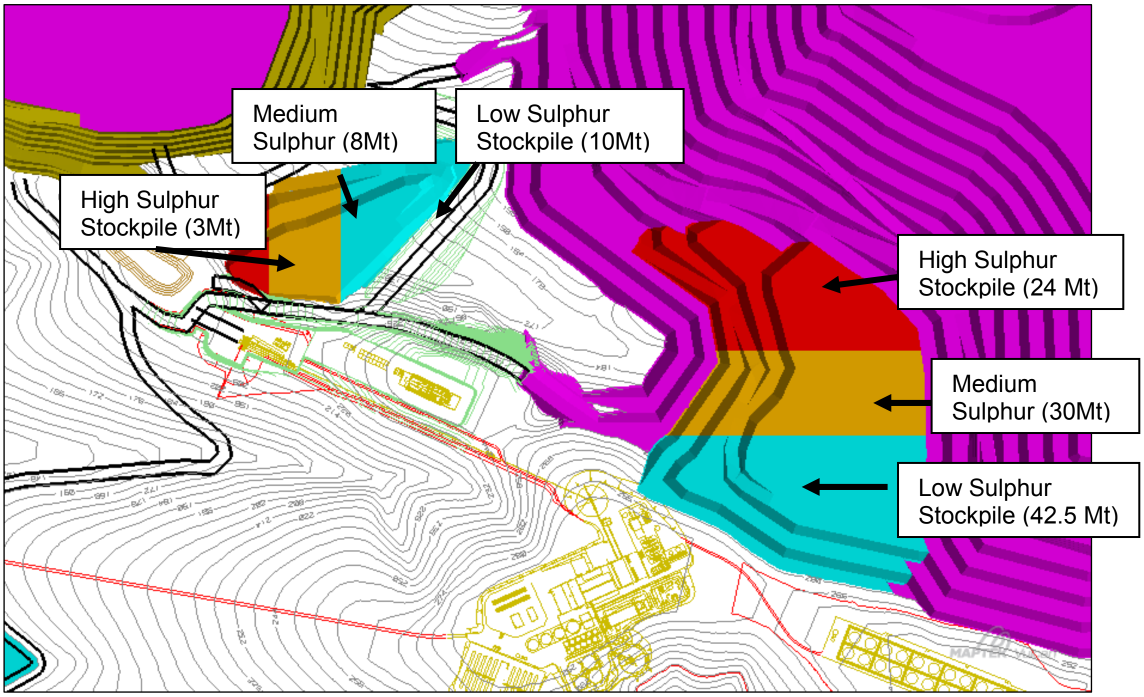
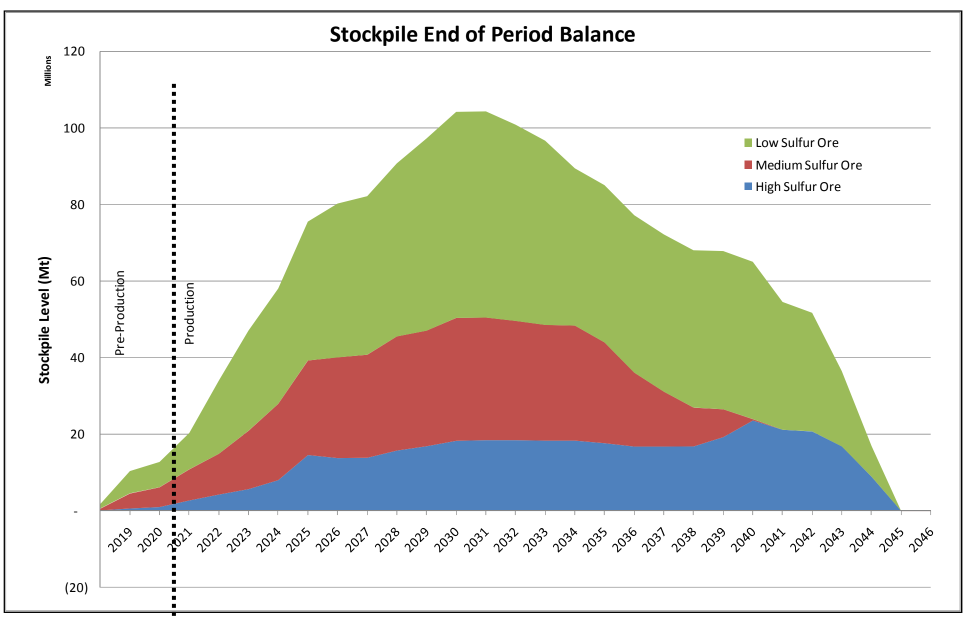


Figure 16-7: Donlin Ore Stockpile Projected Capacity by Year



16.8 Waste Rock Scheduling and NAG/PAG Management

Waste material will consist of overburden, non-acid-generating rock (NAG), and potentially acid generating rock (PAG). The waste will be stored in several dump areas at Donlin (Figure 16-8).

NAG and PAG rock from the ACMA and Lewis pits will be routed to the external waste dump (waste rock facility, WRF) during the first part of the mine life. Later, as mining is completed in the ACMA pit, the remaining waste rock will be routed there. In certain periods, suitable material will be sent to the tailings dam location where it will be used for construction.

Of the total amount of waste rock, 2,232 Mt will be stored in the WRF, and approximately 423 Mt will ultimately be placed in the ACMA pit as backfill. Backfilling will commence in 2035 (Year 18) and continue until the end of mine life. In addition, 103 Mt of waste rock will be used for construction purposes, and 16.6 Mt of overburden will be stored in overburden stockpiles for reclamation purposes.

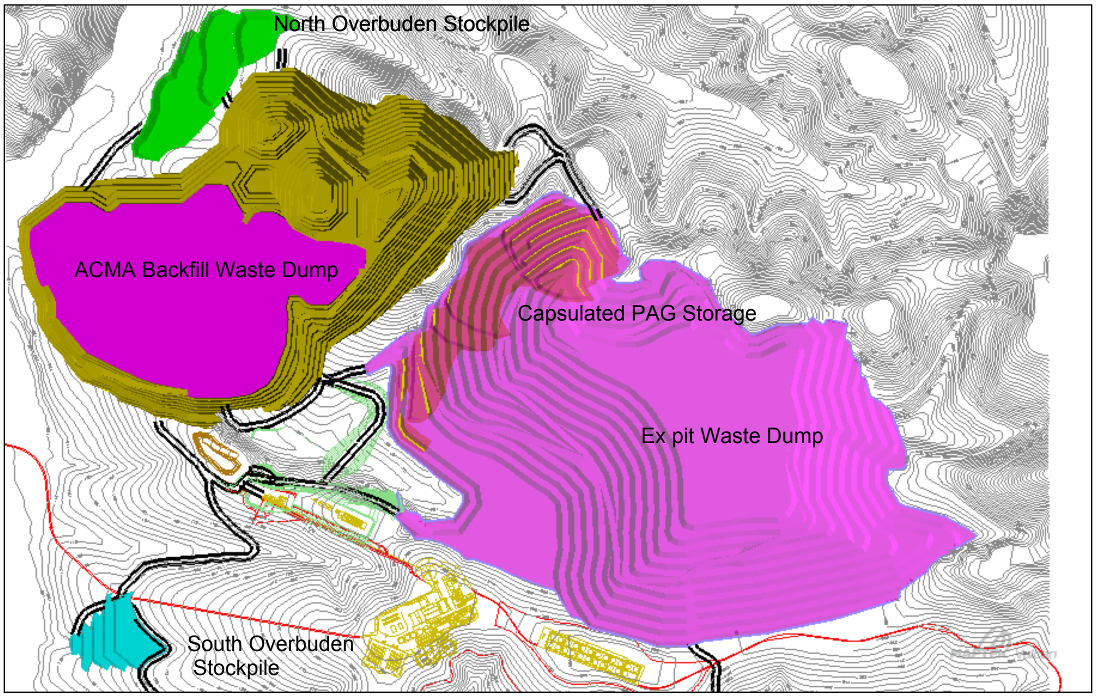
Diversion dam construction is based on WRF construction sequencing; the estimated lifespan of the freshwater diversion dam AMDD is approximately three years, from 2017 to 2020.

The disposal of PAG waste rock will depend on its reactivity category: PAG rock of waste rock management category (WRMC) 7, or PAG-7, 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 Rob's Gulch and Unnamed Gulch sections of 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 will be addressed further during the 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.

Figure 16-8: Waste Dump Locations



16.8.1 Overburden Scheduling and Concurrent Reclamation

Overburden must be pre-stripped from within the pit before production mining begins. The initial stripped materials will be stockpiled for construction and reclamation purposes, or dumped onto the WRF. Materials stripped during the mine production period will continue to be stockpiled or dumped into the WRF, but some will also be used directly for construction or reclamation.

16.9 Water Management and Treatment

Water management will play an important role in the mine development. The ACMA pit area is bisected by American Creek, and the west wall of the ultimate pit will be close to Crooked Creek. Surface ditches, a contact water pond immediately upstream of the pit, plus diversion systems further upstream, will control surface waters in the pit and waste dump areas. Dewatering systems consisting of vertical dewatering wells, horizontal drains, and in-pit sump pumps will be needed to manage groundwater.

To meet the Alaskan criteria for water quality, water from the pit dewatering wells will be treated before release to the environment. The operations water treatment plant (WTP) will have a design capacity of 2,188 USgpm (497 m^3/h). The WTP will be constructed and placed into operation prior to construction of the mill because the dewatering wells need to be placed in service upon commencement of the mine pit development. The WTP process will remove arsenic and manganese by adding ferric sulphate and lime.

16.10 Ore Control

The mining approach chosen for Donlin will require the ability to accurately distinguish the selective mining areas from the bulk mining areas and to accurately predict the actual waste to mineralized material contact. The Ore Control group will be responsible for:

- Managing the RC drilling program
- Sampling and geologic mapping of blastholes and logging of RC drill holes
- Merging assay data with drill holes and blasthole coordinates
- Generating short-range planning block models
- Performing ore and waste delineation, including mine-to-mill reconciliations and quality control.

The Ore Control group will fall under the direction of the Geology Department. Ore Control staffing will include 24-hour coverage by geologists and sampling technicians to assist mine operations with ore–waste decisions. Ore control will rely heavily on digital methods such as high-precision GPS and virtual dig maps because the ore zones are not visually discernable by field personnel. Dark and snowy conditions for much of the year will also add to the reliance on digital techniques.

RC drilling will be the primary method of identifying ore-bearing areas for routing material to the ore stockpiles and crusher. RC drilling has been used extensively in Australia, Africa, and to a lesser extent in other parts of the world to make ore–waste determinations for orebodies that are narrow and steeply dipping. The RC holes at Donlin will be 5½" (140 mm) diameter inclined at 60° to intersect the mineralized structure and drilled to a hole length of 138 ft (42 m). This is within the maximum hole length quoted by the manufacturer of the L8 drill rig. Holes inclined at 60° will provide RC information for three 40 ft (12 m) benches (120 ft or 36 m vertically).

16.11 Blasting and Explosives

A blend of 70% emulsion phase / 30% ANFO will be used for blasting, based on anticipated groundwater conditions.

An explosives supplier will be contracted to provide a "'down-the-hole" blasting service. The supplier will provide the AN, emulsion phase components, and blasting accessories. The supplier will also supply the emulsion plant, explosives magazines, mixing equipment, and delivery trucks. The operator will provide fuel oil and accommodation. Supplier personnel will charge the holes, place the detonators and boosters, and tie in the patterns.

16.12 Mining Equipment

The Donlin FSU2 equipment operating, capital, performance, and specifications were provided by a number of equipment manufacturers. Based the manufacture provided information a total cost of ownership (TCO) analysis was completed to select the mining equipment to support the study. Note, however, that the initial quotes were budgetary only and the final equipment selections will be made via a competitive selection process.

To determine the number of equipment units required for each major fleet, productivities were calculated based on estimated annual operating hours and mechanical availability. Annual operating hours varied by fleet due to associated availabilities. A value of 50 net operating minutes per gross operating hour (GOH) was applied to all equipment to account for time spent on non-primary production tasks. The vendor-estimated mechanical availability of the equipment decreases with hours worked. An average mechanical availability based on the life of the fleet was assigned to replicate the availability for a fleet containing units of mixed ages.

16.12.1 Drilling

Donlin will undertake five different types of drilling:

- RC drilling to provide samples for geological modelling
- Blast pattern drilling to fragment the rock for mining
- Frost drilling to deal with previously blasted material that has become frozen
- Horizontal drain hole drilling to prevent water pressure from building up behind the pit walls
- Vertical dewatering wells. The vertical dewatering well development is a specialized activity and will be performed by a contractor.

To accomplish the five types of drilling, three different types of drill will be used: an Atlas Copco 275 rotary drill for bulk waste with 9⅞" (251 mm) diameter holes for 40 ft

(12 m) benches; an Atlas Copco DML rotary drill with 7⅞" (200 mm) diameter holes for ore and waste in 40 ft (12 m) benches; and an Atlas CopcoL8 hammer drill with 5½" (140 mm) diameter holes for ore and waste in the 20 ft (6 m) benches and for pre-split and RC drilling.

16.12.2 Loading

Primary loading will be performed by Komatsu PC8000 electric-hydraulic shovels with a 50 yd^3 (37 m^3) bucket. One LeTourneau L-2350 53 yd^3 (40 m^3) front-end loader will be used for secondary production, and another for backup production, cleanup, and stockpile rehandling.

16.12.3 Hauling

Large 400 st (360 t) payload Liebherr T282C haul trucks will be used for primary mine production. A maximum of 69 Liebherr T282C trucks are required.

16.12.4 Secondary Fleet

A second fleet of mining equipment will be required for overburden management, concurrent reclamation, snow removal, road maintenance, and special projects. This fleet will consist of smaller, more agile 25.0 yd^3 (18.1 m^3) FELs and 150 st (135 t) trucks and will allow the primary fleet to focus on production ore and waste mining.

16.12.5 Support Equipment

The major tasks to be completed by the support equipment include the following:

- Bench and road maintenance
- Reclamation support
- Stockpile construction
- General maintenance
- Ditch preparation and maintenance
- Tailings dam support
- Shovel support/cleanup.

Additional auxiliary equipment will serve and support the mine operations and maintenance groups. Equipment selection by function is included in Table 16-2 (drilling equipment), Table 16-3 (shovel fleet), Table 16-4 (mine support equipment), and Table 16-5 (auxiliary equipment). Mine equipment requirements over the life-of-mine are included as Table 16-6.

16.12.6 Maintenance Considerations

For the first two years, all mobile fleets will be maintained by contractors under Maintenance and Repair Contract (MARC) agreements. It is anticipated that each fleet will have cost and availability guarantees as well as associated warranties. The Owner will be able to minimize risk of maintenance cost increases due to either equipment reliability or labour shortages.

After 15 months, the Owner will assume more risk but will reduce costs by phasing out the MARC contracts and taking responsibility for all mobile equipment maintenance.

Maintenance on the large drills and excavators will be performed in the field, whereas equipment that can be easily driven or towed will be serviced in the truckshop. The truckshop will be near the long-term ore stockpile, fuel tank farm, and haul road to provide easy access for haul truck maintenance and refuelling.

16.12.7 Health and Safety Considerations

Except for the electric shovels, the main equipment fleet will be diesel-powered. All equipment will be maintained in good working order so as to minimize CO_2 emissions. All main equipment is designed to meet OSHA and MSHA occupational noise criteria.

Routine water spraying by three 30,000 USgal (114 kL) water trucks will suppress dust generated on roads, benches, and dump areas. Non-chloride dust suppressants may be applied in high traffic areas if necessary.

All water captured in the mine area will be diverted to the contact water pond or tailings impoundment by a system of drains and pumps. Any contained sediments or residuals from blasting products will be captured in this system. All ex-pit mining areas will be designed to minimize surface erosion.

Table 16-2: Annual Required Drill Fleet

Year	Atlas Copco PV 275	Atlas Copco DML	Atlas Copco L8
	Fleet Size #	Fleet Size #	Fleet Size #
2018	1	2	2
2019	4	11	10
2020	4	11	10
2021	4	11	10
2022	4	12	10
2023	4	12	10
2024	4	12	10
2025	4	13	10
2026	4	13	10
2027	4	13	10
2028	4	13	7
2029	4	14	6
2030	4	14	5
2031	4	15	5
2032	4	14	5
2033	6	14	5
2034	6	14	5
2035	6	14	5
2036	6	14	5
2037	5	14	5
2038	5	14	5
2039	4	14	5
2040	4	11	5
2041	2	7	3
2042	1	7	3
2043	1	2	1

Table 16-3: Annual Shovel Fleet Required

Year	Komatsu PC8000E	Komatsu PC8000D	LeTourneau L2350	Cat 994F
	Fleet Size #	Fleet Size #	Fleet Size #	Fleet Size #
2018		1	1	1
2019	4	1	2	1
2020	4	1	2	1
2021	4	1	2	1
2022	4	1	2	1
2023	5	1	2	1
2024	6	1	2	1
2025	6	1	2	1
2026	6	1	2	1
2027	6	1	2	1
2028	6	1	2	1
2029	6	1	2	1
2030	6	1	2	1
2031	6	1	2	1
2032	6	1	2	1
2033	6	1	2	1
2034	6	1	2	1
2035	6	1	3	1
2036	6	1	2	1
2037	6	1	2	1
2038	6	1	2	1
2039	6	1	2	1
2040	5	1	2	1
2041	3	1	2	1
2042	2	1	2	1
2043	1		2	1
2044			2	
2045			2	

Table 16-4: Mine Support Equipment

Equipment Unit	Size hp (kW)	Number
Cat D11R Track Dozer	850 (633.8)	6
Cat D10T Track Dozer	580 (432.5)	4
Cat 854G Wheel Dozer	800 (596.6)	6
Cat 24M Grader	533 (397.5)	3
Cat 16M Grader	297 (221.5)	7
Cat 390DL Hydraulic Excavator (5 m^3 Bucket)	513 (382.5)	2
Cat 785D Water Truck	1,450 (1,081)	4
Komatsu PC2000 Backhoe	956 (713)	2

Table 16-5: Mine Auxiliary Equipment

Equipment Unit	Number
Horizontal Drain Hole Drill	1
Cat 345CL with Hydraulic Hammer	1
220 Ton (200 tonne) Class Crane	1
165 Ton (150 tonne) Class Crane	1
66 Ton (60 tonne) Class Crane	1
26 Ton (23.5 tonne) Forklift	1
18 Ton (16.3 tonne) Forklift	1
5 Ton (4.5 tonne) Telehandler	1
5 Ton (4.5 tonne) Forklift	1
Fuel/Lube Truck, CAT 777	3
Medium Mech Truck	4
Large Mech Truck	1
Tire Handler	2
SVE Lift Truck	1
CAT 966H Loader	1
Lowboy/Tow, Cat 793	1
Operations Field Truck	1
Soil Compactor	1
Backhoe Loader	1
Light Plant	20
Skid Steer Trailer	3
Light Vehicle	35
Crew Bus	5
Cable Reeler	1
Shovel Motivator	1
Backhoe PC 2000	2

Table 16-6: Mine Equipment Requirements

	LOM Max	P-P	Prod.	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Installed Fleet																								
Komatsu PC8000 Shovel Electric	6		6		4	4	4	4	5	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Komatsu PC8000 Shovel Diesel	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
LeTourneau L2350 Loader	3	1	3	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	2	2	2
Cat 994F Front-End Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Liebherr T282B Haul Truck	69	9	69	9	28	33	35	35	40	45	47	55	60	60	60	60	60	60	65	69	69	69	69	69
Cat 785C Haul Truck	10	10	10	10	10	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Atlas Copco PV 275	6	1	6	1	4	4	4	4	4	4	4	4	4	4	4	4	4	4	6	6	6	6	5	5
Atlas Copco DML	15	2	15	2	11	11	11	12	12	12	13	13	13	13	14	14	15	14	14	14	14	14	14	14
Atlas Copco L8	10	2	10	2	10	10	10	10	10	10	10	10	10	7	6	5	5	5	5	5	5	5	5	5
Cat D11T Track Dozer	6	4	6	4	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Cat D10T Track Dozer	4	3	4	3	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Cat 854G Wheel Dozer	6	4	6	4	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Cat 24H Grader	3	1	3	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3	3
Cat 16H Grader	7	3	7	3	5	5	5	5	5	5	5	6	6	6	6	6	6	6	7	7	7	7	7	7
Cat 785C Water Truck	4	1	4	1	3	3	3	3	3	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Cat 390DL Excavator	2	1	2	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	1
Komatsu PC2000 Excavator	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Fuel Truck	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2
Service Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mobile Crane	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Low Boy Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tire Handler	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Light Plant	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	10

Used oil will be collected at the truckshop and temporarily stored on site. Some of the waste oil will be used as a constituent of the fuel component in the ANFO mix; the surplus oil will be burned to generate heat in the mine facilities. All water collection facilities surrounding the shops and fuel islands will be routed to oil / water separators for oil removal.

Any oil spills on the mine site will be dealt with by excavating the contaminated material and moving it to a remedial land farm where the soil can be cleaned under controlled conditions.

16.12.8 Communications Considerations

The mining department at Donlin will use GPS machine guidance and a fleet management system to guide and control the mining operation on a near real-time basis. Fleet management is the assignment of equipment to mining tasks, while GPS machine guidance assists the operator with respect to spatial positioning of the ground-engaging tools.

16.13 Consumables

Figure 16-9 summarizes the mine consumables by function over the life-of-mine. The major consumables cost drivers are fuel, tires, and maintenance parts and repair.

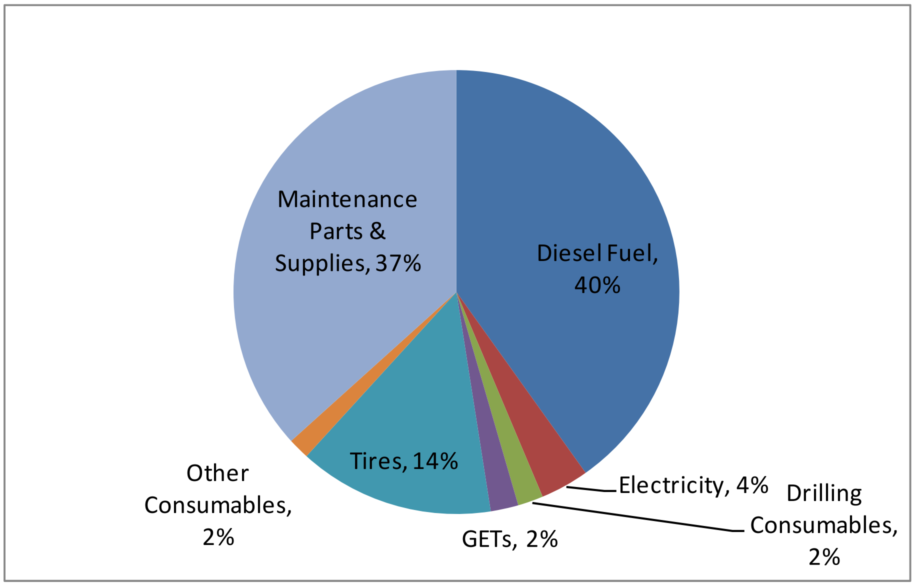
Fuel, electricity, and explosives cost estimates for the period from 2018 through 2045 are:

- Fuel price: $3.03/USgal ($0.804/L)
- Electricity price: $0.119/kWh
- Explosives emulsion price: $963/t.

16.14 Work Schedule

The work schedule assumes mine production will operate 24 h/d, 7 d/wk, 365 d/a. Operations and mining personnel will work on two 12 h/d shifts. All hourly (non-exempt) personnel assigned to the mine or port sites will work a 2-weeks-in/1-week-out rotation, while all salaried (exempt) personnel will work a 12 h/d shift on an 8-days-in/6-days-out rotation, a 2-weeks-in/1-week-out rotation, or a 2-weeks-in/2-weeks-out rotation.

Figure 16-9: Mine Equipment Consumables Distribution



16.15 Comments on Section 16

In the opinion of the QPs, the following conclusions and interpretations are appropriate:

- The proposed Project will be a conventional, large-tonnage, open-pit operation designed to developed for a nominal mill throughput of 21.5 Mst/a (19.5 Mt/a), or 59,000 stpd (53,500 t/d). The operating mine life is estimated Te 25 years based on the processing rate envisioned
- The mine design, complete with haulage access, includes 556,459 kst (504,811 kt) of ore containing 33,849 koz (1,052,815 kg) of in-situ gold and has a strip ratio of 5.48. The mine design is considered appropriate to the quantity of Measured and Indicated Mineral Resources estimated for the Project
- The mine plan developed for Donlin envisages mining nine pit phases at ACMA, and six pit phases at Lewis
- Mineable pit phases were designed based on optimized nested pit shell guidance, gold grade, strip ratio, access, and backfilling of the ACMA phases. Ramps in final walls have a design width of 131 ft (40 m) and a gradient of 10%. A nominal minimum mining width of 492 ft (150 m) was used for phase design
- The engineered pit design includes approximately 5% less ore tonnage and 7% fewer Au ounces than the pit optimization shell it was based on. This is at the

upper end of the generally accepted limit of a 10% reduction in ore tonnes. As such, there is a risk that the engineered pit design contains less ore than optimum

- The SMUman process used to dilute the block model adds diluting waste blocks to the mineral inventory. Globally, all of these blocks are processed over the life of mine; however, a higher degree of selectivity is assumed during periods when elevated cut-off grades are applied (i.e., diluting waste blocks are sent to stockpile and processed later in the mine life). This risk has been mitigated by applying the DILman process, which groups high-grade ore into selectable blocks during periods when the elevated cut-off grades are applied
- Donlin will be mined by a combination of bulk and selective mining. The SMU block size of 6 m x 6 m x 6 m reflects the selectivity of the proposed open pit mine milling rate. The bench height will be either 6 m or 12 m, depending on mining selectivity requirements. Although split-bench mining has been tested successfully at other operations, it is not common practice. The risk associated with split-bench mining is that operations may choose to abandon the practice, potentially increasing dilution
- The base mining cost (before incremental mining cost with depth) is \$1.51/st (\$1.668/t), the average processing cost is \$13.06/st (\$14.39/t), the G&A cost is \$2.06/st (\$2.27/t) and the stockpile rehandle cost is \$0.19/t processed. These costs are considered reasonable
- The processing rate is variable from period to period as a function of sulphur grade and ore hardness. To maximize overall 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
- Short-term ore stockpile rehandle to achieve mill feed sulphur grade targets was assumed for this study to be 45%, but this could potentially be up to 100% (worst case). In the 2007 feasibility study, the assumed rehandle level was only 25%. The new estimate is based on short-term planning simulations and expectations of fleet interactions
- Mining will be carried out using a mixed shovel fleet and trucks. Mining equipment requirements were based on the mine production schedule and equipment productivities, and included consideration of workforce and operating hours. The fleet is appropriate to the planned production schedule
- Mine design has incorporated geotechnical and hydrogeological considerations. Geotechnical information was collected by BGC, who provided the slope design parameters for the open pit

- Surface ditches, a contact water pond (CWP) immediately upstream of the pit, plus diversion systems further upstream, will control surface waters in the pit and waste dump areas. Dewatering systems consisting of perimeter and in-pit vertical dewatering wells, horizontal drains, and in-pit sump pumps will be required to manage groundwater
- A total of 2,460 Mst (2,232 Mt) of waste will be stored in a single ex-pit waste rock facility, in the American Creek Valley, east of the pit area. Another 466 Mst (423 Mt) of waste rock will be stored in the ACMA backfill dump and 18.7 Mst (17 Mt) of overburden in the overburden stockpiles for reclamation use. The remaining 114 Mst (103 Mt) is used as construction material, of which 99 Mst (90 Mt) is for tailings dam wall construction
- The waste rock is characterized by its potential for acid generation and has been assigned reactivity categories. Categories 1 to 4 are non-acid-generating (NAG), and 5 to 7 are potentially acid generating (PAG). 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.

AMEC notes that the actual mining costs are 17% higher than the optimization cost of $2.14/t. At a 20% increase in mining costs, the sensitivity work has indicated that about 3 M ounces will be lost. As a consequence, AMEC notes that any increases in mining costs will be reflected in a decrease of produced ounces.

17.0 RECOVERY METHODS

17.1 Plant Design

The process design criteria are shown in Appendix A. The criteria have been updated as appropriate to reflect the revised mine plan and more complete design in the areas of mercury and cyanide handling.

The numbers shown in the criteria below indicate conditions of a plant operating at full capacity. They are appropriate for plant design, but they do not take into account variation introduced during facility start-up, or tapering off of production near the end of mine life. As the design criteria is not based on mine schedule, it is not appropriate to use for overall metallurgical accounting or the financial plan.

Conversely, the values used in the financial models were generated based upon mine plan projections as part of the operating cost compilation. These operating costs were calculated separately from the design criteria document.

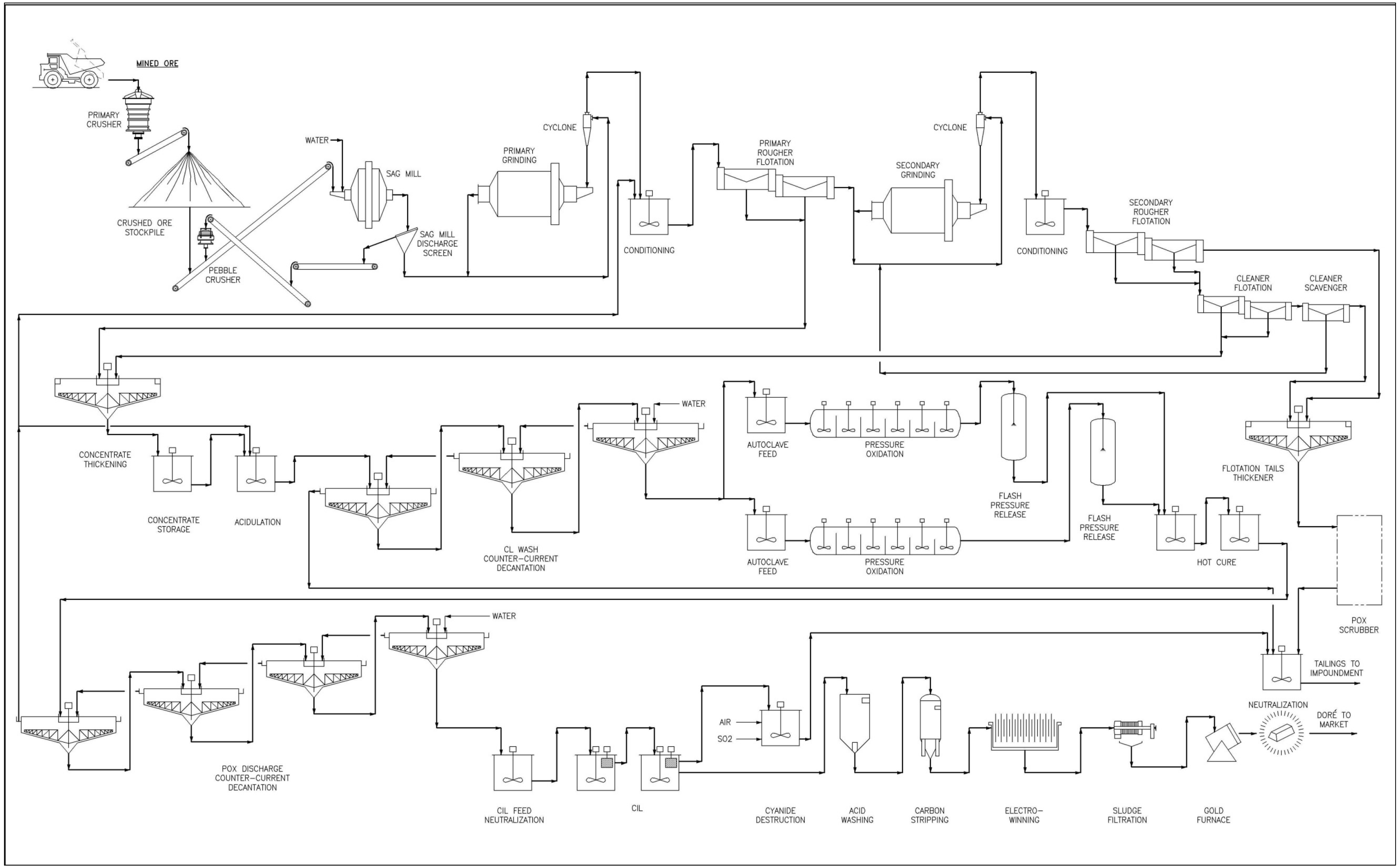
17.1.1 General

The process is based on conventional technology. The concentrator, pressure oxidation, and cyanidation facilities will be at the forefront of technology for large, modern gold processing plants. A simplified block flowsheet of the overall process is shown in Figure 17-1. All process equipment, with the exception of thickening, neutralization tankage and concentrate storage tankage, will be enclosed in buildings. Installed standby pump spares are provided for all critical process streams.

17.1.2 Crushing and Coarse Ore Stockpile

Mine haul trucks (with capacities to 380 st (345 t)) will dump run-of-mine (ROM) open-pit ore directly into dump hoppers ahead of a 60" x 89" gyratory crusher. The maximum design crushing capacity of the crusher is 5,100 st/h (4,630 t/h) at an open-side setting of 6" (160 mm) producing a crusher product at P80 5" (125 mm). The discharge of the crusher will be to a covered coarse ore stockpile with a live capacity of 42,000 st (38,000 t) of ore, representing 16 hours of process plant operation, and a total capacity of approximately 192,000 tons (174,000 t), representing 3.2 days of process plant operation.

Figure 17-1: Donlin Gold Project Process Plant Block Flow Diagram



The reclaim tunnel with feeders will be used to reclaim material from the stockpile discharging onto the SAG mill feed conveyor. The normal SAG mill feed rate will be 2,642 st/h (2,397 t/h). SAG mill critical size material will report to the pebble crusher. The discharge from the pebble crusher will join the new feed from the coarse ore stockpile.

17.1.3 Grinding and Pebble Crushing

The overall grinding configuration will consist of an open-circuit SAG mill followed by the “mill-chemical-float-mill-chemical-float” (MCF2) circuit. The MCF2 circuit will entail a primary ball mill followed by primary rougher flotation; the tailings produced from primary flotation will be 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.

SAG mill discharge will be screened, and oversized pebbles are conveyed to two large cone crushers. Crushed pebbles will normally be returned to the SAG mill. Typical SAG mill discharge will have a P_{80} of 1,700 µm. After primary ball mill grinding, the P_{80} is anticipated to be 121 µm; after secondary grinding, the P_{80} is anticipated to be 50 µm. The total system throughput is expected to average 59,000 st/d (53,500 t/d) at 93% availability.

The SAG mill feed conveyor will discharge into the SAG mill feed chute and then into the SAG mill. Process solution will be added at this point to flush the ore into the mill and provide the correct dilution for grinding. Copper sulphate will be added to the feed end of the SAG mill to activate sulphide mineralization. The SAG mill will discharge into the primary cyclone feed pumpbox. The SAG mill will be powered by a 26,800 hp (20 MW) wrap-around variable-speed drive. SAG mill discharge will be screened with undersize from the trommel screen and vibrating screen will collect in the SAG mill discharge launder.

Optimum economics are expected to be achieved by using the fewest, largest equipment units available. The ball mills will be trunnion-supported units with 24,000 hp (18 MW) wrap-around drive configurations. Discharge from the primary ball mill will exit the discharge trunion into a trommel screen attached to the ball mill. Oversize material will 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 primary cyclone feed pump will be variable-speed and will transport slurry to the cyclone cluster, which will classify particles by size to return coarse particles to the ball mill for further size reduction.

The primary cyclone overflow will be designed to operate at 40% solids, with an anticipated average 80% passing particle size of 121 µm; the cyclone circulating load is estimated to be 210%. The fresh feed for the secondary ball mill will be a combination of the slurry from the 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 will 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, as undersize material will pass through the trommel screen and into the secondary cyclone feed pumpbox, together with the rougher tailings and the cleaner scavenger concentrate. The secondary cyclone feed pump (variable-speed) will transport slurry to the secondary cyclone clusters. The secondary cyclone overflow will be anticipated to be 27.2% solids with an average 80% passing particle size of 50 µm; the cyclone circulating load is estimated at 210%.

17.1.4 Flotation

Donlin ore contains a mixture of intrusive and sedimentary rock containing sulphide mineralization. The optimum gold recovery will be achieved by maximizing sulphide recovery. Producing a bulk flotation concentrate in two rougher flotation steps with relatively low selectivity and high mass pulls has been determined to provide the best results. 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 will be sent to the secondary grinding cyclone feed pumpbox, and the tails will be mixed with rougher tailings flow by gravity to the flotation tailings thickener. The primary rougher concentrate and the concentrate produced from the cleaner flotation will report to the concentrate thickener.

Overflow slurry from the primary grinding circuit cyclone will be fed through a flotation safety screen before entering conditioning tank No. 1. Oversized material will be sent to a hopper, and undersize material will pass through to conditioning tank No. 1.

Acidic solution from the POX CCD washing circuit, as well as flotation process water and copper sulphate, will be added to the first conditioning tank. The slurry then will then pass to conditioning tank No. 2 where potassium amyl xanthate (PAX), dispersant Cytec E-40, and methyl isobutyl carbinol (MIBC) frother will be added. The discharge from the second conditioning tank will be pumped to a distributor box, which will split the feed to two parallel rows of 10,600 ft^3 (300 m^3) primary rougher cells. The rougher flotation feed pump system will include an installed spare. Each bank of cells will have 11 individual units and provide 57 minutes of residence time.

Primary rougher concentrate from the rougher portion will 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 flotation circuit will be fed from the secondary rougher conditioning tank where copper sulphate and MIBC will be added. The flow will be divided into two trains; each train will have 11 individual 10,600 ft^3 (300 m^3) secondary rougher cells providing a total residence time of 57 minutes. Additional MIBC, PAX, Cytec E-40, soda ash, and a second frother, F-549, will be added throughout the secondary rougher trains.

Secondary rougher concentrate will be sent to the cleaner flotation cells bank. Secondary rougher tailings will be sent directly to the flotation tailings thickener.

The secondary flotation concentrate will be cleaned in a bank of six 10,600 ft^3 (300 m^3) tank-type flotation cells, with a bank residence time of 100 minutes. The concentrate will be sent to the cleaner concentrate pumpbox, from where it will be pumped to the concentrate thickener. The cleaner tailings will flow by gravity to the cleaner scavenger flotation cell bank.

The cleaner scavenger bank will have four 10,600 ft^3 (300 m^3) cells with a combined residence time of 150 minutes. The cleaner scavenger concentrate will be sent to the secondary grinding cyclone feed pumpbox. The tailings will be sent to the flotation tails thickener by gravity. Flotation streams will be sampled automatically for metallurgical accounting and control purposes

An on-stream X-ray fluorescent analyzer (on Fe and As) will provide continuous data to enable operators and supervisory control systems to optimize flotation and to respond to upset conditions.

A slip stream from each of the two cyclone overflow streams will be passed through a particle size analyzer to provide information for grinding control.

17.1.5 Thickening, Concentrate Storage, Acidulation, and CCD Washing

Concentrate from flotation will pass to a 11.5 ft diameter x 16.4 ft high (3.5 x 5 m) concentrate thickener de-aeration tank and then to a 148 ft diameter (45 m) concentrate thickener. Thickener overflow will be returned to the grinding and flotation areas as process water while underflow at an estimated 46% solids will be pumped to the concentrate storage tank circuit. A total of 36 hours of concentrate storage will be provided. In normal operation material will be withdrawn from this circuit and sent to the acidulation circuit. A bypass line will be provided to pump slurry directly from thickening to acidulation, where acidic solution recovered from the pressure oxidation (POX) counter-current decantation (CCD) wash circuit will be mixed with the concentrate with the aim of consuming 85% to 100% of the carbonate gangue component of the concentrate.

The acidulated material will be washed in a three-thickener CCD circuit that will displace the solution with raw water to reduce the overall levels of soluble mineral species reporting to the POX circuit in the slurry.

17.1.6 Autoclave Plant

Acidulated feed slurry will be stored in one agitated autoclave feed storage tank adjacent to the POX area. This tank will provide the autoclave plant with a continuous feed unaffected by short-term upstream throughput variations. When full, it will allow the autoclave circuit to continue operating for four hours when upstream equipment is not operating. Slurry will be transferred from the feed tank by two parallel lines, each equipped with a slurry heater feed pump. Slurry will be transferred to the heater vessels that will pre-heat the incoming slurry to varying temperatures, depending on the sulphide sulphur grade of the feed material. Pre-heat temperature will be optimized based on maintaining autogenous conditions in the autoclave while minimizing cooling water addition. Slurry pre-heating will be accomplished with the use of flash steam produced in the pressure letdown flash vessels on the autoclave discharge. Each autoclave train will discharge to two flash vessels and two vent gas cyclones via parallel slurry discharge lines. Slurry heater discharge temperature control will be achieved by bypassing a portion of the feed to each heater to the heater sump. From each heater discharge, a single feed line will feed each autoclave, each with a charge pump to create sufficient pressure for the suction side of the autoclave slurry feed pump; a slurry strainer to remove any scale or oversize particles that could damage the autoclave slurry feed pumps; and a high-pressure piston-diaphragm feed pump with suction accumulator and discharge dampener. The autoclave will be supplied with high-pressure oxygen gas, high-pressure cooling water, and high-pressure steam. Oxygen will be produced at an on-site air separation plant. Cooling water will be distributed to the autoclave from a cooling water tank located locally in the POX area. High-pressure, horizontal multi-stage centrifugal pumps will supply the cooling water to all compartments of the autoclave. A common piping system (and spargers) will be utilized for both the oxygen and high-pressure steam to the autoclave.

High-pressure steam (produced from de-mineralized water) will not be required for normal operation, but is required for autoclave heat-up. De-mineralized water will also be used for the agitator seal water system and the oxygen plant boiler system. The autoclave building will be serviced by an overhead crane, allowing any of the agitators to be removed without need for disc disassembly (impeller blades require removal).

Each autoclave will discharge into two flash vessels in parallel. Autoclave discharge slurry will be depressurized to near-atmospheric pressure, generating flash steam in the process. Flash vessel underflows will be directed by gravity to an oxidized slurry seal tank. Slurry from this tank will be transferred by gravity to the downstream hot cure tanks. Steam generated at the hot cure tanks will be condensed using a spray tree condenser vessel. Vent-gas from the autoclave will be passed into the vent gas quench vessel. Flotation tails will be used as quenching medium, as the steam will be condensed across a baffle arrangement inside the vessel. The quench vessel will reduce the temperature of the vent-gas and the quantity of steam (through condensation) that will be fed to downstream equipment. The quench vessel will also facilitate additional removal of carryover from the autoclave and slurry heater, and pre-heat the flotation tails ahead of the downstream neutralization process, thereby improving kinetics. Vent gas from the quench vessel will be piped to a secondary spray tree condenser vessel where raw water will further cool the gas and steam condenses. The gas will then pass through a venturi scrubber where the gas will be further cleaned of particulates by pressure drop and the addition of fresh water.

Gas will exit the venturi scrubber saturated with water vapour and at a temperature of 150°F (40°C). The process off-gas temperature will be reduced to a target of 40°F (4°C). The gas volume will also be reduced by water vapour condensation. The wet-gas condenser will promote the condensation of elemental mercury during periods of upset conditions with higher than normal gaseous mercury levels. The overall mercury loading on the downsteam carbon adsorption process will be reduced as a result. Gas will then enter a wet gas coalescer to drop out any remaining entrained mercury. Gas will be first subjected to cyclonic separation and then will enter a coalescer prior to the carbon pre-cleaning section.

The combined gas will enter a pre-cleaning carbon bed. The relatively inexpensive activated carbon contained in the pre-cleaning bed will be used to remove volatile organic compounds (VOCs). Exiting gases will immediately enter the mercury removal carbon bed, which will contain sulphur-impregnated carbon specifically designed to adsorb vapour-phase mercury. Oxidized and particulate forms of mercury will also be collected. Cleaned gas will be discharged to atmosphere. The mercury-loaded carbon will be removed periodically in an environmentally safe manner and sent off site for disposal. A standby series of carbon beds will be available so that neither production nor mercury removal will be interrupted when a bed will be taken off line for carbon change-out or maintenance.

A common mercury collection tank will receive all the condensate and scrubber water from the pressure oxidation gas handling systems. The tank will be designed to settle mercury and separated solids removed from the gas, clarifying the water. The clarified water will be passed through coalescing filters to remove any remaining elemental mercury prior to being recycled to the chloride wash circuit. Coalesced elemental mercury will be returned to the mercury collection tank for settling. The resulting sludge will then be drained from the mercury collection tank as contaminated waste.

17.1.7 CCD POX Thickening and Washing

Slurry flow from the POX circuit will be washed in a four-thickener CCD circuit. Reclaim water will be 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 will be pumped to the CIL solids neutralization circuit. Thickener overflow will be treated 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 flotability. The remainder will report to neutralization. Clarifier sludge will be intermittently returned to the first thickener in the circuit.

17.1.8 Flotation Tailings (FT) Neutralization

In this circuit, flotation tailing will be pre-heated to 131°F (55°C) through the autoclave quench vessel and then combined with the excess diluted acidic wash liquor from the chloride CCD wash circuit in a series of large aerated and agitated tanks. The flotation tailings will act as neutralizing material (source of natural carbonates) for reaction with the acidic liquor. All neutralization tanks will be insulated for heat conservation.

Flotation tailings will be collected, sampled, and passed to the flotation tailings thickener. Thickener overflow will be pumped to the flotation process water tank. Thickener underflow will be pumped to the POX circuit autoclave scrubber.

Acidic solution from the POX CCD wash and spent acid from the elution circuit will be combined with autoclave quench tails in the solution neutralization circuit. The circuit will consist of five mechanically-agitated and aerated tanks in series. Enough reaction time will be provided in the front-end portion of the circuit to bring the pH of the solution up to five, utilizing the quench tails. Tailings from the cyanide destruction circuit will be introduced into a lime neutralization tank where lime will be added in the presence of air to bring the pH to seven. This material will then flow by gravity to the final tailings pumpbox. Owing to their size, the FT neutralization tanks will be installed outdoors.

17.1.9 Solids CIL Neutralization

Underflow from the final POX CCD wash circuit thickener will be neutralized in the solids CIL neutralization circuit. The circuit will consist of two mechanically-agitated tanks where lime will be added to the slurry in the presence of oxygen to bring the pH of the slurry to approximately 11.0. This material will then be pumped to the CIL circuit.

17.1.10 Carbon-in-Leach Cyanidation Circuit

A nominal tonnage of 402 st/h (365 t/h) at 35% solids will be pumped to the first of six CIL tanks. The slurry, with a retention time of four hours per tank, will flow by gravity through each of the six tanks, ultimately reporting to the cyanide destruction reactor tank.

The slurry will flow by gravity from the sixth, or final, CIL tank to the safety screen feed distribution box. A distribution box will direct the slurry to either one or both of two carbon safety screens. The safety screens will prevent carbon that passes through the screen in the last CIL tank from leaving the circuit to the tailings impoundment. Screen undersize will flow by gravity to the cyanide destruction reactor tank and will then be pumped to flotation tailings neutralization tank No. 5 by centrifugal slurry pumps. Sodium cyanide solution will be pumped to the CIL circuit for cyanide leaching of gold. 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 11.0.

Oxygen required for cyanide leaching will be supplied as pure oxygen from the oxygen plant.

The CIL tanks and cyanide destruction reactor tank will be covered to contain any HCN gas that evolves during cyanide leaching. The tanks will be ventilated and the gas is passed to an HCN scrubber caustic solution to recycle HCN.

A carbon concentration of 15 g/L will be maintained in each of the CIL tanks. Each tank will be equipped with a carbon retention screen to allow the slurry to pass through to the next tank and will prevent the carbon from passing through. A vertical carbon advance pump with a recessed impeller installed in each tank will pump the carbon upstream, counter-current to the flow of slurry, at a rate of 25.3 st/d (23 t/d). Barren carbon from the carbon transfer tank in the carbon stripping and regeneration circuit will be pumped to the sixth CIL tank to replace the carbon pumped upstream. Loaded carbon will be pumped from CIL tank No. 1 over the loaded carbon screen. The screen oversize will be washed carbon and will report to one of the two carbon acid-wash vessels.

17.1.11 Cyanide Destruction System

Slurry from CIL tank 6 will flow by gravity through the carbon safety screens, and the screen undersize will report to the cyanide destruction reactor tank. This will be a covered, agitated tank where the residual weakly acid-dissociable (WAD) cyanide concentration will be reduced from nominally 100 ppm to the cyanide levels required by permit. Air and SO_2 will combine to oxidize the cyanide to carbon dioxide and ammonia. Copper sulphate solution will be added to top of the tank and will serve as a reaction catalyst to maintain the reaction kinetics. Lime will be added as necessary to maintain an approximate pH level to ensure adequate reaction kinetics. The destruction reactor tank will be sized for one hour of retention time.

17.1.12 Carbon Elution, Electrowinning, Reactivation, and Gold Refining

Loaded carbon, at a nominal gold loading of 140 oz/st (4,800 g/t), will report to one of two 13.2 st (12 t) capacity carbon acid-wash vessels by gravity from the loaded carbon screen. The two acid wash vessels will process 39.6 st/d (36 t/d) of carbon. After the acid wash and neutralization processes are complete, the carbon will be pumped from the acid wash vessel to one of two strip vessels. A carbon strip will begin as soon as the carbon is transferred to the strip vessel and the transport water has completely drained out of the vessel. For ease of operation, the two strip vessels will be the same size as the two acid wash vessels. Barren solution, at a concentration of 1% NaOH and 0.1% NaCN, will be pumped through the bottom of the strip vessel. The pregnant solution will exit the strip vessel and flow through the heat exchanger before reporting to the pregnant tank. The barren solution will be pumped through the strip vessel for a nominal eight hours to complete each strip. When the strip is complete one bed volume of raw water will be pumped through the strip vessel. This solution will rinse the residual solution from the carbon and cool the carbon in preparation for transfer.

After the carbon is rinsed, it will be pumped to the carbon dewatering screen before the kiln. Carbon will be processed through the kiln at the rate of 1.65 st/h (1.5 t/h) for reactivation. The kiln will be sized to process 100% of the carbon stripped to maintain high carbon activity levels throughout the carbon circuit.

The pregnant solution will be pumped through two parallel trains of two electrowinning cells at a nominal flow rate of 211 USgpm (48 m^3/h). On exiting the cells, the solution will report 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 the cell. The cathodes will either be washed in place or removed to a wash tank and 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 for return 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 75 lb (34.5 kg) flasks, which will be 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 will be mixed with the sludge after the retort, and the mixture will be charged to the induction smelting furnace. Doré bars will be poured from the smelting furnace and shipped off site for further refining.

17.1.13 Mercury Abatement Systems

As part of this update, the mercury abatement circuits in gas handling have been improved and designed in more detail. Mercury abatement systems will be required at the following locations:

- Carbon reactivation – kiln feed and discharge
- Electrowinning cell fume hoods
- Gold refinery area
- POX vent gas (designed by Hatch and described in the autoclave section).

In each area, mercury will be expected to volatilize into the gas stream exiting the circuit because of the elevated temperatures. Fume hoods and ducting will be used to transport the gas to mercury scrubbing systems and other mercury removal equipment. Mercury will be collected and disposed of in two forms: condensed liquid, which will be collected in specialized flasks, and mercury-loaded carbon. Both will be shipped off site.

The mercury recovery system for the regeneration kiln will consist of a spray wet scrubber, a venturi scrubber, a wet gas condenser, a coalescer, and carbon columns. The resulting clean gas will be exhausted through a stack to atmosphere. The spent carbon from the carbon filters will be periodically replenished with fresh carbon. Spent carbon transferred out of the columns will be securely packaged and is transported off site to a certified hazardous waste facility.

Raw water will be used as the quenching and scrubbing solution. In the process, various concensate streams will be collected from the scrubbing circuit. In the regeneration abatement system, the excess condensates can be sent directly to the POX blowdown area or via a coalescer to recover remaining amounts of mercury down to regulatory limits before being pumped to the POX area for reuse in the process.

In the electrowinning area, vapour and air from the electrowinning cells and other equipment in the gold recovery area will be discharged at a rate of approximately 8,000 scfm and at temperatures up to 175°F (80°C). The mercury in this stream will be recovered using two wet sprary scrubbers, a venturi scrubber, a wet gas condenser, and a coalescer. Carbon columns set up in a lead-lag fashion will serve as the final capture for mercury down to regulatory limits in this area.

A booster fan will funnel the final air stream out through an exhaust stack to the atmosphere. The carbon from the carbon filters will be periodically disposed of to a certified hazardous waste facility.

Raw water will be used throughout the scrubber system. Overflow water will be sent to a coalescer and settling tank combination to capture elemental mercury before proceeding to cyanide destruction. The underflow discharge of these systems will be sent to a solution collection tank in the cyanide area mercury treatment facilities. The heavier mercury that settles to the bottom of the tank will be pulled out for disposal. The discharge water closer to the top of the tank will be pumped to a recirculating coalescer, from which treated scrubber solution will be pumped to the cyanide destruction circuit.

In the refinery, gas discharge from the induction furnace will leave the unit at an elevated temperature of 175°F (80°C) and low mercury content, but with potential for dusting. Therefore, the mercury entrainment system in this area will consist of a dust capture cyclone and carbon columns. The cyclone will capture gold particulate that has drifted out in the gas stream, which will then pass through two sets of carbon column sets, both filled with sulphur-impregnated carbon. These will pull out any remaining mercury before a booster fan releases the gas to atmosphere through an exhaust stack.

17.1.14 Reagent Preparation

All reagent mixing will take place in a designated area within the mill building. The design of this area will incorporate such features as section bunding with dedicated sump pumps for individual reagent types, and segregated ventilation and dust control for areas with potential for dust or fume release. The design of the reagent preparation area aims to be consistent with the general intent of the International Cyanide Management Code.

Reagents will be received in the followed forms;

- *Xanthate* – will be received as pellets in 1,874 lb (850 kg) bulk bag
- *Frother 1 (Methyl isobutyl carbinol, MIBC)* – will be received in 22 st (20 t) bulk isotainers as a high-strength solution.
- *Frother 2 (F549)* – will be received in 1.1 st (1 t) reagent totes as a high-strength solution.
- *Dispersant* – will be received in 25 st (23 t) bulk isotainers as a high-strength solution.
- *Soda Ash* – will be received as granules in bulk in 20 st (18.1 t) lined sea containers.
- *Flocculant* – will be received as powder in 20 ft (6 m) lined containers.
- *Cyanide* – will be received as briquettes in bulk 24 st (22 t) isotainers. To minimize personnel exposure to the reagent mixing process, each isotainer will be pre-piped to serve as a cyanide mix tank.
- *Carbon* – will be received in 1,000 lb (454 kg) bulk bags.
- *Nitric acid* – will be received in 27 st (25 t) bulk isotainers as a high-strength solution.
- *Caustic soda* – will be received as dry beads in 1.1 st (1 t) bulk bags.
- *Copper sulphate* – will be received in crystal form in 1.4 st (1.25 t) bulk bags.
- *Antiscalant (Millsperse 813)* – will be received in 1.1 st (1 t) tote tanks as a high-strength solution

- *Mercury suppressant (UNR 829)* – will be received in 1.1 st (1 t) tote tanks as a high-strength solution.

Lime

Lime is received as pebble lime in 20 ft (6 m) lined containers and will be transferred to a lime silo beside the reagent area. The pebble lime will be slaked in a grinding-mill-type slaker and will then held in two holding tanks in the reagent area inside the building. The lime will be used for pH control at various points. Lime addition will be by means of a pressurized lime loop distribution system.

Sulphur Dioxide

Sulphur dioxide will be added to the cyanide destruction process. Bulk sulphur will be delivered to site in 1.1 st (1 t) bulk bags. The sulphur will be transferred to the molten sulphur tank, where it will be melted by heat from the carbon elution circuit. The molten sulphur will be pumped to a furnace where it will be mixed with air and combust to yield a discharge gas containing 17% SO_2 by volume. Discharge SO_2 gas will pass through the induced draft (ID) fan/blower for dilution to 2% to 3% SO_2 and will be pressurized to feed the cyanide destruction tanks.

17.2 Process Services

17.2.1 Air

The air separation unit (ASU) will produce ~99.5% purity high-pressure oxygen. Its design production capacity is 1,930 stpd (1,750 t/d) of contained oxygen gas. By design, the ASU will have a production turndown of 50%, or 965 stpd (875 t/d), by shutting down one of the main air compressors. With low-pressure liquid oxygen tanks completely full, the LP-LOX facility will be able to supply another 24 hours, or 1,890 st (1710 t), of stored liquid oxygen before having to shut down.

Waste nitrogen from the ASU will be pressurized with a compressor and will be used in the autoclave gas handling circuit to reduce the oxygen content of the off-gas prior to mercury scrubbing. When completely full, the LP-LIN tanks will be able to supply another 24 hours, or 720 st (660 t), of stored liquid nitrogen.

Plant air and instrument air will be provided by three sets of compressors and an air receiver. Plant air will be delivered directly from this receiver, while instrument air will be further filtered and dried before distribution.

Low-pressure process air compressors will be provided for solution neutralization, cyanidation and cyanide destruction service. Low-pressure air blowers will be supplied for flotation, where air will be supplied directly through manifolds to each flotation cell.

17.2.2 Plant Water Distribution

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

Contact water will come from the mine facilities and waste dump runoff that collects in the contact water pond. Typically it will be of fairly high quality with low levels of suspended and dissolved solids, and would be used for:

- Elution
- Electrowinning and refining
- Autoclave process water system (quench and gland)
- Concentrate ccd wash glands and wash water
- Cyanide, caustic, and other reagent systems
- Primary flocculant mixing
- Make-up water to cooling systems including the oxygen plant.

The distribution system will consist of the contact water tank and a pump distribution system.

During periods of high runoff into the contact pond, when quality degrades and quantities are excessive, contact water will substitute 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 will be raw water which comes from the peripheral dewatering wells in the pit. As long as contact water will be of sufficient quality and quantity, the raw water will be treated in the water treatment plant and discharged to the environment. When required to replace contact water, it will be suitable for all contact water usages. Raw water will also be important as the source of water for charging mill cooling and heat transfer systems. The raw water distribution system will consist of a single raw water tank and two distribution pumps.

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

The reclaim water system will supply water to processes that do not need high-quality water. Water will be reclaimed from the TSF and pumped to a reclaim water head tank. The water will be passed through a double-pipe heat exchanger (the heat being recovered from plant tailings during winter) and will then flow by gravity for distribution to the following areas:

- POX CCD
- CIL feed neutralization and CIL
- carbon regeneration
- Cyanide destruction tails safety screens.

Reclaim water will also be supplied as the feed to the gland water system and flotation process water system.

Reclaim water used in CIL will first be treated for magnesium removal by an ion exchange vendor package.

POX blowdown water has been adopted as the terminology for raw water that has been used in the pressure oxidation process, but is still relatively clean and, more importantly, has not been contaminated by chlorides (above levels in the original raw water). Within the pressure oxidation area, raw water will be supplied to the gas handling systems—secondary condenser, venturi scrubber, mist eliminator—where it will be used to cool and clean particulates from the off-gas. Raw water will also be used as feed to the demineralized water system and in turn as feed to the boiler.

Demineralized water will be produced as a utility in both the POX and concentrator areas serving the POX plant and the combined requirements of the carbon elution circuit and power plant respectively. Demineralized water will be produced from plant raw water using multimedia filters and reverse osmosis system (concentrator/power plant) or ion exchange (POX).

17.3 Process Ventilation

Carbon dioxide will be produced during the acidulation of concentrate. The principal concern for ventilation will be to control the discharge of this gas away from tankage, as it will pose a safety hazard for operations personnel. As a result, the tops of the acidulation tanks will be covered and the gas produced will be vented to a dispersion stack.

Vent gas from the autoclave will contain superheated steam, unconsumed oxygen, and in small quantities, nitrogen, carbon dioxide, carbon monoxide, volatile organic compounds (VOCs), gaseous sulphur, and mercury. In addition, flash steam produced in the discharge flash vessels that is not consumed in the slurry heater vessels will report to the off-gas system. The ventilation in the pressure oxidation off-gas system is included in the description of the autoclave.

Carbon dioxide will be produced in the neutralization of acidic solution in the FT neutralization circuit (these tanks are located outdoors). The principal concern for ventilation will be to control the discharge of this gas away from tankage, as it will pose a safety hazard for operations personnel. As a result, the tops of the acidulation tanks will be covered and the gas produced will be vented to a dispersion stack. The stack will be located away from the immediate areas of personnel travel and will allow the dispersion of the carbon dioxide to the general environment.

The CIL tanks will be located in a dedicated plant building to provide year-round weather protection for operators in the area. The pH set point for cyanide leaching in CIL will be approximately 11.0. The tank will therefore be covered to contain the gases. A scrubber fan will pull air from above the CIL tanks through an HCN scrubber, maintaining good ventilation in the working area above each tank.

In the acid wash area, ventilation will minimize acid vapour from the acid wash process. In carbon elution and regeneration systems, ventilation will be installed to control the evolution of mercury vapours present from the carbon in these areas. The gas stream will report to a mercury abatement system as previously described. .

The principal concern for ventilation in elecrowinning, sludge retort and refining is to control the potential presence of mercury. This has been previously described.

17.4 Control System

The process control system (PCS) for the plant will consist of a network of distributed controllers and human-machine-interface (HMI) equipment. The PCS, HMI stations, and all associated communications equipment will be of current technology that has been proven efficient and reliable in similar installations. The system will be capable of direct expansion to control all equipment required to meet possible future requirements of the mine.

Overall control of the plant will be provided from a centralized control room adjacent to the grinding area. Subsidiary control rooms will be installed in the crusher area, pressure oxidation area, oxygen (air separation) plant, and water treatment (operations) plant. The control system will perform a wide variety of functions, including analog and motor control and sequencing, process control graphics operator interface, historical trending, and alarm displays/logging. Where necessary, operators in communication with the control room will provide control.

Key process parameters will be measured by instrumentation, with regulatory control (valves, for example) adjusted as required to maintain desired process values. Provision will be made to incorporate supervisory control and other advanced control methods in the process control system. In addition to the control system providing stability within the various processes, automatic sampling will be incorporated in the plant to acquire samples of a quality sufficient for metallurgical accounting.

Industry-standard interlocks within the control system and control points will be employed to ensure the safety of personnel and equipment by protecting equipment against overload or damage and preventing improper operation.

The pressure oxidation area will be equipped with a Safety Instrumented System (SIS), which performs the safety functions intended to achieve or maintain safe conditions for the process with respect to a specific hazardous event. The oxygen plant will be equipped with an SIS independent from the DCS.

17.5 Laboratories

Laboratory facilities, including the assay, metallurgical, and environmental laboratories, will be constructed on the ground floor of the office complex within the process plant building.

The metallurgical laboratory is designed for bottle roll tests, grinding tests, screen analysis, flotation testwork, titration, and filtering. The environmental laboratory will be used to digest environmental samples for base metal or cyanide analysis under a fume hood. Mill offices will be provided in the administration.

17.6 Mill Feed Schedule

The detailed mill feed schedule for the life-of-mine and overall recovery for each production period, including both flotation and pressure oxidation recoveries, is shown in Table 17-1. Figure 17-2 shows the planned gold production by year.

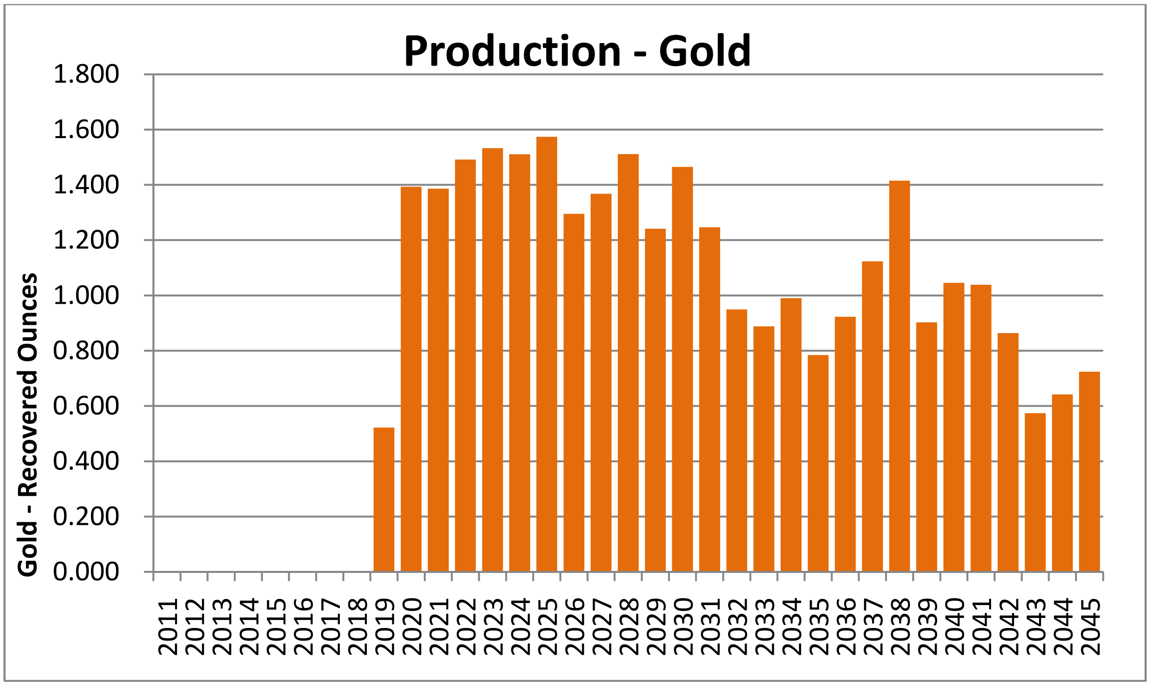
Table 17-1: Projected Process Schedule and Recoveries

Parameter	Unit	Years													
		2019	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**
Oxidized (partially)	t	1,987,398	1,761,840	1,503,643	3,145,828	3,552,938	2,645,576	637,038	447,981	422,061	230,004	1,118,278	53,906	138,358	122,156
Greywacke	t	1,054,359	2,142,474	3,008,608	2,063,301	3,246,542	2,783,814	4,176,156	3,554,924	2,208,569	6,779,429	3,870,714	4,442,630	5,194,025	4,563,419
Shale	t	177,092	1,118,228	834,573	591,570	581,968	549,805	705,748	1,163,538	1,761,582	948,400	928,342	1,376,737	1,118,606	1,265,504
ACMA	t	1,635,282	7,739,033	7,251,604	5,214,996	4,550,999	1,549,690	-	14,282	3,171,074	4,503,389	5,325,813	4,618,313	258,455	369,032
Lewis	t	389,911	781,528	2,927,156	3,705,199	2,049,362	4,643,110	4,072,945	2,108,908	1,752,948	4,528,363	6,779,133	7,202,402	5,686,326	4,868,224
Vortex	t	33,739	216,096	1,530	762,841	1,318,582	4,374,965	3,764,182	1,780,222	576,617	397,500	1,346,080	1,109,190	5,061,486	4,829,487
Akivik	t	645,475	1,899,357	432,666	130,643	2,889,899	53,664	12,862	731,752	1,290,653	598,495	46,533	102,246	2,403,353	2,778,495
400	t	1,254,265	1,774,444	1,920,414	1,563,871	1,137,256	2,463,731	3,949,790	3,808,696	3,331,714	533,903	39,128	408,859	-	-
Aurora	t	361,260	1,263,112	1,324,698	2,233,703	229,926	468,919	1,400,270	5,011,055	5,066,974	1,142,578	591	6,358	-	-
Feed Contained Ounces	Au oz	599,650	1,521,665	1,517,634	1,687,677	1,705,246	1,685,522	1,731,747	1,412,059	1,502,470	1,733,384	1,374,914	1,587,984	1,257,034	1,028,149
Plant Feed Grade	Au g/t	2.47	2.53	2.46	2.70	2.71	2.68	2.88	2.36	2.39	2.74	2.20	2.56	1.97	1.70
Float Recovery	%	89.48	94.67	94.56	91.47	93.03	92.80	94.09	94.14	95.29	93.95	94.06	94.39	94.09	93.80
Float Tail	Au g/t	0.24	0.13	0.13	0.23	0.19	0.19	0.17	0.14	0.11	0.17	0.13	0.14	0.12	0.11
POX Recovery	%	96.6	96.6	96.6	96.6	96.6	96.6	96.6	96.6	96.6	96.6	96.6	96.6	96.6	96.6
Recovered Grade	Au g/t	2.15	2.32	2.25	2.39	2.44	2.41	2.62	2.15	2.20	2.49	2.00	2.33	1.79	1.54
Overall Recovery	%	86.4	91.5	91.3	88.4	89.9	89.6	90.9	90.9	92.0	90.8	90.9	91.2	90.9	90.6
Final Tail Grade	Au g/t	0.32	0.21	0.21	0.31	0.27	0.28	0.26	0.21	0.19	0.25	0.20	0.23	0.18	0.16
Recovered Ounces	Au oz	522,160	1,393,001	1,387,112	1,491,277	1,532,461	1,510,915	1,573,962	1,284,174	1,382,989	1,573,188	1,249,289	1,447,947	1,142,524	931,589

		Years													
		2033	**2034**	**2035**	**2036**	**2037**	**2038**	**2039**	**2040**	**2041**	**2042**	**2043**	**2044**	**2045**	**Total**
Oxidized (partially)	t	483,995	2,464,918	769,304	1,069,689	730,892	1,071,461	1,136,536	1,126,932	1,147,109	23,800	2,319,194	5,309,490	3,690,124	**39,110,451**
Greywacke	t	4,911,357	3,146,147	2,800,535	4,051,701	3,255,519	2,875,685	2,744,980	4,269,503	3,989,224	3,525,348	4,430,219	4,715,208	3,172,669	**96,977,059**
Shale	t	662,674	988,861	1,638,519	820,665	793,370	1,336,787	1,545,713	601,644	803,436	558,252	1,270,101	1,205,331	688,607	**26,035,652**
ACMA	t	3,474,862	5,378,559	4,256,276	5,222	1,627,159	6,718,463	6,814,697	1,955,494	158,545	-	322,993	1,600,019	2,969,891	**81,484,140**
Lewis	t	3,675,772	5,899,590	7,976,732	8,760,838	7,891,393	3,734,394	2,995,792	3,303,994	7,563,716	13,515,436	6,206,166	3,673,347	2,935,047	**129,627,731**
Vortex	t	3,770,250	743,305	1,804,110	3,344,643	4,100,798	2,857,096	2,618,909	4,264,064	2,169,032	1,568,842	2,824,626	1,806,871	974,434	**58,419,498**
Akivik	t	1,353,568	215,468	180,118	478,039	1,067,457	903,596	758,808	1,175,934	669,482	-	1,206,373	153,116	1,192,187	**23,370,238**
400	t	62,017	179,882	7,703	447,006	31,482	47,619	-	1,227,557	2,152,427	-	785,673	422,453	940,990	**28,490,881**
Aurora	t	-	41,791	4,613	521,273	39,688	83,329	-	365,934	548,126	-	125,260	663,682	392,098	**21,295,240**
Feed Contained Ounces	Au oz	969,940	989,842	1,068,528	958,737	1,027,311	1,295,580	1,505,365	1,227,048	911,615	1,099,906	772,766	827,737	847,674	**33,847,184**
Plant Feed Grade	Au g/t	1.64	1.62	1.71	1.53	1.64	2.05	2.52	2.09	1.48	1.78	1.23	1.32	1.56	**2.09**
Float Recovery	%	93.54	92.36	93.56	92.20	93.60	93.98	94.24	92.64	91.39	93.66	89.06	81.06	87.48	**92.99**
Float Tail	Au g/t	0.11	0.12	0.11	0.12	0.10	0.12	0.14	0.15	0.13	0.11	0.13	0.25	0.19	**0.15**
POX Recovery	%	96.6	96.6	96.6	96.6	96.6	96.6	96.6	96.6	96.6	96.6	96.6	96.6	96.6	**96.6**
Recovered Grade	Au g/t	1.48	1.44	1.55	1.36	1.48	1.86	2.29	1.87	1.30	1.61	1.06	1.03	1.31	**1.87**

		Years													
		2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	Total
Overall Recovery	%	90.4	89.2	90.4	89.1	90.4	90.8	91.0	89.5	88.3	90.5	86.0	78.3	84.5	**89.8**
Final Tail Grade	Au g/t	0.16	0.17	0.16	0.17	0.16	0.19	0.23	0.22	0.17	0.17	0.17	0.29	0.24	**0.21**
Recovered Ounces	Au oz	876,426	883,169	965,768	853,927	928,838	1,176,162	1,370,399	1,098,145	804,775	995,187	664,821	648,180	716,358	**30,404,744**

Figure 17-2: Planned Gold Production by Year



Note: Years shown in Table 17-1 and Figure 17-2 are indicative, and are for mine planning purposes only. Formal approval of any mining operation is contingent on approval from the Boards of Donlin Gold, Barrick, and NovaGold, and receipt of appropriate regulatory and social permits.

17.7 Comments on Section 17

In the opinion of the QPs, the following conclusions are appropriate:

- The process as currently envisaged will consist of sulphide flotation pre-concentration with pressure oxidation followed by CIL to produce gold at acceptable recoveries through the application of autoclave sulphide oxidation technology.
- A semi-autogenous grinding circuit followed by series ball milling and pebble crushing has been selected for the comminution requirements based on the ability to handle the hardness and quality variability of the ore.
- The air flotation flowsheet will provide a circuit design that will maximize sulphide recovery as demonstrated during numerous bench and pilot plant metallurgical tests.
- Pressure oxidation of the flotation concentrate will allow high recoveries in CIL cyanidation, as demonstrated through extensive pilot-plant testing. Controlling

autoclave residence time to match the requirements of the concentrate feed will be important.

- Effective neutralization of acidic solutions can be carried out through the efficient use of the natural carbonate content of the flotation tailings, thereby reducing the amount of lime or limestone that must be transported to site.
- Effective detoxification of WAD cyanide species in CIL tails through SO_2/Air destruction.
- Effective management of arsenic in the milled ore is possible through the use of the pressure oxidation circuit. The ore has sufficient iron content to permit proper precipitation of arsenic with iron to form acceptable forms of arsenic precipitation products, suitable for long-term storage in the tailings facility.
- Mercury emissions can be effectively controlled with the proposed mercury abatement systems on the pressure oxidation, carbon regeneration, electrowinning, retort, and bullion smelting processing equipment. The proposed mercury recovery technology for the autoclave off-gas systems exceeds current known best practices for this application.
- The final tailings stream will be a combination of WAD CN detoxified CIL tails and neutralized acidic liquors generated from the autoclave, using flotation tails and lime, for final pH adjustment to a pH of 7. The tailings will be non-acid producing. The circuit design as described will permit maximum recycle of tailings decant water back to the plant to eliminate the need for disposal of process waters.

18.0 PROJECT INFRASTRUCTURE

18.1 Access and Logistics

18.1.1 Port-to-Mine Access Road

The port-to-mine access road (Jungjuk route) will traverse varied terrain from the mine site to the Kuskokwim River dock site near the mouth of Jungjuk Creek to the mine site (Figure 18-1).

The port site is located on the north bank of the Kuskokwim River, 8 miles (13 km) downstream of the village of Crooked Creek; there is currently no road connection between the two locations. To the mine site battery limits, the road will be 27 miles (44 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. The road route will traverse mostly upland terrain. A 3 mile (4.8 km) long spur road, beginning at route mile 5.4 (km 8.7), will serve the Project airstrip. The mine camp facilities will be located at mile 2.4 (3.9 km).

There are 50 identified stream or drainage crossings along the road route, but only six of them are significant and will require bridging. Bridge lengths vary from 25 ft (7.5 m) to 82 ft (25 m).

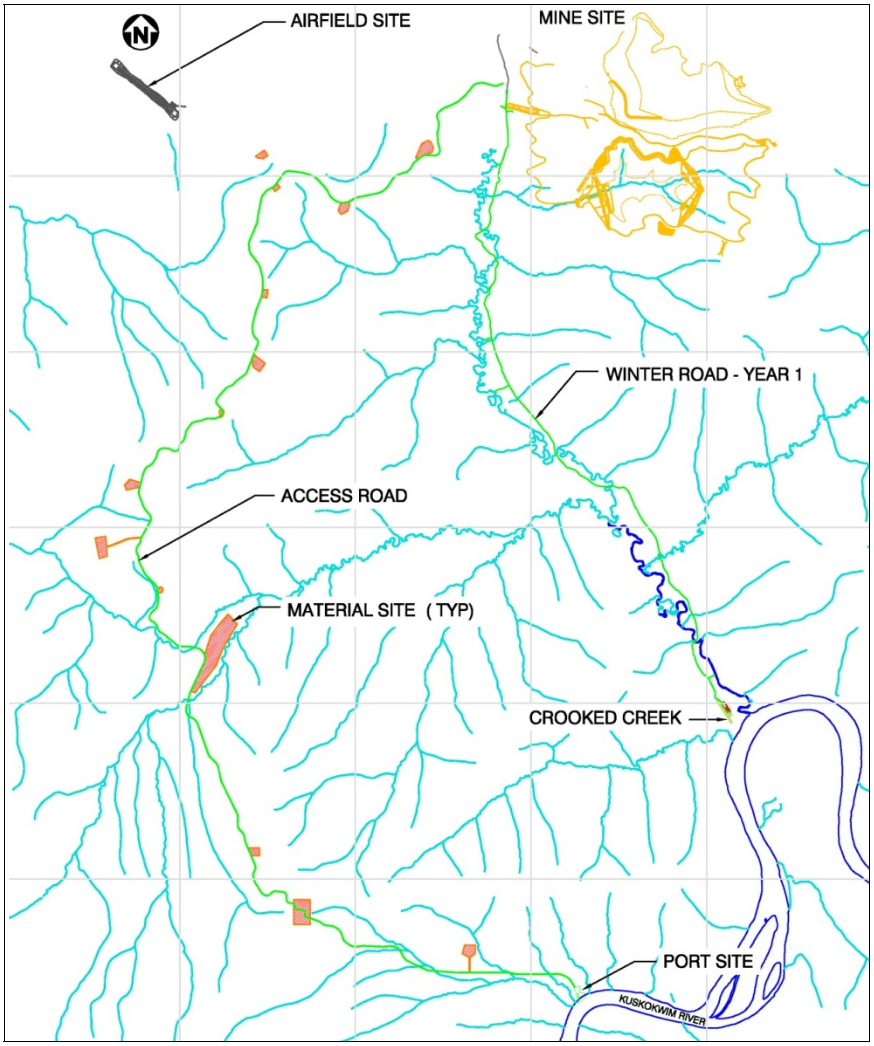
18.1.2 Road Construction

Road construction activities will be divided among three distinct areas: the site access roads to ancillary facilities (explosives storage, airstrip, haul roads); the port site permanent access road; and the Crooked Creek winter construction access road. In general, roads will be constructed using conventional cut-and-fill techniques, except for the winter construction access road, which will be developed as an "ice road".

Simple fill-type embankments will be constructed on most of the ridge-line sections to preclude surface disturbance and reduce snow-drifting on the roadway. In general, material sites have been located at regular intervals along the alignment, precluding the need for extended haulage of construction rock.

A geofabric underlay will be provided in areas identified as having significant amounts of permafrost or wet soils.

Figure 18-1: Basic Route of Mine Access Road



18.1.3 Airstrip

The airstrip will be approximately 9 miles (14 km) by road west of the mine site. The airstrip design is based on U.S. Department of Transportation, Federal Aviation Administration (FAA) 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.

18.1.4 Cargoes

General cargoes sourced globally will be shipped in containers or as break-bulk to marine terminals in Seattle and Vancouver where they will be loaded onto ocean barges for transport to Bethel. Because the Kuskokwim River begins to freeze up in October and the mouth of the river does not usually clear of ice until late May, the shipping season for both ocean and river barges will be limited to 1 June to 1 October each year. The first general cargo barge of each shipping season will load and leave Seattle or Vancouver in early-May to be off the mouth of the river by the beginning of June.

Because of draft restrictions on the river downstream of Bethel later in the season, fully-laden ocean barges will discharge part of their cargo direct to river barges at Oscarville Crossing about 5 nm (nautical miles) (10 km) downstream from Bethel. Those cargoes unloaded at Bethel will be placed into temporary storage or transferred directly to river barges for shipment to Jungjuk, about 168 nm (312 km) upstream. At Jungjuk general cargo will be off-loaded and either placed in temporary storage to await transport or loaded directly onto trucks for transport to the mine site.

Empty container storage yards, local transport, marine terminals in Seattle and Vancouver, and the ocean and river general cargo barge fleets will be operated by third parties.

18.1.5 Fuel

The fuel supply chain will include the following major components:

- Ocean transport by double-hull barge from refineries located in the Pacific Northwest
- Fuel storage and transfer terminal at Dutch Harbor
- 13 USMgal (49 ML) fuel storage tanks at Dutch Harbor
- Ocean transport by double-hull barge from Dutch Harbor to Bethel
- Fuel storage and transfer terminal at Bethel
- 10 USMgal (38 ML) of dedicated fuel storage tanks at Bethel
- River transport by double-hull barges from Bethel to Jungjuk
- Storage and transfer terminal at Jungjuk
- 2.6 USMgal (9.9 ML) capacity storage tanks at Jungjuk
- Transport by tanker truck from Jungjuk to the mine site

- 38.5 USMgal (146 ML) fuel storage tanks at the mine site.

Fuel sourced at refineries in the Pacific Northwest will be delivered by barge to Dutch Harbor, where it will be stored to await onward transport to Bethel. The first shipments should leave the refineries in early May. Fuel will be transported from Dutch Harbor to Bethel in a 2.9 USMgal (11 ML) capacity ocean barge, which will be loaded so that it would not have to lighter fuel to reduce draft prior to crossing shallower reaches downstream of Bethel. At Bethel fuel will either be placed in storage or loaded direct to a river barge for transport to Jungjuk. At Jungjuk fuel will be unloaded to storage tanks. Fuel will be transported by a fleet of 13,500 USgal-capacity (0.51 ML) tanker trucks to the mine site. The fuel terminals at Dutch Harbor and Bethel and the ocean and river barge fleets will be operated by third parties.

18.2 Site Facilities

18.2.1 Site Investigations

The site investigation work carried out at the Donlin Gold Project site includes evaluations in the technical areas of geotechnical engineering, surficial geology, hydrogeology, climate, hydrology, and seismicity. Evaluations for the plant site and associated facilities, contact water, freshwater and diversion dams, tailings storage facility, open pit, and waste rock facility were performed by BGC Engineering Inc. (BGC).

RECON, LLC (RECON) conducted the evaluation for the port site and the access corridors between the port site and the mine site. Both BGC and RECON contributed to an assessment of potential borrow sources for construction materials.

Key findings from the work performed between 2004 and 2010 were:

- The bedrock beneath the primary crusher, truck shop, plant site, and fuel farm facilities consists of interbedded, blocky, and weathered siltstones, shales, and greywackes (sandstones) with some minor igneous intrusions (dykes and sills). Geotechnical logging of drill core indicates average rock mass rating (RMR '76) and geological strength index (GSI) values ranging between 35 and 45, corresponding with "poor" to "fair" bedrock, respectively.
- Hydraulically, the bedrock unit is considered to be undifferentiated. The hydraulic conductivity of the bedrock tends to decrease with depth, although it has been found to vary over approximately three orders of magnitude at any given depth. Numerous faults are present in the bedrock; however, to date there is no strong

evidence to suggest any particular fault has a significant control on groundwater flow.

- Average annual precipitation at Donlin is estimated at 19.6" (499 mm), consisting of 13.6" (345 m) of rainfall (69%) and 6.1" (154 mm) of snow (31%). Snow typically starts to accumulate in mid-October, while snowmelt occurs on average between early April and early May.
- Discontinuous permafrost is expected at the Project site. Observations to date have confirmed the presence of discontinuous permafrost. Data collected to date also indicate localized warm permafrost.
- The Donlin site lies in a seismically active region. The hazard potential classification for the water and tailings dams is rated as “high” in view of the potential for loss of life and severe environmental damage that could result from uncontrolled release of the stored tailings and supernatant water in the event of an earthquake. The maximum design earthquake is characterized by a relatively robust peak horizontal ground acceleration of 0.36 *g* from a magnitude 7.8 event or by a peak ground acceleration of 0.07 *g* from a magnitude 9.2 subduction event. For the design of plant site buildings and other structures, the seismic design provisions of the 2006 International Building Code have been adopted, and appropriate design response spectra have been developed based on ground motion parameters with a 2% frequency of exceedance in 50 years.

18.2.2 Plant Site Design Considerations

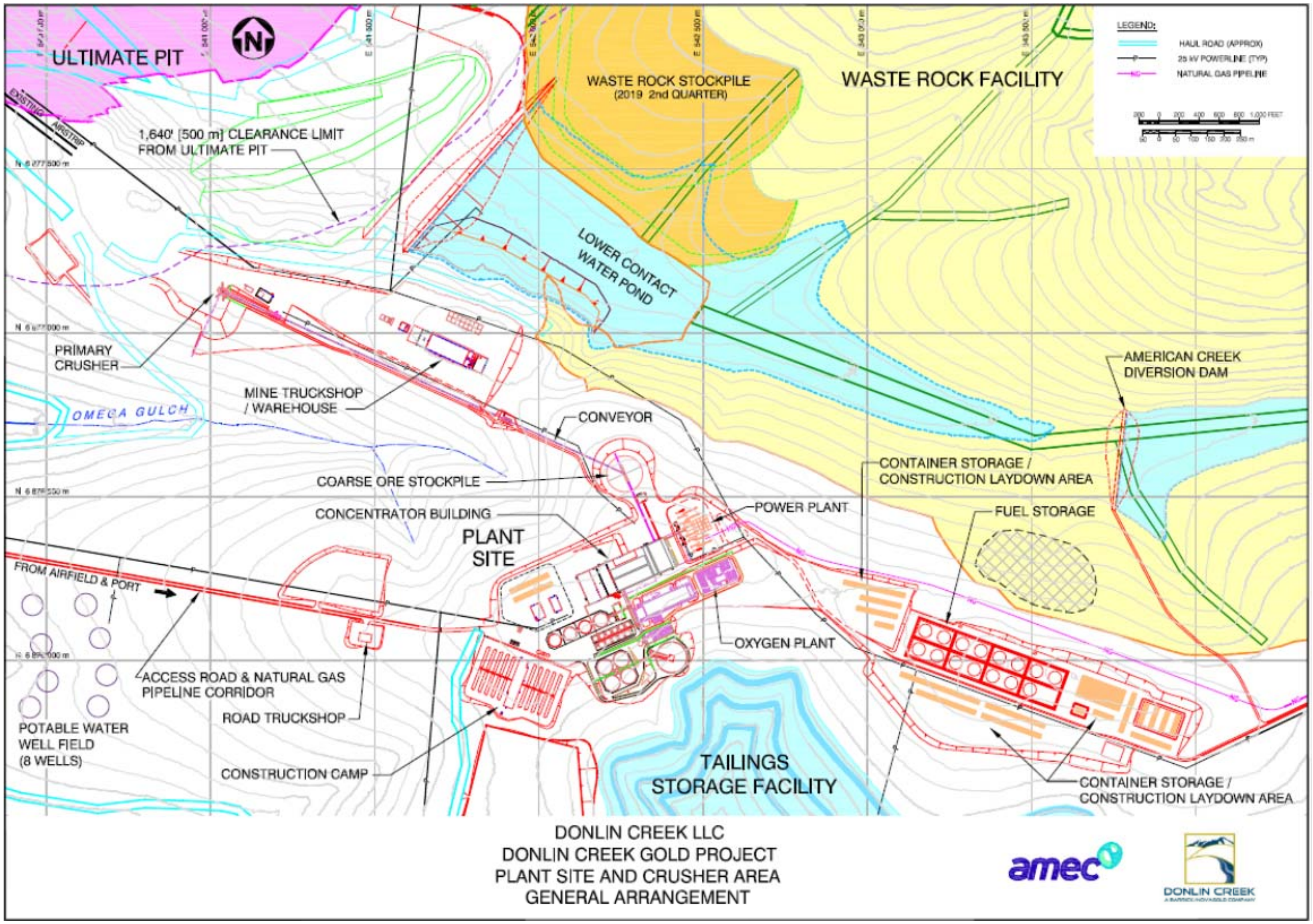
The plant site and fuel tank farm will be in the Anaconda valley on the ridge just northwest of the TSF at a nominal elevation of 984 ft (300 m). This is the same site selected during FSU1 (Figure 18-2). The proposed facility layout contains the entire process area within the Anaconda and American Creek valleys and therefore minimizes any direct impact on Crooked Creek.

The layout of the plant site was designed to take maximum advantage of the topography, ranging from an elevation of 1,000 ft (305 m) at the grinding area and stepping down through the process to elevation 853 ft (260 m) at the flotation tailings thickener. The layout also provides for efficient movement of equipment and material products around the site.

The locations of other facilities relative to the process plant are as follows:

- The fuel tank farm is 1.2 miles (2 km) to the east, at elevation 980 ft (299 m).
- The open pit(s) are approximately 1.86 miles (3 km) to the northwest, at elevation 525 ft (160 m) (top of ramp).

Figure 18-2: Plant Site Layout



- The crusher is approximately 0.9 miles (1.5 km) to the northwest, at elevation 755 ft (230 m) at the dump pocket and 648 ft (197.5 m) at the low end of the primary feed conveyor.
- The tailings pond is approximately 0.6 miles (1 km) to the southeast, initially at elevation 558 ft (170 m) and ultimately at 833 ft (254 m), permitting gravity-flow of tailings from the process plant.

The layout of the plant site takes advantage of the prevailing wind direction from the southeast. The flow of air will mostly be from the oxygen plant to the COS and then toward the pit. As such, any dust from the plant site and the COS will be blown toward the ore stockpile, contact water pond, open pit, and adjacent waste dump.

The plant platform will be graded to allow for gravity flow through the process plant area and southwest to the thickeners, then southeast down to the TSF.The primary crusher will be constructed on a ridge on the south side of American Creek, orientated to make use of the southern slope of the ridge, minimize the length of the coarse ore conveyor to the plant site, and permit the vertical and horizontal alignment to tie into the coarse ore stockpile at the plant site. The crusher structure will be a concrete tower founded on a raft-type foundation with a reinforced earth wall around three sides. Crushed ore will be fed to a 1.46 mile (2.3 km) long coarse ore conveyor running to a coarse ore stockpile (COS) at the process plant.

The truckshop will be on the ridge east of the primary crusher to provide easy access for maintenance and refuelling of the haul trucks. The truckshop will house ten heavy vehicle repair bays, various specialty service bays, a warehouse, changerooms, and offices. The mine rescue truck, fire truck, and first aid facility will also be housed within this building. The layout has been designed to allow the building to be extended in the future by up to four bay-lines, for a total of eight additional maintenance bays.

To eliminate the transport of explosives through the plant area, the explosives plant will be located north of the pit. Access to the site will utilize the service road out to the Snow Gulch dam. The site will be located just behind a ridgeline to protect the explosives plant from potential flyrock from mining activities in the pit.

18.2.3 Plant Site Facilities

Plant site facilities will include:

- Crusher
- Coarse ore conveyor

- Coarse ore stockpile
- Pebble crusher
- Conveyor galleries
- Concentrator
- Utilidors and access walkways
- Power plant
- Truckshop
- Road fleet truckshop
- Truck wash
- Cold storage building
- Warehouse
- Administration Offices/ Changerooms/ Assay Laboratory
- Construction camp
- Permanent accommodation complex.

With the supply of natural gas to the mine site, heating for the plant site buildings and modules, including the truckshop and permanent accommodation complex, will be provided by natural-gas-fired air unit heaters and makeup air units. The accommodation complex will be heated by natural gas furnaces with a forced-air system and supplemental baseboard heaters.

Continuous ventilation will be provided for all human-occupied and selected unoccupied spaces. Ventilation rates will vary depending upon the level of occupancy and the intended use of the space as per ASHRAE, OSHA, and the State of Alaska building code and standards. Ventilation systems will include make-up air units for continuous supply of tempered air, summer supply fans to provide extra cooling during the warmer months, and general exhaust fans.

Dust control systems will include hoods, ductwork, dust collectors, and enclosures designed to prevent fugitive dust or fume emissions at their source. Dust collectors will be designed and selected to reduce particulate emissions from the discharge air to meet the applicable air pollution control regulations.

18.3 Camps and Accommodation

The construction camp will be built on a bench near the process plant. The camp will include 14 stand-alone, three-storey dormitories designed to accommodate 2,560 people and a stand-alone, single-storey core services facility.

The permanent accommodation complex will be located just off the main access road, approximately 4 miles (6 km) west of the plant site. The complex will house an estimated 434 people in Year -2 and be expanded in Years 1 and 6 to accommodate a maximum of 638 people during operations.

Two modular sewage treatment plants (STPs) will be provided: one for the permanent accommodations facility, and one for the construction camp immediately west of the plant site. The sewage treatment plant for the construction camp will later be reduced in size to accommodate the operational requirements of the plant site area.

18.4 Waste Storage Facilities

18.4.1 Location

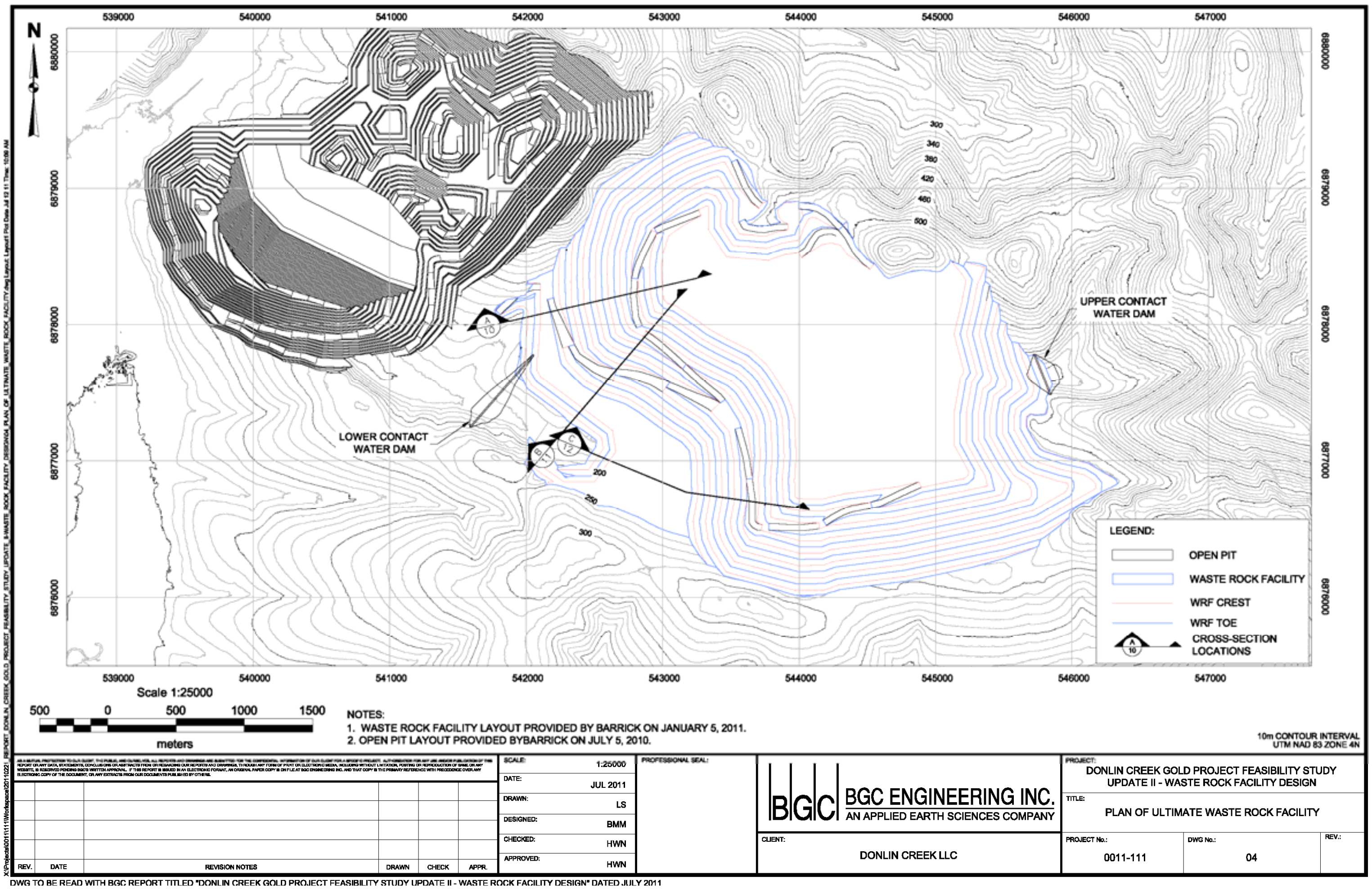
The WRF will be located in the American Creek valley, east of the open pit. The ultimate footprint of the WRF covers an area of approximately 3.5 square miles (9 km^2). Approximately 2,449 Mst (2,222 Mt) of waste rock and 46 Mst (42 Mt) of overburden will be placed in the WRF. The top lift of the WRF will be at an elevation of approximately 1,705 ft asl (520 masl), resulting in a maximum dump height of about 1,115 ft (340 m) and thickness of about 985 ft (300 m). Most of the WRF will be constructed in 100 ft (30 m) lifts. The toe of each subsequent lift will be set back 155 ft (47 m) from the crest of the previous lift, resulting in an overall dump slope of 3H:1V.

A plan showing the layout of the ultimate WRF is given as Figure 18-3.

18.4.2 Acid-base Accounting

Acid-base accounting indicates that the potential for acid rock drainage (ARD) will be highly variable; however, most of the waste rock has low potential because of the excess neutralization potential of calcium and carbonate minerals. Sulphur concentrations vary from near detection limit (0.01%) to above 3%. Sulphur distribution is bimodal with typical modes at about 0.1% and 1%, depending on rock type. The rhyodacites tend to be more mineralized than the sedimentary rocks. Laboratory tests have confirmed that rock containing higher sulphur concentrations will generate ARD.

Figure 18-3: Plan of Ultimate Waste Rock Facility



In addition to sulphur mineralization, arsenic concentrations (occurring mainly as arsenopyrite) are elevated throughout the rock and are broadly correlated with sulphur content. Arsenic concentrations vary from near 10 mg/kg to up to 1% in highly mineralized rock. Laboratory and field experiments have demonstrated that arsenic and several other elements (sulphate, antimony, selenium) leach at elevated levels with respect to Alaska water quality standards. Arsenic leaching is likely to vary but is expected to be generated from all types of waste rock. As a result of the identified variable potential for ARD and leaching of arsenic, seven waste rock management categories (WRMCs) were initially identified, then reduced to five because significant arsenic leaching occurs for all categories.

Three of the initial WRMCs (designated 5, 6, and 7) were defined as having significant potential for ARD but with different timeframes for ARD generation. WRMC 4 has lower ARD potential, and WRMC 2 is defined as having negligible ARD potential.

Categories 5, 6, and 7 are considered potentially acid generating (PAG) waste rock. Shortly after production begins some of these materials will be placed in the WRF. The PAG 5 category waste rock will be blended with non-acid generating (NAG) materials to provide buffering capacity and can then be disposed of in the WRF following typical dumping procedures. The PAG 7 waste rock will either be sent to the mill or temporarily stockpiled and then placed back into the open pit where it will eventually be submerged. The PAG 6 waste rock will be kept as dry as possible, isolated from the other waste rock within the WRF.

A dumping plan has been developed that separates the PAG 6 materials from the NAG materials within the dump by placing them into isolated “cells” within the Rob’s Gulch area of the WRF. Each cell will be placed in 33 ft (10 m) high intermediate lifts to an overall lift height of 100 ft (30 m). The PAG 6 cell will then be capped with a 3.3 ft (1 m) thick layer of lower-permeability natural materials consisting of colluvium or terrace gravels. NAG materials will be dumped around the PAG 6 rock, isolating it from the final surface of the dump and from the surrounding natural ground.

18.4.3 Construction Plan

The waste rock facility will be constructed from the bottom up. Close attention must be paid to construction of the rock drains and initial lifts as they will serve as the foundation for the remainder of the WRF. The results of the stability analyses of the overall WRF indicate that under static and pseudo-static conditions the WRF meets or exceeds the design criteria.

Waste rock will initially be developed from both the Lewis and ACMA pit areas in the preproduction years. In terms of geotechnical stability, preproduction (Year -1) and the first few years of production are likely to be the most critical in the development of the WRF. During this time there will be little variety in the types of waste rock available for placement, minimizing flexibility in material selection for specific construction purposes. This also represents the time when personnel will have the least experience in handling waste materials and seasonal issues affecting waste rock placement will become apparent. As a result close attention will need to be paid to the development of the WRF during this period.

The WRF will be built from the bottom up along the American Creek valley, with construction of the first lift initiated along the north side of American Creek and into Rob's Gulch at the beginning of the preproduction period. A 270 ft (83 m) wide section of the foundation along the toe of the WRF will be prepared by pre-stripping the organic and ice-rich overburden in areas considered important for dump stability. The foundations for the downstream facilities (Lower CWD) will be cleared, the first stage of the upstream water collection and diversion measures will be constructed, and the first phases of the rock drains in Rob's Gulch and the Unnamed Gulch will also be established during this time.

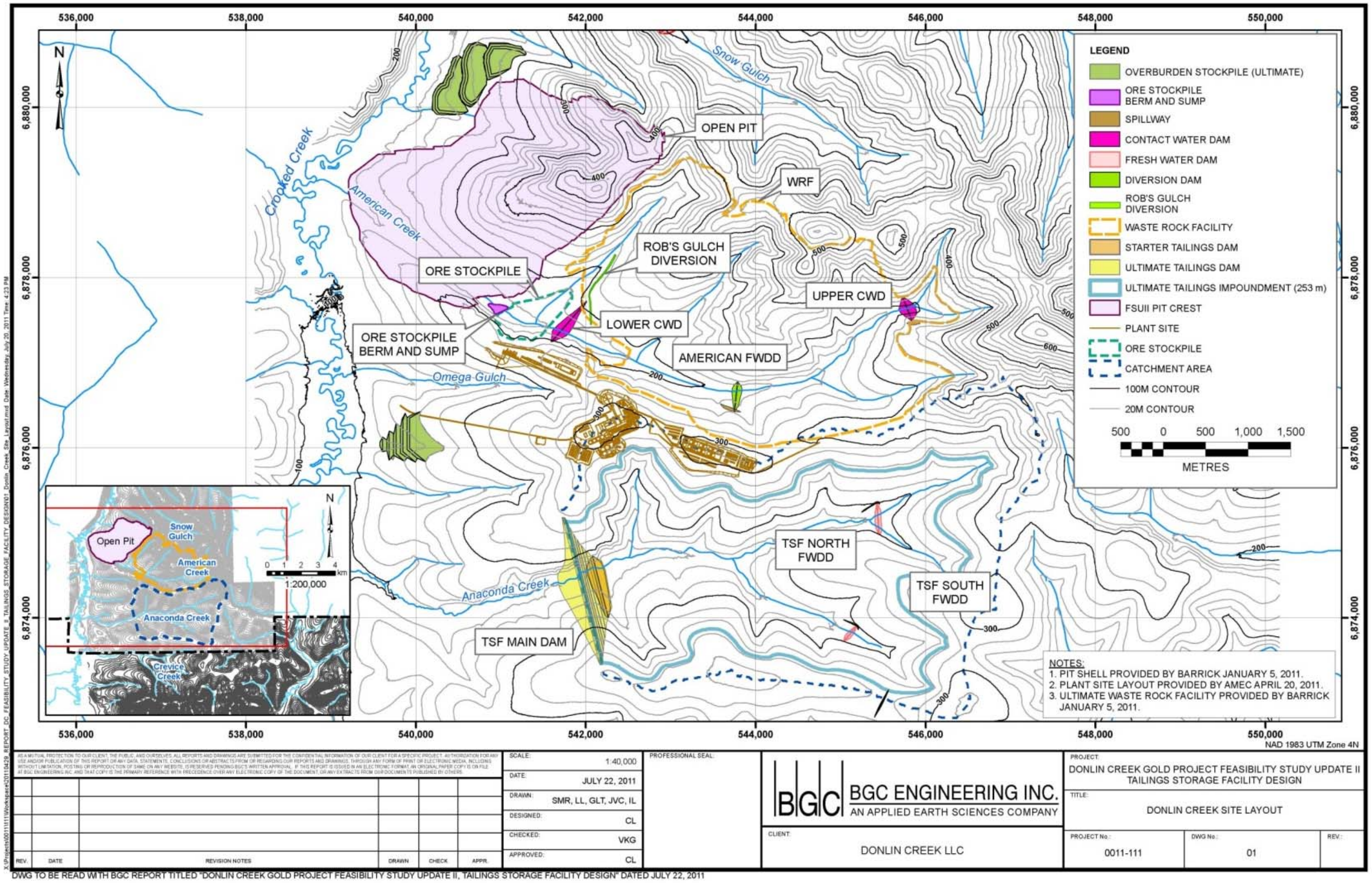
The FSU2 mine plan establishes the northwestern area of the ultimate WRF in Year -1. Subsequent WRF development generally progresses upward until hauls for dump construction become restrictive and it is more cost effective to progress up the American Creek valley. Near the end of Year 2 waste rock will be placed in the bottom of the valley. The WRF then advances eastward, covering most of the valley bottom by Year 4. By the end of Year 6 the waste rock will reach the Upper CWD. The WRF will then be built upward in lifts until reaching its full height in Year 21. Placement of waste rock as backfill in the ACMA pit will begin in Year 20.

Before advancing the WRF into the main drainages (American Creek, Rob's Gulch, the Unnamed Gulch) portions of the primary rock drains will need to be constructed to stay ahead of the WRF development.

18.5 Tailings Storage Facilities

The TSF will be constructed in the Anaconda Creek valley, 2.2 miles (3.5 km) south of the open pit and 1.9 miles (3 km) east of the confluence of Anaconda and Crooked creeks (Figure 18-4). Anaconda Creek flows west through the gently sloping valley bottom before joining Crooked Creek.

Figure 18-4: Location of Tailings Storage Facility



The starter dam is sized to store one year of tailings plus the 200-year return period snowmelt, 24-hour PMP rainfall event, excess water accumulation under average conditions in the site water balance, and emergency freeboard. To meet these requirements, the starter and ultimate dam will be 177 ft (54.0 m) and 462 ft (141.5 m) high, respectively, as measured from the crest to the downstream toe. The ultimate capacity of the lined tailings impoundment will be 334,298 acre-ft (412.35 Mm^3).

Construction of the tailings facility will need to be scheduled with special allowances for seasonal constraints and effects on earthworks and liner placement productivity. Construction will take at least two years and will commence no later than two years before mill commissioning, starting in a winter construction period, assumed to be from November to March. The summer construction period is from May to October. The month of April is assumed to be non-trafficable as the ground thaws. November is also likely to be a transition month from summer to winter, but the trafficability is expected to be adequate to maintain some construction activities.

Surface water management will be required during construction and will comprise conventional best-management practices such as cofferdams, sumps, sediment traps, and pumped diversions.

The proposed hazard classification of the tailings dam in Anaconda Creek is Class I, or "High," according to the *Dam Safety Guidelines* (ADNR, 2005).

18.5.1 Design Considerations

Standard procedures from the following agencies were also applied for the design of the tailings facility and appurtenant structures, where appropriate:

- U.S. Bureau of Reclamation (USBR)
- U.S. Army Corps of Engineers (USACE)
- Federal Emergency Management Administration (FEMA)
- International Congress on Large Dams (ICOLD)
- Canadian Dam Association (CDA).

Key features of the TSF design are as follows.

In accordance with FEMA (2004), the Inflow Design Flood (IDF) is the Probable Maximum Flood (PMF), including allowance for snowmelt, for dams of significant hazard classification. The corresponding IDF is the 200-year snowmelt plus runoff from a 24-hour PMP. The PMP is assumed to occur at the end of the 200-year snowmelt with the ground fully saturated; therefore the entire PMP runs off the catchment area. The TSF will store the full volume of the IDF without discharge.

The TSF will store water in excess of the site water balance under average operating and flood conditions with consideration for the potential of ice loading on the liner.

The tailings facility is designed to contain the IDF without release from the impoundment. Consequently, no spillway will be required during operations. A spillway will be required upon closure, with the peak design flow based on the PMP rainfall event.

The freeboard above the IDF allows for wind-generated wave height, setup, and run-up; for estimated settlements and seismic deformation; and for hydrologic uncertainty, evaluated according to USBR (1981), USACE (1991), and CDA (2007).

The Maximum Design Earthquake (MDE) for the tailings dam is based on the Maximum Credible Earthquake (MCE) determined following the procedures described in USACE (1995). The seismic hazard for the Donlin site area has been evaluated using both deterministic and probabilistic methods of analysis. The recommended MDE for the tailings dam design is characterized by a peak horizontal ground acceleration on rock of 0.36 *g* from a magnitude 7.8 earthquake for the deterministic analysis and 0.44 *g* for the 1 in 10,000-year earthquake for the probabilistic analysis; both were evaluated for the TSF. The recommended MDE for ancillary structures is the 2,475-year return period event characterized by a peak horizontal ground acceleration on rock of 0.25 *g*.

The criteria outlined in Seed (1979) were used to evaluate the suitability of the dam side-slopes under seismic loading conditions. Seed (1979) recommends a horizontal seismic coefficient (kh) of 0.15 *g* for Magnitude 7.8 earthquakes, satisfying a pseudostatic factor of safety greater than 1.15. The Bray and Travasarou (2007) simplified procedure for estimating earthquake-induced slope displacement was used to estimate mean seismic displacements for the tailings dam. A full dynamic analysis for the ultimate tailings dam was completed using the numerical modelling program FLAC. The maximum allowable deformation used for evaluating suitable dam side-slopes corresponds to a maximum crest settlement on the upstream face of 3.3 ft (1.0 m).

The tailings dam will be constructed of compacted rockfill using the downstream method, with a 60 mil (1.5 mm) LLDPE liner on the upstream face. The tailings will be transported to the TSF at a slurry density of 36% solids by weight and will be discharged subaerially around the entire perimeter of the impoundment. Tailings beach slopes of 0.5% above the water surface and 1% below the water surface have been assumed for capacity and deposition planning purposes (Barrick, 2007).

Filter zones are required between the tailings and the rockfill that constitutes the body of the dam for protection in the event of a puncture in the liner. The dimension and gradation of the filters and the flow capacity of the filter zones are designed according to USACE (2000). The flow capacity of the underdrain has been designed according to the Wilkins Equation (1956). A factor of safety of 10 is generally considered acceptable for the underdrain design flow to account for uncertainties in the flows and drain construction and has been applied to the TSF impoundment drains.

Two freshwater diversion dams are required to limit surface water reporting to the tailings dam and are in place during the first three years of operation. During construction, these diversion dams will divert fresh water from the impoundment to facilitate construction of the TSF starter dam and liner placement. The dams will also minimize runoff to the TSF during operations and thus provide a significant control on impoundment volumes during the first three years of operations. Following Year 3, their liners will be removed and the dams re-graded.

A seepage recovery system (SRS), consisting of a pond excavated into bedrock, lined rockfill berms, diversion ditches, and monitoring/collection wells, will be installed immediately downstream of the closure footprint of the tailings dam. The seepage recovery pond and collection wells will provide secondary and tertiary containment, respectively, while the TSF liner provides primary containment.

18.6 Water Management

18.6.1 Water Balance

The site-wide water balance model (WBM) for Donlin was calibrated to predict the precipitation/runoff relationships observed at site. This model uses precipitation and potential evapotranspiration as input parameters and provides volumetric discharge as output.

The WBM for operations assumes that when the tailings slurry is initially deposited in the TSF the settled dry density is 0.69 st/yd^3 (0.82 t/m^3), increasing to 1.051 st/yd^3 (1.249 t/m^3) by the end of operations. The model accounts for the concurrent, slow release of water from the tailings void space as the tailings load increases and pore pressures dissipate (consolidation). Another 52 years is required at closure for full consolidation of the tailings to a final settled density of 1.10 st/yd^3 (1.30 t/m^3). During operations, the average void loss is 4,429 USgpm (1,006 m^3/h).

18.6.2 Construction Water Management Strategy

Water infrastructure required for the construction phase is illustrated in Figure 18-5

American Creek Runoff

Based on geochemical assessment, it is anticipated that runoff from the WRF would need to be treated before discharge. Currently there are no plans to discharge this water, and therefore all water coming into contact with waste rock or exposed bedrock in the mineralized zone must be stored in the Lower CWD during construction. Pre-stripping/waste rock excavations are scheduled to commence in Q2 of Year -1 with mill start-up in Q3 of Year 1. Therefore, there is a 15-month period where runoff coming into contact with the waste rock, the ore stockpile, and the pre-stripped footprint of the pits must be stored in the Lower CWD.

A freshwater diversion dam (American FWDD) upstream of the WRF will limit inflows to the Lower CWD during the 15-month exposure period. Runoff captured in this diversion dam is pumped to Omega Gulch, which discharges into Crooked Creek.

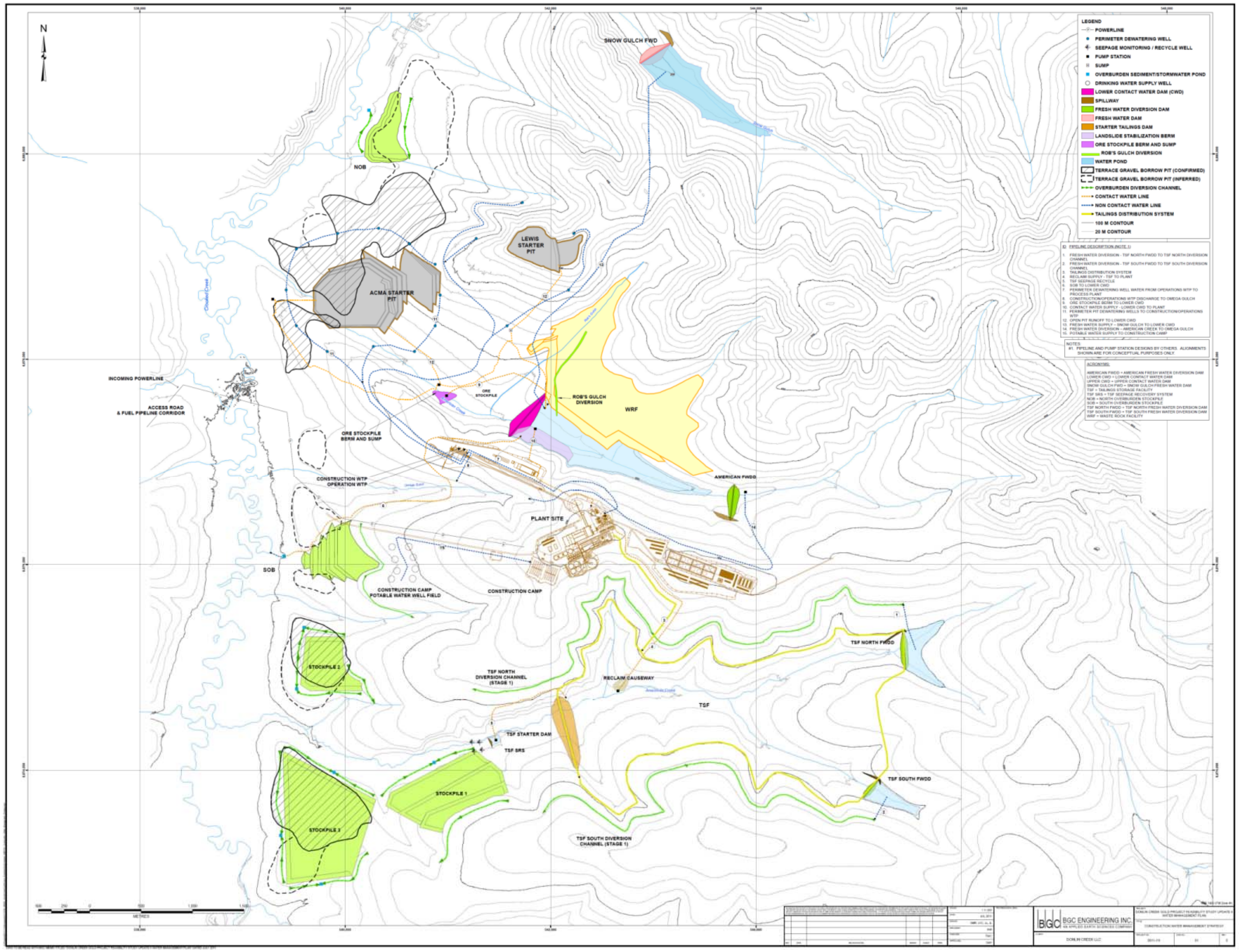
Pit Dewatering Groundwater

Pre-stripping begins in ACMA and Lewis pits in Q2 of Year -1. To achieve pit depressurization targets, pumping from perimeter dewatering wells should begin six months before the start of pre-stripping. The flow of these perimeter wells during the construction period is currently estimated to range seasonally between about 1,286 USgpm (292 m^3/h) and 2,065 USgpm (469 m^3/h).

Ore Stockpile Berm

Immediately downstream of the Lower CWD, contact water will be generated from the ore stockpile, shallow seepage from the Lower CWD, and the lower reaches of Rob's Gulch below a proposed diversion. To capture this contact runoff, a 10 ft (3 m) high berm will be constructed on American Creek, immediately downstream of the proposed ore stockpile.

Figure 18-5: Construction Water Management Layout



Snow Gulch FWD

The Snow Gulch freshwater dam (FWD) is a contingency source of fresh water to the process plant during operations and as such is required for the start of operations. This freshwater dam will be constructed and completed in Year -2, allowing at least 18 months for runoff to accumulate in the dam before it becomes operational. Except when water is being withdrawn from the pond for use in process, the dam will be kept at its maximum storage capacity of 3,243 acre-ft (4 Mm^3).

Anaconda Creek Runoff

The TSF will not be completed until the end of October Year -1, and so minimal water management will be required during construction. The TSF North and South FWDDs will be complete by the end of October Year -2. Water accumulating in these dams will be pumped around the TSF dam via the North and South TSF diversion channels. The pumping configuration used during the construction phase will be the same as that proposed for operations. Pond volumes in both dams will be kept to an absolute minimum with constant pumping to avoid the build-up of water. Both diversion dams are sized to contain the 100-year snowmelt runoff.

Overburden Stockpiles

A number of overburden stockpiles are required to store material that will be used to reclaim the tailings and waste rock facilities. All of these stockpiles lie beyond areas that drain into proposed dams and will need sediment control structures. Overburden stockpiles requiring sediment control are include:

- The north overburden (NOB) stockpile north of American Creek on the east side of Crooked Creek
- The south overburden (SOB) stockpile between the American Creek and Anaconda Creek valleys on the east side of Crooked Creek
- Three stockpiles downstream of the TSF dam.

Runoff from these stockpiles will be managed by intercepting and directing surface runoff toward sediment ponds sized to contain the 10-year return period, 24-hour duration storm. The diversion channels will be sized for the 100-year rainfall event. Two sets of diversion channels are proposed. Upslope diversions will limit runoff to the overburden dumps, while channels on the downslope side will direct surface runoff to the sediment ponds.

Construction Camp Potable Water Supply

The first 500-person phase of the construction camp is scheduled to be complete in late March Year -3. Installation of six potable groundwater supply wells will need to begin as soon as possible (November Year -4) to allow completion prior to start-up. The construction camp will eventually need a total of eight potable groundwater supply wells for the maximum camp capacity of 2,500 persons.

18.6.3 Operations Water Management Strategy

Water infrastructure required for the operations phase is illustrated in Figure 18-6.

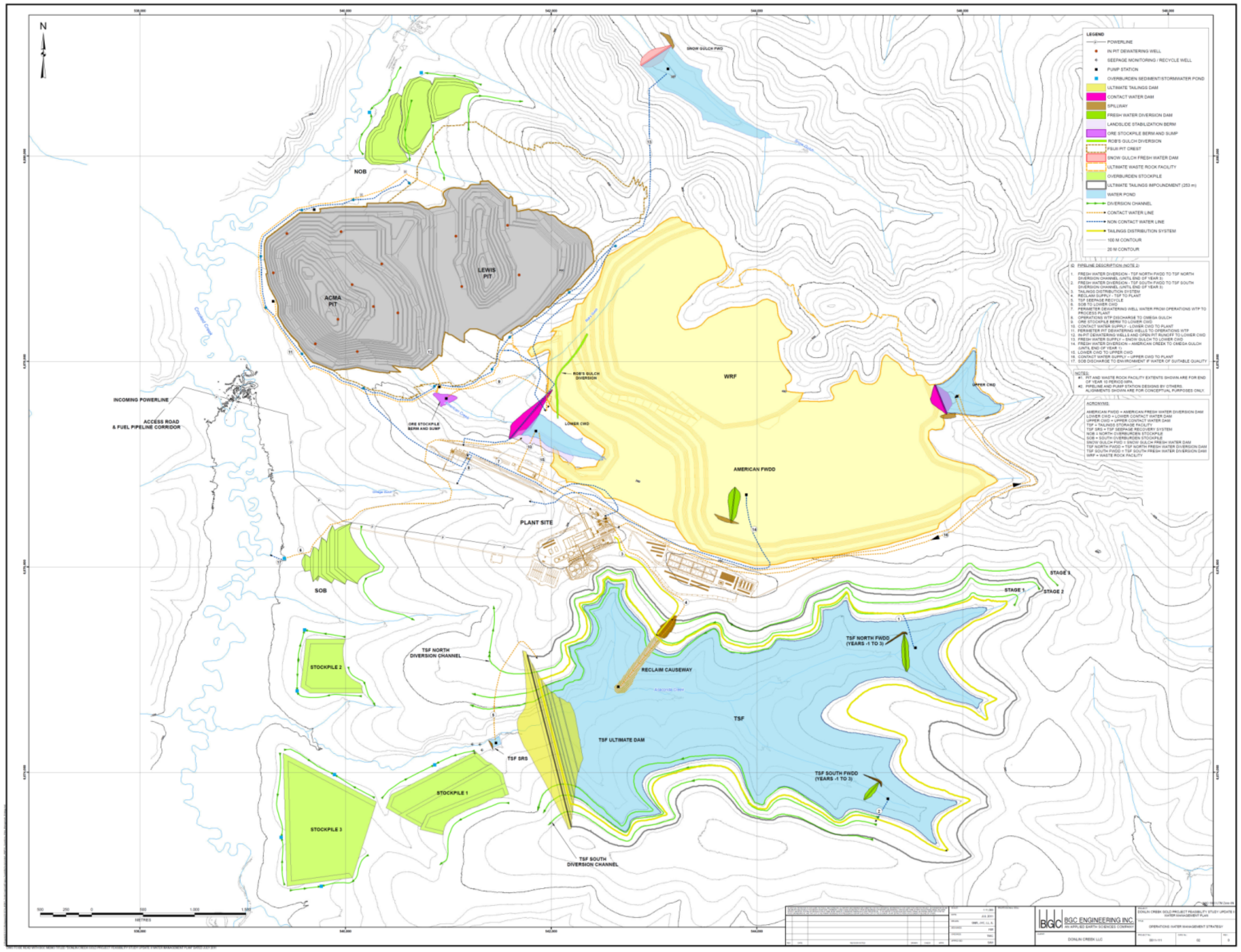
Plant

Water requirements depend on mill feed rates and vary on an annual basis. Based on a mill throughput of 59,000 st/d (53,500 t/d):

- The process plant requires a total water supply of 17,838 USgpm (4,051 m^3/h).
- Mill process losses are 247 USgpm (56 m^3/h)
- The process plant requires a minimum 3,170 USgpm (720 m^3/h) of fresh water; this includes contact water, pit dewatering water, and non-contact water, but does not include TSF reclaim water.
- Reclaim water from the TSF is pumped back to the process plant at a minimum rate of 9,397 USgpm (2,134 m^3/h).
- Reclaim water is maximized at 14,668 USgpm (3,331 m^3/h) when TSF pond volumes exceed 405 acre-ft (0.5 Mm^3).

In years with average and below-average precipitation, the contact water ponds and pit dewatering system will not be able to meet the year-round fresh water requirements for the plant. In this case, additional water will be obtained from Snow Gulch. The priority for use of fresh water is as follows: contact dam water, pit dewatering groundwater, and then water from Snow Gulch.

Figure 18-6: Operations Water Management Layout



Tailings Storage Facility

Runoff to the TSF will be minimized by the construction of staged diversion channels and two upstream diversion dams. The diversion channels will remain in operation until Year 17, while the diversion dams will be removed after Year 3. Without diversions, the TSF has a total catchment area of 3,756 acres (1,520 ha). Even with the diversion structures, however, water is expected to build up in the TSF under average precipitation conditions. This water will be managed through storage during mine operations and by pumping the excess to ACMA pit at closure. Flood storage for the TSF includes runoff from the 200-year snowmelt and 24-hour Probable Maximum Precipitation (PMP)—a total of 6,688 acre-ft (8.25 Mm^3). An additional freeboard of 6.6 ft (2 m) is added to this flood storage.

A seepage recovery system (SRS) downstream of the TSF dam will serve as backup to the TSF liner. The SRS will be available to capture potential seepage from the TSF and flow from the underdrain, and pump it back to the TSF if necessary.

Lower and Upper Contact Water Dams

The Lower and Upper CWDs will be located in American Creek with the objective of managing contact water runoff from the WRF and open pit. The Lower CWD will receive runoff from a variety of sources:

- Surface and seepage runoff from the waste rock
- Runoff from undisturbed ground upgradient of the WRF
- Surface runoff within the open pit footprint
- In-pit dewatering wells and horizontal drains from the open pit
- Runoff collected behind the ore stockpile berm
- Runoff collected in a sediment pond downstream of the SOB stockpile.

During operations, the Lower CWD will have sufficient capacity to store the 24-hour PMP and a maximum operating pond of 811 acre-ft (1 Mm^3). Based on stochastic water balance modelling, the Upper CWD will have a maximum storage capacity of 2,432 acre-ft (3 Mm^3). Because the pond for the Lower CWD has sufficient capacity to store the 24-hour PMP, no spillway is proposed. The Upper CWD, however, will have a spillway with sufficient capacity to convey the Probable Maximum Flood (PMF).

Open Pit

Runoff into the open pit will be managed with a combination of horizontal drains, surface water ditches, pumps, and sumps. The proposed pit drainage system consists of two major components. The first system will be constructed around the rim of the ultimate pit and includes a perimeter ditch and three pumping stations.

The second system will consist of pumps for lifting water out of the pit. Ditches will be constructed along all roads and on other strategically located benches while excavating the pits. These ditches will intercept surface and horizontal drain runoff and direct the water into sumps. The sump water will then be picked up with a primary pump and discharged into the perimeter system.

Additional information on the pit water management system is provided in Section 16.9 of this Report.

South Overburden Stockpile

The south overburden (SOB) stockpile will consist of waste terrace gravels and colluvium sourced from the pit excavation. Because these materials are potentially metal leaching, surface runoff from the SOB needs to be captured in a pond and sent to the Lower CWD during operations. The pond is designed to store the runoff associated with a 24-hour, 10-year return period rainfall event.

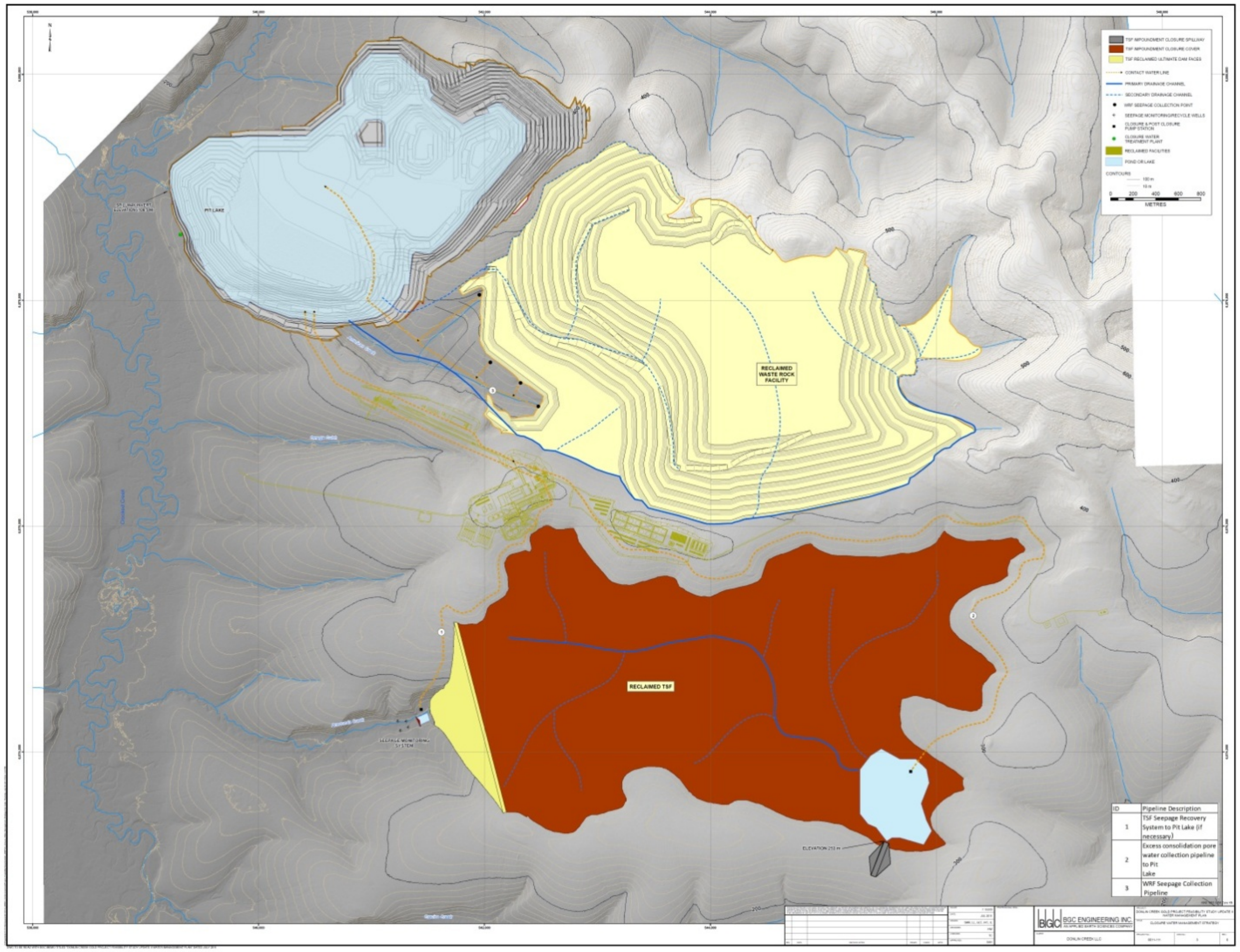
18.6.4 Closure Water Management Strategy

Water infrastructure required for the closure phase is illustrated in Figure 18-7.

The overall site plan for closure includes the following components:

- A covered and reclaimed TSF
- A contoured TSF surface so that runoff is to the southeast corner, where a large rock cut will convey flows into Crevice Creek
- Breaching, liner removal, and grading of the Snow Gulch FWD, Lower CWD, and Upper CWD
- A contoured, covered, and reclaimed WRF
- A pit lake in the partially backfilled ACMA and Lewis pits
- Once the pit lake is near capacity; treatment and discharge of pit lake water to Crooked Creek.

Figure 18-7: Closure Water Management Layout



18.6.5 Potable Water

The source of water supply for the construction camp, and later the permanent accommodation complex and the plant site potable water systems, will be an array of eight deep wells south of Omega Gulch, near Crooked Creek.

Water supply will be pumped to freshwater storage tanks, and will be treated prior to consumption. A potable WTP, including storage tanks and a distribution system, will be provided.

18.6.6 Fire Water

Fire protection water supplies for the construction camp and permanent accommodation facility will be provided from the freshwater storage tanks. The tanks will have a reserve supply for fire protection capable of providing 500 USgpm (1,900 L/min) for one hour, in accordance with NFPA requirements for ordinary hazard occupancies.

18.7 Bethel Marine Terminal

Bethel is about 56 nm (nautical miles) (105 km) upriver from the mouth of the Kuskokwim River where it empties into Kuskokwim Bay. It is the only port on the Kuskokwim River accessible to ocean barges.

Three adjacent parcels of land downstream from the Port of Bethel have been identified as possible sites for a general cargo barge terminal dedicated to the Project. All have road access and could be connected to the local power grid. While all three sites remain under investigation, as with prior studies, one of these sites, which is known as the Knik Construction Yard, a division of Lynden Transport, was used as the basis of the FSU2 evaluations.

Ocean barges will be unloaded of their cargoes, which will either be transhipped directly to river barges or placed in storage to wait for an available river barge. The salient features of the terminal will include:

- A berth for ocean barges
- A berth for river barges
- A roll on/roll-on berth
- A storage area for containers and break-bulk cargoes
- Buildings and ancillary infrastructure and services.

To provide enough space for five ocean barge loads, the terminal yard will need to be about 12.4 acres (5.1 ha) in area. With a further 3.5 acres (1.4 ha) required for the wharf, buildings, access roads, and ancillary services and facilities, the terminal will have a total area of 16.0 acres (6.5 ha).

In 2017, the peak construction year, up to 30 ocean barges will call at Bethel and unload construction cargoes.

The wharf structures, fender systems, and mooring systems will be designed for a minimum service life of 25 years. Other conditions factored into the design of the wharf include site conditions, ice, environmental conditions, a corrosion allowance, and the vessels that will use the terminal.

Electrical power for yard lighting and office needs at the port will be provided from the local grid. Potable water will be obtained from a well, and sewage will be sent to a septic field. Water for fire protection will be pumped from the river through a temporarily deployed pump and hose into a heated, insulated, above-ground 237,800 USgal (900 m^3) dedicated firewater storage tank.

To accommodate the increase of about 40 USMgal (151.42 ML) in annual throughput attributable to the Project, storage capacity at the existing tank farms in Bethel will be increased by 6 USMgal (22.71 ML).

The following mobile equipment will be required at the Bethel port:

- 2 mobile harbour cranes
- 1 wheel loader
- 1 forklift
- 6 forklift – 30 t capacity
- 2 crew cab 4x4 pickup trucks
- 6 container tractors
- 12 container trailers
- 1 highboy trailer
- 1 vehicle maintenance truck.

18.8 Kuskokwim River Dock Site

The Jungjuk dock site is located on the north bank of the Kuskokwim River, 8 miles (13 km) downstream of the village of Crooked Creek and 194 miles (312 km) upstream of Bethel. A viable site has been identified 1,000 ft (305 m) upstream of the mouth of Jungjuk Creek. A staging area of up to 24.7 acres (10 ha) can be developed on adjacent uplands immediately north of the dock site.

The port will be developed on a south-facing slope rising from the Kuskokwim River to ridges that crest at an elevation roughly 1,000 ft (305 m) above river level at a distance approximately 2 miles (3.2 km) from the river's north bank. Elevation at the port site is roughly 131 ft (40 m) above sea level, on the "high ground" between the Jungjuk Creek drainage to the west and a minor stream that discharges into the Kuskokwim River about 1,640 ft (500 m) upstream of the mouth of Jungjuk Creek.

Facilities at Jungjuk will include two river barge berths, a barge ramp, container-handling equipment, seasonal storage for containers, break-bulk cargo, and fuel, and barge-season office/lunchroom facilities.

The wharf structures, fender systems, and mooring systems will be designed for a minimum service life of 25 years. The river barges will arrive in tows of four (two-by-two configuration). River barges will be unloaded of their cargoes, which will be placed in storage to await onward transport to the mine site by truck.

The terminal yard will be capable of storing about 1,000 TEU containers.

Containers and other cargo will be trucked to the mine throughout the summer barging season. Fuel will be off-loaded from barges and temporarily held in a storage tank before being pumped to B-train trucks for transport to the main fuel storage facility at the mine site.

Electrical power for yard lighting and office needs at the port will be provided by diesel generators. Potable water will be obtained from a well, and sewage will be sent to a septic field. Water for fire protection will be pumped from the river through a temporarily deployed pump and hose into a heated, insulated, above-ground 237,800 USgal (900 m^3) dedicated firewater storage tank.

The following mobile equipment is required for port operations:

- 2 mobile harbour cranes
- 1 wheel loader

- 4 container forklifts
- 4 crew-cab pickup trucks
- 5 semi-trailer tractors
- 1 4 x 4 pickup truck
- 8 container trailers (bomb carts)
- 4 terminal tractors
- 1 highboy trailer
- 1 fire truck.

18.9 Power and Electrical

Electric power for the Donlin Gold Project site will be generated from a natural gas-fuelled combined-cycle gas turbine (CCGT) power plant and standby simple-cycle gas turbines (SCGTs). The CCGT power plant conceptually consists of two equal halves, each with two gas turbines, two once-through steam generators (OTSGs), and one steam turbine (ST) with an air-cooled condenser (ACC). Standby power generation will consist of two simple-cycle gas turbines (SCGTs). The total generation facility is nominally rated 240 to 247 MW (summer/winter ratings).

To minimize electrical distribution costs and load losses, the power generation facility will be strategically located adjacent to the two major process electrical loads: the oxygen plant and the grinding mills.

The total connected electric load is estimated to be 227 MW and the average running load approximately 153 MW with a peak load of 184 MW. Motors constitute most of the electrical loading. The largest are the grinding mill motors, which are gearless (wrap-around) type that use cyclo-converter variable-speed drives with soft-start features. The oxygen plant will have three large synchronous motors that use a load commutated inverter (LCI) controller to provide motor soft-start to minimize the stress on the power supply during starting and to reduce voltage flicker.

Power will be distributed to the main process areas by metal-clad cable feeders in trays mounted on racks routed to the local electrical rooms through utilidors, process buildings, and conveyor galleries. This will include the grinding, oxidation, leach, refinery, and coarse ore reclaim areas. Overhead power lines will be run to the more remote areas, including the primary crusher, the water system, pumping, tailings, mine open pit electric shovels, and pit dewatering sites. Each process area load centre will have associated electrical rooms, distribution transformer(s) to bring the voltage down to utilization levels, secondary distribution centres and switchgear, and motor/feeder distribution equipment. To minimize costs the electrical rooms will be pre-fabricated off-site with all electrical equipment installed, pre-wired, and tested prior to shipping to site.

Power will be provided to the permanent accommodation facility from the mine/process plant site on a 25 kV power line. In addition, a local emergency power diesel genset will be installed to provide backup power in the event of a failure of the pole line. There are no major facilities located at the airport. The airstrip will have a 100 kW genset to run fuel pumps and lights as needed and will not be connected to the site power supply.

Facilities at the port site have been considerably downsized since the FSU. The port site will have an independent, stand-alone power generation facility equipped with two 600 kW (one backup) gensets. The power station will be supplied completely self-contained with all controls and ancillary equipment, ready for installation on site. The unit will include all accessories, radiators, cooling fans, exhaust systems, air and fuel filters, engine control panels, alternators, controls, starting batteries, and chargers within a modular enclosure suitable for shipping and travel on standard highways. The module will be complete with mounting skids, access (maintenance, equipment, and man-doors), ventilation, interior and exterior lighting, general power receptacles, provision for grounding, fuel day tank, supply and return fuel lines to engines, and circuit breaker with control, protection, indication, and alarms.

The port site will also include a control module unit to house the electrical switchgear, synchronization controls, and operator interface equipment. The three modular units will be arranged together and will be installed on concrete foundations.

18.10 Gas Pipeline

The 12-inch (356 mm) natural gas pipeline proposed for the Project would be approximately 313 miles (503 km) long. It would commence at the west end of the Beluga Gas Field, approximately 30 miles (48 km) northwest of Anchorage at a tie-in near Beluga located in the Matanuska-Susitna Borough and would run to the mine site (refer to Figure 2-2). Donlin Gold advised AMEC that the proposed pipeline route crosses an area with no significant preexisting infrastructure and does not follow any existing utility corridors.

The pipeline would receive booster compression supplied by one compressor station located at approximately mile post 5. No additional compression along the pipeline route would be required. The pipeline would transport approximately 50 million standard cubic feet per day (mmscfd) of natural gas (1,415,842 m^3pd).

The pipeline would be regulated by the US Department of Transportation (DOT) under Title 49 of the *Code of Federal Regulations,* Part 192 – Transportation of Natural Gas and Other Gas by Pipeline: Minimum Federal Safety Standards (49 CFR 192). The pipeline would be designed, constructed, and operated in accordance with the applicable requirements of 49 CFR 192 and would incorporate pig launching and receiving facilities (receipt, midpoint, and delivery), approximately 19 mainline block valves (MLVs), cathodic protection, leak detection, a supervisory control and data acquisition (SCADA) system, and possibly a fiber optics cable to the mine site.

The design life of the proposed pipeline is 30 years; however, design life may be able to be extended with additional maintenance and repair.

18.11 Fuel

18.11.1 Diesel

The maximum diesel fuel storage capacity at the plant site is estimated to be 37.5 USMgal (142 ML). Storage for 2.5 USMgal (9.5 ML) of fuel will be provided at the Jungjuk port site, but this is only intended for short-term use while the fuel barges are being unloaded during the summer, and is not included in the design of the overall site storage capacity.

The plant site fuel storage facilities are sized for a ten-month supply plus one month of contingency for the mine fleet and site mobile equipment. Fifteen fuel tanks, each having a capacity of 2.5 USMgal (9.5 ML), will be installed within a HDPE-lined and bunded tank farm approximately 1,970 ft (600 m) east of the plant site and at an elevation of approximately 981 ft (299 m). This arrangement provides gravity-assisted flow to the plant area at an average elevation of 952 ft (290 m) and to the truckshop at elevation 679 ft (207 m).

During the summer shipping season, fuel tanker trucks will transfer diesel fuel from the 2.5 USMgal temporary storage tanks at the Jungjuk port to the plant site fuel farm. From there, the fuel will be distributed to various day tanks around the site.

18.11.2 Natural Gas

Natural gas will be supplied to the various buildings at the plant site for heating through an underground network of pipes. The main distribution line will extend 4.5 miles (7.2 km) along the main access road to the permanent accommodation complex and supply fuel for the forced-air heating system and for the cooking appliances in the camp kitchen.

The supply of construction diesel fuel and propane will be delivered during the summer shipping season. The diesel fuel will be stored in temporary 500 m^3 tanks until the first two 2.5 USMgal storage tanks are completed and can be filled with fuel. The propane will be stored in fourteen 10,000 USgal (38,000 L) mobile tanks.

18.12 Comment on Section 18

In the opinion of the QPs, the infrastructure requirements and current designs for such infrastructure for the Project are appropriate to the mine and process plan, and to the Project setting:

- In general, the design and construction of the mine site infrastructure will be relatively straightforward, although the scope of the work is extensive, especially in terms of the water systems. In addition, the Project involves several development sites considerable distances apart, incurring high infrastructure costs to provide interconnecting roads, pipelines, services, and utilities. The decision to use material from the plant site excavation as a borrow source for constructing the tailings dams is an effective way to reduce the site preparation costs
- There are known to be intermittent areas of permafrost and poor ground conditions at the various facility sites that could affect foundation design and site preparation. This risk can be mitigated by carrying out more thorough geotechnical investigations at all sites
- A local source of concrete aggregate has not been identified. Rock excavated from the plant site ridge is considered marginal for use as structural backfill, based on laboratory testing. If no local source of aggregate or backfill can be identified, suitable material may have to be brought to site
- Off-site infrastructure will be arranged, designed, and constructed using techniques that are proven to result in functional and durable facilities suited to their remote location and cold environment. For the detailed design phase of the Project, further geotechnical investigations should be done along the access road and at Jungjuk and the Bethel cargo terminals. In addition, river water levels should be monitored through future barge seasons to gain a better understanding of year-to-year variability
- The construction schedule for the initial phase of the TSF is aggressive. In particular, the ability to install the liner system in the specified schedule may be challenging. Any significant weather or permitting-related delays could affect successful completion of the TSF on schedule. The consequence of not meeting the current construction schedule is deferral of mine start-up

- Water management is required during construction, operations (27 years), and closure. For the construction phase the critical infrastructure components are the WRF and the ACMA and Lewis pits, which begin development in Year -1. Runoff from these areas is contact water that cannot be discharged to American Creek unless it meets discharge water quality standards. This runoff must therefore be stored in the Lower CWD for use during operations. An upstream freshwater diversion dam will minimize the area draining to the Lower CWD during construction and the first year of operations. Water impounded behind the Lower CWD will be the primary source of water to the process plant during the start-up period. Start-up water requirements are 2,513 acre-ft (3.1 Mm^3) of non-turbid water. Under very dry conditions, there could be a shortage of freshwater supply during the first few years of operations. To mitigate this potential, runoff water will be impounded behind a freshwater dam constructed in Snow Gulch during the preproduction period

- The construction schedule for the Lower CWD is aggressive, with a great deal of work to be completed in a short duration. Weather delays could affect completion of the dam on schedule. The consequences of not meeting the current construction schedule are deferral of any construction activity upstream of the Lower CWD that generates contact water and possible delays in revenue generation

- In the American Creek watershed, a lake will be allowed to develop in the mined-out pits. The pit lake will receive runoff from the TSF reclamation activities (including the supernatant pond volume remaining at the end of operations), the covered WRF, and undisturbed areas upslope of the WRF. Seepage from the reclaimed WRF will be collected and piped to the bottom of the pit lake to encourage pit lake stratification. Modelling of the pit lake geochemistry indicates that treatment is required before ACMA pit lake water can be discharged to Crooked Creek. The plant has been designed for a maximum capacity of 6,604 USgpm (1,500 m^3/h) based on an average annual operating period of six months; longer operating periods will be required in years when precipitation is above average

- There is a potential risk that the mine may generate more surplus water than the design conditions, increasing the amount of TSF pond water. The potential implications include increased water treatment costs (and potential permitting issues), increased TSF operating costs for ice mitigation, and an overall increase in the vulnerability of the liner system integrity during operations

- Seasonal constraints on construction of the PAG cover could limit the ability to use this portion of the WRF during certain periods of the year. This could require temporary stockpiling of the PAG materials and changes to the mine plan

- The electrical power generation and distribution system has been designed to meet the requirements of the proposed loads, the site location, and facility layout. The power plant is based on combined-cycle gas turbines and steam turbines with associated equipment and backup simple-cycle gas turbines. This is a complete system with built-in redundancy providing reliable power for safety during emergencies and unscheduled equipment outages. The modular nature of the proposed electrical infrastructure provides the flexibility needed to allow the configuration to be adapted to meet final design electrical requirements
- In a typical year, the Donlin mine will consume about 115,000 st (105,000 t) of general cargoes and 40 USMgal (152 ML) of fuel.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Marketing Partnership Agreement

The Limited Liability Company Agreement of Donlin Creek LLC dated 1 December 2007 (the LLC Agreement) provides for the members of DCLLC (the Members) to take in kind their respective shares of the gold production, which they can then sell for their own benefit.

The LLC agreement further 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.

19.2 Gold Marketing

NovaGold's portion of gold production is likely to be sold on the spot market, by marketing experts retained by or on behalf of NovaGold. 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, NovaGold would not be dependent upon the sale of gold to any one customer. Gold could be sold to various gold bullion dealers or smelters on a competitive basis at spot prices.

NovaGold expects that terms contained within any sales contract that could be entered into would be typical of, and consistent with, standard industry practices, and be similar to contracts for the supply of gold elsewhere in the world.

19.3 Comments on Section 19

In the opinion of the QPs, NovaGold will be able to market gold produced from the Donlin Gold Project. Sales contracts that could be negotiated would be expected to be within industry norms. However, the majority of production would be expected to be spot marketed.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Baseline Studies

Baseline studies completed in support of Project development are as indicated in Table 20-1.

Baseline data were collected to support Project design; to determine and implement environmental controls to mitigate impacts; and to sufficiently characterize the environment in support of permit applications and environmental impact assessments. The environmental baseline data will also provide a reference point against which environmental conditions can be evaluated during operations to facilitate early detection of potential changes that may occur during Project development and future operation.

20.2 Environmental Issues

Based on Project design concepts developed through the prefeasibility and feasibility engineering work and on the results of four years of community interaction, several key environmental issues of concern have been identified for the Project; these areas and the planned mitigation measures are summarized in Table 20-2.

Donlin Gold is of the opinion that these issues have been, or can be, addressed and mitigated through a combination of good baseline data collection, diligent engineering and Project design, and thorough public consultation.

20.3 Closure Plan

Reclamation will begin during construction, when topsoil and overburden stockpiles and cut-and-fill slopes are stabilized. Significant reclamation will be carried out immediately after construction, particularly at material borrow sites and where opportunities exist to replace disturbed wetland and enhance aquatic and terrestrial habitat. Reclamation will also be performed concurrently with mining and with the cessation of mining and milling operations.

Area and component-specific reclamation plans governing actual reclamation activities will be developed further as Project designs become more refined and alternatives are identified during permitting. Alternatives will be evaluated and incorporated into the reclamation plan where appropriate at the commencement of permitting and throughout the operational phase of the Project.

Table 20-1: Environmental Baseline Studies (1995 to 2010)

Baseline Study	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2008	2009	2010
Water Quality																
Surface Water Quality	X	X	X	X	X	X		X	X	X	X	X	X	X	X	X
Groundwater Quality									X	X	X	X	X	X	X	X
Air Quality																
Meteorological Data	X	X	X	X	X	X			X	X	X	X	X	X	X	X
Precipitation				X	X	X				X	X	X	X	X	X	X
Ambient Air									X	X		X	X	X	X	X
SODAR														X	X	
Aquatic Studies																
Bio-monitoring/Tissue Sampling				X	X					X	X	X	X	X	X	X
Spawning Surveys		X	X	X	X					X	X	X	X	X	X	X
Resident Fish Surveys			X	X	X					X	X	X	X	X	X	X
Terrestrial Wildlife																
Habitat Mapping										X	X		X			X
Wildlife Surveys										X		X	X	X	X	X
Avian Studies																
Avian Surveys										X		X	X	X	X	X
Wetlands Program																
Wetlands Delineation		X	X	X					X	X	X	X	X	X	X	X
Waste Rock Characterization																
ARD/ML Studies			X	X	X					X	X	X	X	X	X	X
Cultural Studies																
Cultural Site Surveys										X		X	X	X	X	X
Socioeconomics									X			X	X	X	X	
River Studies																
River Use/Fishing Activity Surveys												X	X	X	X	
Marine/River Wildlife Surveys												X	X	X	X	
Erosion Studies												X	X			
Barge Studies													X			
Other Programs																
Mercury Baseline												X	X	X	X	X
Noise Surveys											X					

Table 20-2: Key Environmental Issues

Issue Identified	Mitigation Measures Proposed
Mercury	Collection of detailed mercury baseline information; the Kuskokwim region has naturally elevated levels of mercury. Ongoing modelling studies based on a combination of local, regional, and global mercury measurements State-of-the-art engineered mercury abatement controls have been designed for the milling operations. These include emission controls on the autoclave, carbon regeneration kiln, electrowinning cells, and mercury retort. Stakeholder discussions in ongoing community outreach efforts
Cyanide	The Project is designed to adhere to the International Cyanide Management Code (ICMC) for the transport and use of cyanide. Engineering criteria include use of purpose-built stainless steel ISO (International Standards Organization) containers for the transport of solid sodium cyanide briquettes, incorporation of a cyanide detoxification system and specific cyanide containment areas, and monitoring plans.
Water Management	The Project design has addressed specific water management areas, including a detailed water balance model, design of containment structures with appropriate freeboard allocations, freshwater diversion structures, water treatment systems, a fully-lined tailings disposal facility, and inclusion of flexibility in water management designs and for treating excess water during operations, as necessary. Following closure a pit lake will develop over a period of many decades. The current design provides for water treatment in-perpetuity following the filling of the pit lake to ensure compliance with water quality standards.
Waste Rock and Stormwater Management	The engineering design includes classification and segregation of waste rock. Rock classified as having acid rock drainage potential will be backfilled into the pit at a location below the eventual water table and a dedicated cell constructed within the waste rock facility will handle additional quantities of waste that cannot be blended or backfilled into the pit. Concurrent reclamation will be practiced over the waste rock facility. All stormwater runoff from related facilities will be captured in a contact pond, or pumped to it, and be used during operations. At closure, all runoff and seepage will be collected and managed in the pit.
Transportation	Engineering considerations include incorporating a natural gas pipeline component into the FSU2 design to fuel power generation at site; using shipping containers that exceed industry standards for the transport of hazardous materials; using custom-designed double-hull barges for fuel transport; implementing modern scheduling and navigation aids on the Kuskokwim River; developing detailed ocean, river, and land response plans including training of personnel. Additional considerations include developing communication strategies to provide timely advance warning of barge passage on the river

Issue Identified	Mitigation Measures Proposed
Flora and Fauna	Consideration of impacts on wildlife from the tailings storage facility include incorporation of high-density polyethylene (HDPE) or synthetic netting material, and regular inspection and sampling protocols. Consideration of impacts on wetlands will be included in the Project permitting plan.

The design of long-term water management systems is an integral part of planning for site closure, particularly with regard to the tailings storage facility (TSF), waste rock facility (WRF), and contact water dams (CWD). Closure of these facilities, in turn, involves post-closure water monitoring, maintenance, and treatment systems to ensure that water discharged to the environment meets all applicable water quality criteria. The lake to be formed in the mined-out ACMA pit is the central feature of all post-closure water management in the foreseeable future.

Surface water and groundwater monitoring systems for process components will remain in place up to and possibly beyond 30 years, depending on compliance history and until each specific facility has been stabilized, physically and chemically, to the satisfaction of the applicable state and federal regulatory agencies. Once physical reclamation begins, temporary diversions and sedimentation control systems will be constructed and routinely monitored. These systems will be cleaned, repaired, and modified as necessary.

Long-term or permanent diversions, water treatment, physical barriers, and signage will be monitored and maintained as needed until all closure standards are met, reclamation surety has been released, and property management reverts to the landowner. The decision on what constitutes final closure and the release of any outstanding financial surety will require the concurrence of ADNR, ADEC, and applicable federal agencies.

20.3.1 Water Treatment Plant

A post-closure water treatment plant (WTP) will be built at the site. The proposed WTP is based on the projected pit lake water quality and design water flows, and is essentially similar to that proposed during the first feasibility study update. Testwork conducted from 2006 to 2008 demonstrated that arsenic and manganese in pit dewatering water, which has chemistry similar to the pit lake water at closure, could be treated to meet relevant water quality standards by using a combination of iron co-precipitation, oxidation, pH adjustment, and solids removal. Testwork with a manganese-dioxide-based filter, equivalent to manganese greensand, also demonstrated reduction of manganese to low concentrations.

With the exception of selenium, all metals exceeding standards will be removed through conventional chemical precipitation technology. Treatment for sulphate has been eliminated in the FSU2 because updated predictions of pit lake water quality by Lorax have found that sulphate concentrations will be well below the most stringent Alaskan water quality standards and therefore will not require treatment. Selenium will be treated in an Octolig resin column package. The sludge from the WTP will be chemically stable and will be sent to the bottom of the pit lake for final disposal.

More testwork will be conducted during permitting to confirm the final design and flowsheet selection for the WTP. The results of ongoing bench- and pilot-testing will also be used to update the water treatment process design for the post-closure WTP.

20.3.2 Tailings Storage Facility

Several years before the end of operations, the tailings deposition procedure will be modified to direct the operating pond toward the southeast corner of the TSF. This will be done in anticipation of closure, when the tailings runoff will be directed to Crevice Creek through the closure spillway after runoff water meets water quality standards; this is anticipated to take approximately five years. Construction of the closure cover on the TSF is expected to take approximately five years after initial tailings consolidation (approximately one year after operations have ceased), and final tailings consolidation is expected to take approximately 52 years. At closure, the TSF water will be pumped back to the ACMA pit through the reclaim pipeline.

A closure cover over the tailings impoundment will be required to minimize groundwater interaction and reduce salt mobilization and subsequent transpiration. Surface runoff on the cover will be directed to a lined pond in the southeast corner of the reclaimed TSF. The downstream face of the main tailings dam will be flattened to 3H:1V at closure to accommodate placement of the required colluvium and organic cover material.

20.3.3 Waste Rock Facility

The WRF has been designed to maximize concurrent reclamation, minimize the effects of PAG materials, add flexibility to the site water balance, and optimize the cost of closure. The facility will be constructed in lifts and will be reclaimed as soon as practical after each lift or portion of the facility is complete. After active dumping ceases on each lift, the slopes will be regraded to less than or equal to an overall 3H:1V slope.

A series of channels is required to collect and convey runoff from the surface of the reclaimed WRF to the pit lake during the closure period, as summarized below. In addition, seepage from the WRF will be collected and piped to the bottom of the ACMA pit lake.

20.3.4 Roads and Airstrip

Under both Donlin Gold's corporate standards and regulatory standards, the mine site roads will need to be reclaimed. However, the Jungjuk Creek access road and on-site access roads required for use by monitoring personnel will remain into the foreseeable future after mine closure is initiated. The airstrip will also remain as a long-term asset and is therefore not proposed to be reclaimed and is not included in the reclamation cost estimate.

20.3.5 Foundations and Buildings

With the exception of the WTP, buildings must be removed from their foundations and the debris buried on site or transported elsewhere, per regulatory requirements. Once the buildings are demolished, the foundations must also be broken up and removed to prevent them from being an impermeable impediment to natural percolation of meteoric waters. A minimum thickness of cover will be established over the buried debris to ensure that it remains below surface into the foreseeable future.

20.3.6 Waste Disposal

Non-hazardous construction debris will be placed in the pits or used to fill subsurface voids exposed during the demolition of facilities.

20.3.7 Port Facilities, Access Road, Airstrip, and Personnel Camp

The Jungjuk port facilities will be reclaimed, leaving only a small barge landing area and the access road to the mine site. All mine support facilities except the airstrip and a small camp to support post-closure activities will be removed and reclaimed.

20.3.8 Mobile Equipment

Logistical constraints (access road and barge) preclude the decommissioning and removal of the mobile equipment fleet from the site for the purposes of the FSU2. Therefore, this equipment will be buried in the WRF at closure.

20.3.9 Trust Fund

Alaska state legislation requires that a "Trust Fund for Reclamation, Closure & Post-Closure Obligations" (the Trust Fund) be established. That fund must be sufficient to generate sufficient cash flow to cover all reclamation, closure, and post-closure costs, including maintenance of the spillway from the TSF, employee severance payments, capital to construct the WTP for perpetual water treatment, and associated facility and access maintenance, etc.

A model was developed for this Trust Fund calculation. The funding amount is estimated at $8.6 million provided annually over the construction and operating period, for a total of $274 million (2011 dollars on a real basis). The following assumptions were made in determining this annual funding requirement:

- Income of 5% per annum is estimated to be earned on the cumulative trust fund. This discount rate is recommended by Barrick's Treasury department based on a 30-year risk-free rate of ~ 2.8%.
- All operating expenditures for monitoring, water treatment, etc., and all capital expenditures are adjusted for inflation at 2.56% based on values used internally by Barrick for its long-term forecasts for Provision for Environmental Reclamation (PER). The long-term pre-tax nominal rate is 4.09% and the long-term pre-tax real rate is 1.53%. The difference between these two rates is the inflation rate of 2.56%.
- The WTP will operate about six months of the year.
- Annual costs are estimated for approximately 185 years after closure of the mine.

20.3.10 Closure Cost Estimate

The final reclamation cost estimate for the FSU2 is as shown in Table 20-3.

The additional cost of closure/abandonment of the natural gas pipeline is estimated at $9.6 million. These amounts are portions of what is included to establish the Trust Fund.

Table 20-3: Estimated Reclamation Costs 2012–2264

Item	Unit	Total
Reclamation, Closure & Post-Closure		
BRCE Estimate	$M	132.8
Additional costs	$M	40.3
Post-Closure Monitoring	$M	16.1
Very Long Term Water Treatment	$M	482.3
Total	**$M**	**671.4**

20.4 Permitting

Donlin Gold will require a considerable number of permits and authorizations from both federal and state agencies. During the permitting process, the agencies will review the proposed Project, evaluate impacts associated with each facet of the Project, consider alternatives, identify compliance conditions, and ultimately decide whether or not to issue the requested permits. Much of the groundwork to support a successful permitting process is done before the permit applications are submitted, 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 80 permits this Project is likely to require, extensive baseline environmental information, supporting scientific analysis, and detailed engineering have been compiled. The Project has invested significant resources and time in acquiring this information over more than 15 years. Acquisition of these baseline data in parallel with design, and in advance of filing permit applications, has resulted in a Project that is designed to mitigate impacts on the environment where practicable, affording due consideration to all environmental concerns.

Throughout the preceding seven years, Donlin Gold has conducted extensive communication with a wide variety of regulatory agencies, stakeholder representatives, and recognized technical experts to ensure that all required information is collected appropriately. Agency update meetings and regulatory reviews of reports and documents ensure that the baseline data will meet the requirements of the formal permitting process, while giving regulators the opportunity to become familiar with the Project. This open communication will assist in evaluating and addressing concerns and in minimizing potential changes during the permitting and NEPA processes.

The comprehensive permitting process for the Project can be divided into three phases, 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; includes the collection of environmental baseline data and consultation with stakeholders and regulators
- The NEPA process and formal permit applications – formal agency review and analysis of the Project, resulting in the issue or denial of permits.

20.4.1 Exploration Stage Permitting

From the start of exploration activities in 1995 through advanced exploration and feasibility engineering, numerous permits or authorizations have been issued by federal agencies, state agencies, and Native Corporations to support ongoing operations. These include:

- Approximately 34 permits issued or modified by USACE
- 3 permits/authorizations issued or reviewed by the EPA
- 6 land use or right-of-way permits from the Bureau of Land Management (BLM)
- 4 airstrip authorizations from the Federal Aviation Administration (FAA)
- 22 mineral exploration or temporary water use permits issued or modified by the Alaska Department of Natural Resources (ADNR)
- 8 permits certified, issued, or modified by the Alaska Department of Environmental Conservation (ADEC)
- 8 Fish Habitat (Title 16) Stream Crossing Permits issued or modified by the Alaska Department of Fish and Game (ADF&G)
- Multiple Land Entry Agreements with Calista Corporation.
- Multiple Land Entry Permits with The Kuskokwim Corporation (TKC).

20.4.2 Pre-Application Phase

The Project has conducted a comprehensive stakeholder interaction and consultation process. This process has been an important component of the pre-permit application phase of the Project and is crucial to successful completion of the permitting and NEPA processes.

An equally important component of this phase has been ongoing interaction with permitting agencies and individual regulators who will be responsible for reviewing the

Project's permit applications. This process began in 1995 when Placer Dome, then the Project operator, first met with State of Alaska regulators and industry personnel to develop an understanding of the regulatory process in the state.

The concept of the state's Large Mine Permitting Team (LMPT) was developed in the process of permitting the Fort Knox gold mine near Fairbanks, and this model has been used successfully in the permitting of all major mining projects in the state since that time. Under this concept a company with a major project can enter into a reimbursable services agreement (RSA) with ADNR (and separately with ADEC) to compensate the state for expenses incurred in permitting and consultation activities related to the project. ADNR is responsible for administering the RSA, and the large mine coordinator assigned to the project is responsible for coordinating all state regulatory input and for coordinating with federal regulatory personnel. This process is specifically structured to be implemented early in the project life so that agencies can provide input into project design and baseline data collection ahead of the permitting and NEPA process.

The LMPT coordinator works with the company to organize inter-agency project update meetings and identify required expertise in regulatory agencies. The coordinator also works with stakeholders to provide mining education and to address questions about the state regulatory process. Formal Project consultation with the LMPT began in 2003, and the first RSA was implemented in 2004. Since that time, state regulators have been continuously involved in reviewing Project data collection, participating in Project update meetings, providing input to Project design decisions, and participating in stakeholder meetings related to the Project throughout the Kuskokwim region.

In addition to the LMPT interaction, Project personnel have regularly consulted with federal agency personnel, principally with the EPA, USACE, USFWS, and BLM. Meetings have also been held with the National Marine Fisheries Service, (NMFS) and the U.S. Coast Guard (USCG). The USACE and EPA have each designated a Manager to the Donlin Project to oversee the regulatory process in advance of the commencement of NEPA.

Table 20-3 includes some of the pre-application regulatory meetings in which Project personnel have participated in recent years.

Table 20-4: Key Pre-Application Regulatory Meetings

Agency	Date of Meeting
All-agency Project update to state and federal agencies	April 2009
Alaska LMPT Donlin Creek Project tour	August 2008
Alaska LMPT	August 2008; November 2010
LMPT update to all state and federal regulatory agencies	April 2004; May 2007
ADEC	August 2004; July 2005; September 2005; December 2005; March 2011; July 2011
ADEC Air Quality	June 2009; June 2010
ADF&G Habitat	June 2004; May 2005; July 2005; March 2011
ADNR SHPO	June 2004; February 2010; May 2010; March 2011
ADNR SPCO	December 2009; February 2010; June 2010; April 2011
BLM	May 2005; July 2005; December 2006; December 2009; March 2010; February 2011
BLM Field Office	January 2010; April 2011
EPA	June 2005; December 2005; February 2007; September 2008; March 2011; July 2011
EPA Donlin tour	September 2005; June 2009
NMFS	May 2009
USACE	May 2004; August 2004; May 2009; January 2011
USACE Donlin Tour	June 2009
USFWS	October 2004; September 2005; October 2005; June 2006; April 2009

Note: SHPO = State Historical Preservation Office; SPCO = State Pipeline Coordinator's Office; NMFS = National Marine Fisheries Service

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

20.4.3 The NEPA Process and Permit Applications

The full participation of the federal and state agencies in Project updates and review of environmental baseline studies has set the stage for submission of the Project permit applications and has identified the processes required to fully address all facets of the Project in a reasonable timeline. Once the first federal permit application is filed, the NEPA process commences.

Permits issued by federal agencies constitute "federal actions." Any major federal action requires review under NEPA. All elements of a project and their cumulative impacts are considered and evaluated in a NEPA review. In addition, alternatives to the proposed action are evaluated and potential mitigation measures are identified. For the Donlin Gold Project, the NEPA process will require the preparation of an Environmental Impact Statement (EIS). Typically, the federal agency with the predominant federal permit is designated the lead for the NEPA process; for the Project, this will be the USACE. Donlin Gold holds a Memorandum of Understanding with the USACE that defines the NEPA process. A Section 404 permit application (wetlands dredge and fill) will likely be the major federal action that initiates the NEPA process for Donlin. USACE will then request the participation of cooperating agencies, which will likely include the EPA, USFWS, and BLM.

Upon completion of the NEPA process, a Record of Decision (ROD) will be prepared that supports issuance of the permit for the preferred alternative for the Project, describes the conditions of the decision to issue the permit, and explains the basis for the decision. The state permitting process typically is not finalized until the NEPA process is completed. Each federal and state permit will have compliance stipulations requiring review and possibly negotiation by the applicant and appropriate agency. The LMPT coordinates state regulatory reviews and timelines to coincide with the federal permitting process.

20.4.4 Laws, Regulations, and Permit Requirements

Table 20-4 lists the potential federal permits and authorizations that Donlin Gold will have to obtain for the Donlin Gold Project and Table 20-5 lists the potential state permits and authorizations.

20.5 Considerations of Social and Community Impacts

Donlin Gold is committed to corporate social responsibility, strong collaboration with communities, and leaving a positive, sustainable legacy in the Y–K region. This requires a well-founded understanding of the social and economic relationships between the mine and the surrounding communities. Donlin Gold is focusing on sustainable development to benefit local communities over the long term by providing opportunities for direct employment, local procurement, and community development projects. Associated with these examples are efforts to develop lasting capacities that will continue after mine closure. The following principles, which underpin Donlin Gold's approach to community engagement activities, have been actively applied since early exploration and will continue throughout the life cycle of the mine:

- Engage with communities in a respectful and culturally sensitive manner
- Develop long-term mutually beneficial relationships.

Table 20-5: Potential Federal Agency Permits and Authorizations

Federal Agency	Permit or Approval
Bureau of Land Management	Surface estate lease (facilities on BLM managed lands) Land use permit (borrow pit activities on BLM managed lands) Access right-of-way (BLM managed lands) Grant for right-of-way (natural gas pipeline on BLM managed lands) Temporary use permits (construction outside ROW) Mineral material sales contracts (material sites on BLM managed lands)
Environmental Protection Agency	PSD air quality permit review CWA Section 402 APDES permit review CWA Section 404 permit review Spill prevention, control, and countermeasure (SPCC) plan approvals (construction and operations) Facility emergency response plan approval Emergency planning and Community Right-To-Know Act (hazardous substances) Hazardous waste generator identification number Used oil generator notification and identification number
U.S. Army Corps of Engineers	Nationwide Permit 6 – survey activities (wetlands) CWA Section 404 permit (wetlands dredge and fill) River and Harbors Act (RHA) Section 10 permit (structures in navigable waters) Section 106 Historical And Cultural Resources Protection Act clearance RHA Section 9 (dams and dikes 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 [mooring blocks] requires a permit) Anchorage permit Application for cargo transfer operations Port operations manual approval Facility response plans Private aids to navigation authorization Tug and barge vessel inspections Notice to mariners
Bureau of Alcohol, Tobacco, and Firearms	License to transport explosives Permit and license for use of explosives
Federal Communications Commission	Radio license
Federal Aviation Administration	Notice of landing area proposal (existing airstrip) Notice of controlled firing area for blasting
Homeland Security	TSA inspection program at airport Chemical facility anti-terrorism standards
U.S. Department of Transportation	Hazardous materials registration
Mine Safety and Health Administration	Mine identification number Notification of legal identity Training and retraining of miners plan

Federal Agency	Permit or Approval
National Marine Fisheries Service	Marine Mammal Protection Act consultation Essential fish habitat (EFH) consultation Section 7 of the Endangered Species Act (ESA) consultation
U.S. Fish and Wildlife Service	Section 7 of the ESA consultation

Table 20-6: Potential State Agency Permits and Authorizations

State of Alaska Agency	Permit or Approval
Alaska Department of Natural Resources	
Division of Coastal and Ocean Management	Alaska Coastal Management Program Consistency Determination
Division of Mining, Land, and Water	Plan of Operations review Reclamation plan approval Mining license (required for tax revenue regardless of land tenure) Land use permits and leases (activities on state land) Right-of-ways, easements, material sales, etc. Right-of-way (natural gas pipeline) Certificate of approval to construct a dam Certificate of approval to operate a dam Water dam operation & maintenance manual approvals Temporary water use permits Water rights permit/certificate to appropriate water Tidelands permit Shoreland permit
Office of History and Archaeology/ State Historic Preservation Office	Section 106 Historical and Cultural Resources Protection Act clearance Archaeology collection permit Field archaeology permit
Alaska Department of Fish and Game	
Habitat Division	Title 16 fish habitat permits Fish passage permits (culverts and bridges) Special Use Area permit (pipeline construction within State Game Refuge)
Division of Forestry	Burning Permits
Alaska Department of Environmental Conservation	

State of Alaska Agency	Permit or Approval
Division of Water	Alaska Pollution Discharge Elimination Permit (APDES) – pit perimeter dewatering APDES – domestic wastewater disposal APDES – pit lake discharge Section 401 water quality certification (CWA 404 permit) Stormwater multi-sector general permit (msgp) – construction, operations, and closure (mine site) Stormwater pollution prevention plan review (mine site) Construction stormwater discharge permit (pipeline) Construction stormwater pollution prevention plan review (pipeline) Approval to construct and operate a public water supply system
Division of Environmental Health	Waste management permits (waste rock dumps and TSF) Solid waste permit (construction and demolition debris) Food establishment permit
Division of Air Quality	Air quality PSD construction permit Air quality Title V operating permit Air quality permit to open burn
Division of Spill Prevention and Response	Oil discharge prevention and contingency plan (ODPCP) approval – operation of vessels and oil barges on state waters ODPCP approval – oil terminal/storage facility capable of storing 10,000 barrels or more Above-ground storage tank program (>420,000 gallons)
Department of Public Safety Office of the State Fire Marshal	Approval to Transport Hazardous Materials Life and Fire Safety Plan Check Plan Review Certificate of Approval for Each Building
Alaska Department of Labour and Workforce Development	
Division of Labor Standards and Safety	Certificates of Inspection for Fired and Unfired Pressure Vessels Occupational Safety and Health (inspections and certificates) Employer Identification Number
Alaska Department of Commerce, Community, and Economic Development	
Regulatory Commission of Alaska	Certificate of Public Convenience and Necessity for Natural Gas Pipeline

- Be responsive to stakeholders' concerns and questions.
- Build trust and confidence through accountability and transparency
- Understand the complex interests among diverse communities
- Adapt Project activities to fit with local needs and contexts
- Plan activities with closure in mind
- Monitor results and impacts.

Donlin Gold has a community relations team dedicated to understanding the concerns and issues facing the local communities. The Project's approach is to build trust and mutually beneficial relationships to help guide the development of mitigation plans and to manage risks responsibly. This engagement ensures that the potential impact of mining is adequately addressed while fostering community empowerment and self-sufficiency.

Donlin Gold has clearly defined responsibilities and commitments that adhere to Barrick policies, standards, and management systems.

Donlin Gold's stakeholder groups vary at the local, regional, and national level. Stakeholder mapping has been undertaken to identify stakeholders at each level and what the key issues are for each group. Stakeholder mapping forms the foundation for community engagement programs.

Stakeholder identification and issues mapping will continue throughout the permitting, construction, operation, and closure phases of the Project, and results will be updated annually. Cultural awareness is one of the many keys to identifying all relevant stakeholders, including possible vulnerable and minority groups.

Donlin Gold recognizes and respects that Alaska Native groups have strong attachments to their traditional lands and livelihoods. Donlin Gold's engagement with these communities is based on honest, open dialogue and providing information in a fair, timely manner that is culturally appropriate.

Donlin Gold promotes economic self-reliance among these Native communities through employment opportunities, business enterprise support, economic diversification, and where possible, preferential contract consideration for Native-owned suppliers. As the Project progresses, Donlin Gold will continue to focus on developing programs that benefit local communities, including improved infrastructure, support for education and health services, cultural heritage preservation, employment and business opportunities, increased income flows through royalty streams and compensation payments, and environmental restoration and protection.

Donlin Gold adheres to the principles and commitments in the *International Council on Mining & Metals Position Statement on Mining and Indigenous Peoples*. The statement promotes constructive relationships between the mining and metals industry and indigenous peoples based on respect, meaningful engagement, and mutual benefit. Barrick will finalize an Indigenous Peoples Policy in 2011 that will apply to all its sites and operations, including the Donlin Gold Project.

The NEPA process and the EIS) will assess the social and environmental impacts of the proposed Project. To inform that process, a Social Baseline Report will be completed in 2011 that provides sufficient information to support assessment of social impacts during NEPA. A comprehensive social impacts assessment includes a solid baseline; stakeholder and community engagement; analysis of direct, indirect, and cumulative impacts; and development of mitigation and monitoring plans. In addition, as the Project moves forward, Donlin Gold plans to produce a Community Engagement and Sustainable Development Plan that will comprehensively address all aspects of stakeholder participation in the Project, from permitting through operations to post-closure.

20.5.1 Stakeholders

The region has a complex political and social structure, represented by a diverse group of social, business, and governmental entities. Relationships between these various entities are often complex and are influenced by competing political and economic interests.

Entities within the region can be split into two primary categories: non-profit organizations (tribal/cultural/social) and for-profit corporations. Tribal organizations will play an important role in the NEPA process, as the NEPA process includes a government-to-government consultation requirement for the federal agency leading the permitting. Calista Corporation and The Kuskokwim Corporation, the two primary Native business entities of the region, each have a financial interest in the Project. A variety of other Native business entities and associations are also interested in the Project and its potential impact on the region.

As the Project scope has increased to recently incorporate the natural gas pipeline component, the area of influence has increased and with it, a wider group of stakeholders. A Project-wide stakeholder database is currently being maintained to help manage Project communications and consultation efforts.

Calista Corporation

Calista is one of the 13 regional Alaska Native corporations established as part of the *Alaska Native Claims Settlement Act* (ANCSA) of 1971. As part of this settlement, Native residents of much of Southwest Alaska became shareholders of Calista, a "for-profit" corporation that was given title to large areas of subsurface estate in the region. Calista has approximately 17,000 shareholders. The Donlin Gold Project is located on Calista mineral lands, and the Project operates under a mining lease with the

Corporation. Calista is a partner in the Project, having contractual rights and obligations related to Project development.

The Kuskokwim Corporation

At the same time that ANCSA established the regional corporations, it established local "village" corporations to take title to the surface estate of lands granted to the regional corporations. TKC is a "for-profit" corporation formed from the amalgamation of the 10 village corporations on the middle Kuskokwim River region between Lower Kalskag and Sleetmute, plus the village of Stony River. The Project operates under a surface use agreement with TKC. Authorization for use of TKC lands includes provisions that ensure the protection of subsistence activities.

Village Corporations

Individual for-profit village corporations exist for each village not included in the TKC consortium and were formed to serve the same function as TKC does for the villages closest to the Project. Many of these entities are interested in pursuing a business relationship with the Project.

Local Government and Non-profit Native Business Entities and Associations

The following non-profit entities also represent region stakeholders:

- Kuskokwim Native Association (KNA) – A non-profit association formed to provide social and other services to the villages of the TKC region. KNA represents the interests of people in the immediate area surrounding the Donlin Gold Project.
- Association of Village Council Presidents (AVCP) – A non-profit association formed to provide social and other services to residents of the Calista region.
- Village Tribal Councils – Each village within the region has a tribal council, sometimes referred to as a traditional council, to represent tribal interests and provide services to residents of the village. There are a total of 56 villages in the Calista region, many along the Yukon River.
- Crooked Creek Tribal Council – The village of Crooked Creek is closest to the Project and has a long history of involvement in pProject activities. Many of the Project's earliest and longest-term employees are residents of Crooked Creek. As representatives of the closest village, the Crooked Creek Tribal Council will be an important participant in the NEPA process.

- City councils – Larger towns within the region, such as Bethel and Aniak, have a formally-established government and an elected council that is separate from the tribal or “traditional” councils.
- Various boards and councils – These include entities such as subsistence advisory boards, watershed councils, and similar organizations, many of which receive government funding to promote their activities and which may participate during the permitting process.

To successfully acquire the support and “social license” required to develop and operate this Project, a process of ongoing engagement and consultation will be continued with all of these entities throughout the permitting process, construction, operation, and closure of the Project.

20.5.2 Community Development and Sustainability

Donlin Gold has defined three major components to its sustainable community development process.

Local Workforce Development and Training

A workforce training and development plan will be produced for the Project in advance of any construction or operating deadlines to allow enough time to implement the required training programs. The Operational Readiness Plan (Appendix J4) describes the mechanics of this process.

In addition to providing the required training, the company will need to implement a work schedule and environment that recognizes the unique nature of the Alaskan Native culture in the region. Issues that will need to be addressed, not all of which are unique to this region, include:

- The importance of subsistence harvesting activities
- Cross-cultural training for all employees and supervisors
- The importance of family relationships and participation in family events
- The historically seasonal nature of employment in the region
- Support for employees with family and dependency issues.

The success of the local hire program initiated during the exploration phase clearly demonstrates that development of a skilled local workforce is an achievable goal. Some of the Project employees have been working on the exploration program for more than 10 years and may be available to form the nucleus of future workforce development and training efforts.

Local Procurement

The Calista agreement grants a contracting preference to Calista with respect to procurement of services provided for the Project, subject to suitable qualifications. The success of Chiulista Camp Services throughout the exploration phase clearly demonstrates the Project's commitment to this process. In addition, Donlin Gold's policies clearly address the need to offer business and procurement opportunities for other entities and individuals within the region. Such opportunities have produced some of the most visible and direct benefits to local communities, such as the success of local air charter providers.

Community Development Activities

The Project has initiated informal discussions with village leaders in Crooked Creek to begin identifying potential community development projects that could benefit the village, its residents, and future employees. Other nearby villages that may warrant consideration for some level of community development projects include Kalskag, Aniak, Chuathbaluk, Napaimute, Georgetown, Red Devil, and Sleetmute.

Barrick has participated in a wide variety of community development projects around its operations, in fields such as infrastructure development, housing, health, and education.

20.6 Comments on Section 20

In the opinion of the QPs, the following conclusions are appropriate:

- There has been a focused effort for at least 15 years to collect comprehensive environmental baseline data and lay the groundwork with local and regulatory stakeholders for the successful permitting and development of a large-scale mining operation at the Donlin Gold Project
- Development and operation of the Donlin Gold Project will require a considerable number of permits and authorizations from both Federal and State agencies. Donlin Gold has identified the key State and Federal permits that will be required for construction of a mine under the assumptions in the FSU2. Much of the groundwork to support a successful permitting effort is done before permit applications are submitted, 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

- 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 a NEPA review and move forward with permit applications for construction, operations, and closure.
- Permitting 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
- Upon completion of the NEPA process, a ROD will be prepared that approves the preferred alternative for the Project, describes the conditions of the approval, and explains the basis for the decision.
- The State permitting process typically is not finalized until the NEPA process is completed.
- Each Federal and State permit will have compliance stipulations requiring review and possibly negotiation by the applicant and appropriate agency.
- Upon final issuance of permits and authorizations, the Environmental Management System (EMS) for the Donlin Gold Project will be fully implemented
- The comprehensive permitting process will determine the exact number of management plans required to address all aspects of the Project to ensure compliance with environmental design and permit criteria
- A Prevention of Significant Deterioration (PSD) air quality permit to construct will be required to construct and initially operate the mine, mill, and power plant facilities. Due to the proposed plans to site a large power generation facility adjacent to mine and mill facilities, estimated emissions are significant enough to trigger permitting under the PSD program. PSD requires modelling of potential air quality impacts and demonstrating that the applicant has necessary control of the land tenure to prevent public access to the area where air quality impacts occur. A combination of significant estimated emissions, combined with very stringent new air quality standards, including NO_2, could create some scenarios where a permit may be challenging to obtain
- The prevention of unregulated discharge of water that does not meet water quality standards is a primary criterion for overall Project design. This risk will be managed by continuing to optimize the use of process and contact water during operations, and ensuring that the capacity of the water treatment plant is sufficiently flexible to handle excess water during increased precipitation scenarios

- A modified version of the Barrick Reclamation Cost Estimator (BRCE) was used to develop reclamation and closure cost estimates. The final reclamation cost estimate is $132.8 million. This amount is included in a Trust Fund for Reclamation, Closure costs and Post-Closure Obligations model prepared to determine the funding required to generate sufficient cash flow to cover the following costs: spillway construction from Anaconda creek to Crevice Creek; capital to construct the WTP; perpetual water treatment; long-term monitoring; and associated facility and access maintenance. The total amount to cover reclamation and closure costs and post-reclamation and closure maintenance is estimated at $671.4 M; the primary component of which is the allocation of $482.3 M in long-term water treatment costs.
- Donlin Gold is focusing on sustainable development to benefit local communities over the long term by providing opportunities for direct employment, local procurement, and community development projects.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

21.1.1 Basis of Estimate

AMEC was responsible for preparing the estimate for all areas except mining (Barrick), the autoclave, autoclave support facilities, and oxygen plant (Hatch), the natural gas pipeline (CH2MHill), and the indirect costs associated with those areas. Closure and reclamation costs were prepared by SRK Consulting. AMEC has not checked for accuracy or verified costs and quantities for other parties' cost estimates. Warehouse inventory is excluded from the capital cost estimate but is included in Donlin Gold's financial model.

Where design was not of sufficient detail to prepare material take-offs, the estimate was based on factors or allowances. The estimate was prepared by area, using a Project work breakdown structure (WBS). The capital cost estimate was organized in accordance with the WBS defined for both direct and indirect areas. The estimate was also classified by discipline codes

Engineering material take-offs (MTOs) were based on "neat" quantities derived from Project drawings and sketches. Normal and accepted allowances were included in the estimate, as appropriate by discipline. Conceptual quantities were prepared where drawing information was not available. Metric units were assumed throughout the estimate, with the exception of pipe sizing, which was described in inches of nominal diameter.

The engineering, procurement, and construction management (EPCM) cost estimate was based on the FSU2 execution plan and labour estimates prepared by the Project leads based on deliverables and staffing durations. The EPCM estimate for the autoclave and oxygen plant facilities was prepared by Hatch, based on its experience. EPCM costs for both initial and sustaining capital for the tailings storage facility, contact water dams, and waste rock facility were prepared by BGC Engineering. EPCM costs for the off-site Jungjuk and Crooked Creek roads were prepared by RECON. The natural gas pipeline EPCM costs were provided by CH2MHill. The mine engineering and process engineering, including preproduction and commissioning costs, were prepared by Donlin Gold and are included in the Owner's cost.

Owner's costs were developed by Donlin Gold. Owner's costs are included in the capital cost estimate indirects and are to be spent over a five-year period prior to commencement of commercial operations.

Wage rates for construction crews were established based on recent AMEC in-house data and/or recent published rates for similar industrial construction and basic labour rate quotes from a contractor.

Construction camp sizing was based on camp loading and the construction schedule. The estimate includes a 2,560-person camp at site, one 49-person camp at Crooked Creek, which will be moved to Jungjuk with one unit at the airstrip for the access road contractor, a 49-person camp at Getmuna Flats, and an allowance to expand the existing camp at Donlin to 200. An average allowance has been made for catering costs based on vendor quotations.

Start-up and commissioning costs include allowances for the cost of commissioning assistance by the contractor for a period of three months.

Freight costs were estimated from information provided by vendor quotes to staging areas in either Seattle or Vancouver. Where information was not provided, freight costs to the staging area were estimated based on previous experience. Freight costs from marshalling areas for ocean and river barging were based on estimated loads and weights together with estimated freight rates.

Capital spares for the AMEC-priced processing areas were based on 5% of mechanical, mobile, and electrical equipment purchase value, and on 10% of mining equipment value.

The cost of capital spares for two of the areas for which Hatch did the estimating was estimated using vendor-recommended spare parts quotes, where provided. Barrick provided an estimate of spares for mining based on vendor-recommended spare parts quotes and the balance as a percentage of total direct costs, based on historical data from similar projects.

Warehouse inventory is excluded from the capital cost estimate but is included in Donlin Gold's financial model. The first fill requirement is included in the estimate.

There are no allowances for taxes, duties, or escalation in the capital cost estimate.

21.1.2 Contingency

Contingency was calculated using a risk analysis program (@RISK) to generate a range of probable costs. Donlin Gold selected the contingency value for the 85% probability level, which represents a 17.5% contingency on the AMEC and Donlin Gold scopes of work. Contingency (at 13.9% confidence) for the autoclave and oxygen plant scope of work was prepared by Hatch and was added to this contingency (Table 21-1).

21.1.3 First Fill

First fill is calculated as approximately 21 days of four months' supply. First fills are estimated as $20.1 million, 15% of consumables inventory, or approximately three weeks' supply. The consumables inventory estimate was developed by Donlin Gold in conjunction with AMEC based on the following assumptions:

- First production – 1 July 2019
- Mill consumables – $7.9 million
- Fuel requirement – $12.2 million.

Of the total consumables inventory, 85% is included in the financial model. Mine consumables, including fuel requirements, were included in preproduction operating costs.

21.1.4 Sustaining Capital

Sustaining capital requirements total $1,504 million and include the following considerations:

- Mining sustaining capital includes replacement of mining equipment based on hours of equipment use. Also included are the costs of purchasing additional units of equipment to match the mining plan.
- Mining dewatering sustaining capital includes provisions for additional well drilling, pumps, and pipelines to match the requirements of the mine pit.
- Tailings storage facility sustaining capital comprises the associated costs for overburden excavation, peat clearing, liner bedding, HPDE liner, filter zone material, and rockfill required to increase the capacity of the tailings storage facility.
- American Creek waste rock facility sustaining capital covers the costs of additional diversion dams.
- Airstrip sustaining capital includes an allowance for replacing regular maintenance gear.
- Plant mobile equipment sustaining capital covers scheduled replacement of equipment based on age.

Table 21-1: Capital Cost Contingency

Contingency Area	Total Project Cost Including Contingency ($M)	Contingency ($M)	Contingency (%)
AMEC, Donlin Gold	5,025,679	749,802	17.9
Hatch	680,321	94,964	13.9
CH2MHill	973,000	139,221	16.7
Total	*6,679,000*	*983,987*	*17.2*

- Administration building sustaining capital is an allowance for replacement of office equipment and furniture
- Truckshop sustaining capital allows for an expansion.
- Accommodation complex sustaining capital costs allows for expansion.

21.1.5 Capital Cost Summary

The total estimated cost to design and build the Donlin 59,000 stpd (53,500 t/d) Project described in this Report is $6,679 million, including an Owner-provided mining fleet and Owner-performed pre-development. The feasibility study update (FSU2) 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% / +30% of estimated final costs, per AACE Class 3 definition.

Table 21-2 summarizes the capital cost estimate by discipline. AMEC notes the following in relation to the proposed natural gas pipeline. The direct costs of the pipeline are estimated at $758.1 M, with indirect costs of an additional $75.7 M ($38.7 M engineering procurement, $32. 5M construction costs and, $4.4 M Owners' costs, primarily for land), totalling $829.4 M, excluding contingency. When contingency is included, the pipeline costs are estimated to total $973 M.

Table 21-3 shows the costs broken out by major area.

Table 21-2: Summary of Capital Costs by Discipline

Discipline	Cost ($000)
Direct Costs	
Civil	394,531
O/L Piping	123,717
Natural Gas Pipeline	758,100
Mining	266,142
Concrete	201,150
Structural	170,927
Architectural	186,735
Mechanical	1,375,700
Piping	211,556
Electrical	203,117
Instrumentation	97,155
Insulation and Coatings	20,445
Estimated Total Direct Costs	4,009,275
Indirect Costs	
Owner's Costs	414,414
Project Indirects	1,271,325
Estimated Total Indirect Costs	1,685,739
Subtotal	5,695,014
Contingency @ P_{85}	983,986
Estimated Total Project Cost	*6,679,000*

Table 21-3: Summary of Capital Costs by Major Area

Major Area	Cost ($000)
1000 – Mining	344,776
2000 – Site Preparation and Roads	235,487
3000 – Process Facilities	1,326,001
4000 – Tailings Management and Reclaim Systems	119,704
5000 – Utilities	1,302,030
6000 – Ancillary Buildings and Facilities	304,418
7000 – Off Site Facilities	243,328
8000 – Owner's Costs	414,414
9000 – Indirects	1,404,856
9900 – Contingency @P85	983,986
Total Project Cost	*6,679,000*

21.2 Operating Cost Estimates

21.2.1 Basis of Estimate

Salary and wage rates were based on a labour survey conducted for Donlin by an Alaska-based consulting company in 2010. The survey was benchmarked by Barrick and adjusted as needed. This report was prepared using a classification system similar to those currently in use at Barrick operations.

The work schedules assume that production will operate 24 h/d, 7 d/wk, 365 d/a. All hourly (non-exempt) personnel assigned to positions that form part of continuous operations will work a two-weeks-in/one-week-out rotation. These operations and mining personnel will work on two 12 h/d shifts.

Hourly personnel assigned to positions required only during the 12-hour day shift will work a two-weeks-in/two-weeks-out rotation. Hourly personnel working at the Bethel port site will work on three 8 h/d shifts: seven-days-on/two-days-off, seven-days-on/two-days-off, seven-days-on/three-days-off. All salaried (exempt) personnel and a few hourly personnel will work a 12 h/d shift on an eight-days-in/six-days-out rotation.

An estimated West Texas Intermediate (WTI) fuel guidance price of $85.00 per barrel was used, as provided by Donlin Gold. Based on this calculation, the price of diesel for the Project was set at $3.03/USgal ($0.80/L), including delivery costs but excluding any taxes.

Gas fuel price was provided by Barrick Energy and is based on the WTI guidance of $85.00 per barrel of oil. The pricing assumes the import of liquefied natural gas (LNG) to Anchorage; total delivery costs associated with purchase, delivery, transportation, and regasification of the LNG; delivery through the Cook Inlet pipeline network; and operating costs for the Cook Inlet-to-Donlin pipeline. The delivered cost of gas is $13.33/MBtu.

The estimated cost of electricity from gas-fired CCGT generation is $0.11978/kWh, not including capital costs.

The total unit operating cost of the general cargo supply chain was estimated to be equivalent to $264.89 per cargo tonne. The unit operating cost for delivery of fuel to the mine site was estimated to be $0.544/USgal.

21.2.2 Mine Operating Costs

The mine operating cost estimate incorporates costs for operating and maintenance labour, staff, and supplies for each year. Operating and maintenance supplies are based on North American supply and include an allowance for freight and delivery to marshalling yards at the ports of Vancouver or Seattle, as appropriate, and then to site. Taxes are not included. Consumables (fuel, explosives, supplies) were calculated from expected use, unit consumptions, and allowances for minor items. All mining costs are based on production Years 1 to 25 (2019 to 2043). Preproduction costs have been capitalized and included in the capital cost estimate.

The expenditure breakdown percentages by area are shown in Figure 21-1.

21.2.3 Process Operating Costs

Processing costs cover operation and maintenance of the processing facilities, from the coarse ore dump pocket at the primary crusher through to the bullion room, as well as process and reclaim water pumping. The processing costs account for the expenses associated with purchasing consumables, equipment maintenance, personnel, and power consumption. Costs are summarized in Table 21-4.

Process consumables consist of comminution media, reagents, office supplies, and waste/hazardous waste disposal. Freight-associated charges were based on a rate of $264.89/t for cargo transport from Vancouver and Seattle. An additional $20/t was added for the grinding media freight; grinding media needs to be shipped in special gondola barges, which resulted in a higher cost for the FSU2 study.

Equipment maintenance (including liners) was developed for all areas from first principles for major equipment and from relative factors for minor equipment, and was then expressed as a percentage of the capital cost of equipment for all areas except pressure oxidation and the oxygen plant. Annual estimated maintenance costs for these two areas were developed from first principles by Hatch. The maintenance costs are also divided by area according to Barrick's asset naming convention.

Figure 21-1: LOM Mine Production Operating Costs

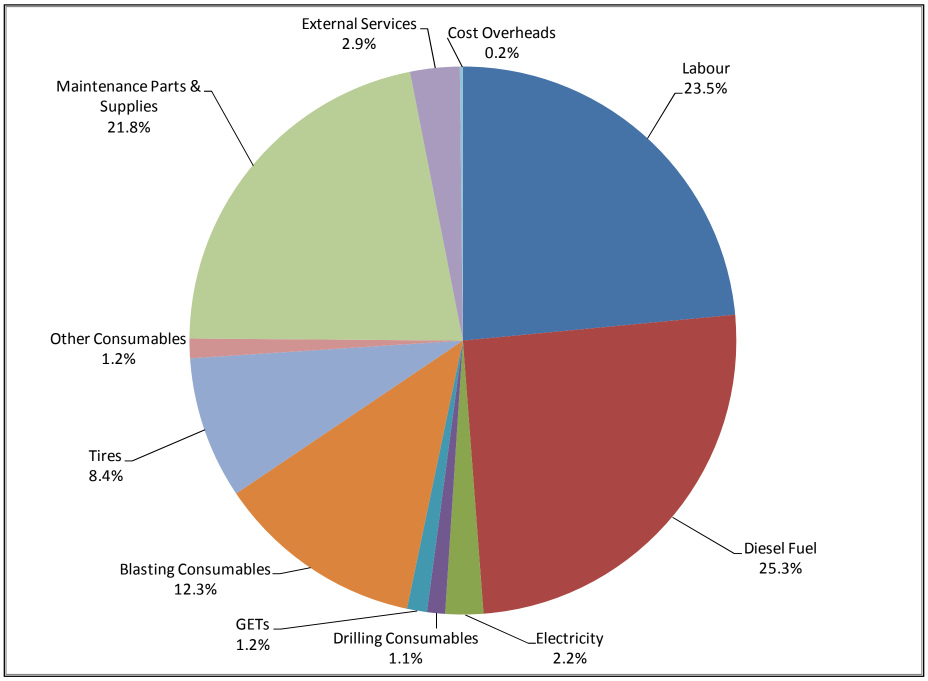


Table 21-4: LOM Direct Process Operating Costs ($000)

Item	Total	$/t Milled
Labour	615,825	1.22
Reagents and Consumables	2,586,751	5.12
Power	3,496,941	6.93
Maintenance Supplies	859,904	1.70
Subtotal	7,559,401	14.97
G&A/Rehandling Reallocations	248,172	0.49
Total	*7,807,593*	*15.47*

Power consumption was derived from the estimated load of individual pieces of equipment on the equipment list combined with power requirements for the crushing and grinding circuits. Power consumption was estimated at an average of 1,049,496 MWh/a. The cost of power was provided at $0.119/kWh. The calculation of primary crusher power consumption was based on the crusher running at 80% of full power, available 65% of the time.

21.2.4 General and Administrative Operating Costs

The general and administrative (G&A) operating costs are expenses for cost centres not directly linked to the mining and process disciplines, and include management, safety, security, environmental, information services, warehouse and other overheads. The G&A for each cost centre was estimated either from first principles or was based on experience at Barrick operations.

Some costs included in G&A have been 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.

21.2.5 Operating Cost Summary

Operating costs over the life of mine are indicated in Table 21-6. A breakdown by operating year is included as Table 21-7. Operating costs were prepared in second quarter (Q2) 2011 U.S. dollars with no allowances for escalation, sales tax, import duties, or contingency.

Life-of-mine operating costs, excluding community and social development costs and refining charges, were estimated at $5.85/t mined, or $37.79/t milled.

Table 21-5: Summary of G&A Cost Estimate by Cost Centre ($000)

	LOM Annual Average				Allocations		Net	
Cost Centre	**Personnel**	**Expenses**	**Total**	**$/t milled**	**Total**	**$/t milled**	**Total**	**$/t milled**
Logistics	10,450	4,381	14,831	0.78	10,719	0.57	4,112	0.21
Camp & Catering	–	12,085	12,085	0.63	–	–	12,085	0.63
Finance & Administration	7,672	14,821	22.493	1.17	–	–	22,493	1.17
Insurance	–	6,792	6,792	0.36	–	–	6,792	0.36
Site Maintenance & Mobile Equipment	3,175	2,384	5,559	0.29	–	–	5,559	0.29
Aviation	164	3,669	3,833	0.20	–	–	3,833	0.20
Power	-	4,245	4,245	0.22	–	–	4,245	0.22
Environmental	2,337	284	2,621	0.14	–	–	2,621	0.14
Subtotal	23,798	48,661	72,459	3.79				
Allocated to Process & Mine					10,719	0.57		
Total							*61,740*	*3.22*

Table 21-6: LOM Operating Costs ($000)

Area	Total LOM	$/t Milled	$/t Mined	$/oz
Mine Operations	8,200,480	16.24	2.52	270
Processing Operations	7,807,593	15.47	2.40	257
Administration	1,626,247	3.22	0.49	53
Refining	31,069	0.06	0.01	1
Total	*17,665,389*	*34.99*	*5.42*	*581*

Note: operating costs per tonne milled in this table do not include $3.13/t milled of land costs incorporated in the financial analysis and cashflow model as additional general and administrative costs. Life-of-mine operating costs, excluding community and social development costs and refining charges, were estimated at $5.85/t mined, or $37.79/t milled

Table 21-7: Annual Operating Costs ($000)

Year	Mining	Processing	Admin	Refining	Total	$/t Milled	$/t Mined	$/oz
1	200,288	128,910	40,765	534	370,496	49.15	3.61	710
2	250,201	310,835	64,228	1,424	626,688	33.52	5.12	450
3	275,714	304,623	61,451	1,417	643,205	33.49	5.16	464
4	301,291	299,252	61,434	1,524	663,501	34.18	4.95	445
5	322,753	302,622	61,066	1,566	688,008	35.18	4.83	449
6	346,057	302,801	60,599	1,544	711,001	36.40	4.64	471
7	345,843	295,531	61,256	1,609	704,239	37.62	4.54	447
8	367,020	293,290	62,204	1,323	723,837	38.87	4.79	559
9	392,133	298,901	61,852	1,398	754,284	38.52	5.03	552
10	357,425	306,258	60,580	1,545	725,808	36.91	4.84	480
11	347,772	304,932	60,916	1,269	714,889	36.75	4.77	576
12	365,053	299,003	61,887	1,497	727,440	37.65	4.85	497
13	372,109	300,566	62,159	1,274	736,108	37.06	4.91	590
14	387,990	289,983	62,112	970	741,056	39.43	4.94	781
15	406,258	286,235	62,458	907	755,859	41.09	5.04	851
16	424,279	286,313	62,798	1,012	774,402	40.63	5.16	782
17	412,287	292,383	62,013	801	767,484	39.48	5.12	979
18	406,205	296,269	61,833	943	765,250	39.25	5.75	829
19	410,167	294,723	61,877	1,148	767,914	39.30	6.09	684
20	418,548	299,773	61,434	1,446	781,202	39.80	5.21	552
21	390,864	295,429	60,961	923	748,177	40.19	4.99	829
22	248,198	291,067	58,453	1,069	598,787	32.74	5.01	573
23	179,550	296,981	57,307	1,061	534,899	27.86	7.20	515
24	165,332	294,946	57,921	882	519,081	27.05	9.11	601
25	72,498	288,375	56,788	587	418,247	21.46	32.21	729
26	19,390	289,795	57,176	656	367,017	18.77	N/A	572
27	15,254	257,798	62,721	740	336,513	19.85	N/A	465
Total	*8,200,480*	*7,807,593*	*1,626,247*	*31,069*	*17,665,389*	*34.99*	*5.42*	*581*

Note: operating costs per tonne milled in this table do not include $3.13/t milled of land costs incorporated in the financial analysis and cashflow model as additional general and administrative costs. Life-of-mine operating costs, excluding Community and Social Development costs and refining charges, were estimated at $5.85/t mined, or $37.79/t milled

21.3 Comments on Section 21

The QPs have reached the following conclusions regarding the operating and capital cost estimates prepared as part of the FSU2:

- The total estimated capital cost to design and build the Donlin Gold Project described in this Report is $6,679 million, including an Owner-provided mining fleet and self-performed pre-development. Included in the estimate are:
  - Direct capital costs: $4,009 million
  - Owner's costs: $414 million

  - Other indirect costs: 1,271 million
  - Contingency: $984 million.

- The 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% +30% of estimated final costs, per AACE Class 3 definition. All costs are expressed in second quarter (Q2) 2011 U.S. dollars. No allowances are included for escalation, interest during construction, taxes, or duties
- Sustaining capital costs are estimated at $1,504 million
- The estimated life-of-mine operating costs are $5.42/t mined or $34.99/t milled, or $581/oz
- The general and administrative operating costs in this section, of $3.28/t milled (total operating cost of $34.99/t milled), includes G&A expenditures, community and social development costs and refining charges, but excludes land-related costs and payments of $3.13/t milled. These land-related payments of $3.13/t milled (total operating cost of $38.13/t milled) are included in the financial model and cash flow results
- Life-of-mine operating costs, excluding community and social development costs and refining charges, were estimated at $5.85/t mined, or $37.79/t milled
- Operating cost estimates have been assembled by area and component, based on estimated staffing levels, consumables, and expenditures, according to the mine plan and process design. Operating costs have been prepared in second quarter (Q2) 2011 U.S. dollars with no allowances for escalation, sales tax, import duties, or contingency.

22.0 ECONOMIC 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. Forward-looking information includes Mineral Resource and Mineral Reserve estimates, commodity prices and exchange rates, the proposed mine production plan, projected recovery rates, uncertainties and risks regarding the estimated capital and operating costs, uncertainties and risks regarding the cost estimates and completion schedule for the proposed Project infrastructure, in particular the need to obtain permits and governmental approvals.

22.1 Valuation Methodology

The overall economic viability of the Donlin Gold Project has been assessed using both undiscounted and discounted cash flow techniques. Undiscounted techniques include total net cash flow, payback period (measured from start of production), EBITDA, and cash costs. Discounted cash flow techniques include IRR and NPV. Discounted values are calculated using a 5% discount rate and a discrete end-of-year convention relative to reference dates of 1 January 2012 and 1 January 2014. A period of approximately 3.5 years for permitting, starting 1 January 2012, is included prior to start of construction.

Estimates have been prepared for all the individual elements of cash receipts and cash expenditures for ongoing operations. Capital cost estimates have been prepared for initial development and construction of the Project and for ongoing operations (sustaining capital). Cost estimates have also been prepared for reclamation and closure of the mine and for post-closure obligations. These form the basis for the annual funding requirements over the LOM required to establish an endowment to meet these obligations.

The economic evaluation of the Donlin Gold Project was prepared by Donlin Gold and is based upon:

- Capital cost and sustaining capital cost estimates prepared by AMEC, BGC, and Hatch
- Owner's capital costs prepared by Donlin Gold
- Reclamation and closure costs prepared by SRK
- Post-closure obligations prepared by Donlin Gold

- Funding requirements for the reclamation, closure, and post-closure obligations endowment prepared by Donlin Gold
- Mine schedule prepared by Barrick
- Resource estimate prepared by Donlin Gold
- Mine equipment costs based on quotes received from equipment suppliers
- Estimated mine, process plant, and general and administration operating costs prepared by Donlin Gold, AMEC, Barrick, and Hatch, based on budget quotations, first principles, and/or costs at operating mines similar to that proposed at Donlin such as Barrick's Goldstrike operation
- An allowance for supply inventory and working capital (including doré transportation, in-process inventory, and payment delays); these values sum to zero over the life of the mine.

22.2 Financial Model Parameters

Parameters assumed for the financial model included the following areas.

22.2.1 Production Forecast

The FSU2 is based on a 59,000 st/d (53,500 t/d) open pit gold mine with ore processing by means of flotation, pressure oxidation, and cyanidation. The pit designs and production schedules were based on the in-situ gold mineral resource drilling at Donlin as of December 2009. Annual LOM gold production will average 1.13 Moz per year and 1.46 Moz for the first five full years of production.

22.2.2 Metallurgical Recoveries

Recovery is estimated to average 89.8% over the LOM based on work and testing performed for feasibility study purposes.

22.2.3 Smelting and Refining Terms

Doré refining and shipping charges were estimated at $1.02/oz based on actual refining charges for Barrick's Goldstrike operations and a quotation for transportation and insurance costs from the planned Donlin mine site to a U.S.-based refinery. In addition, 0.1% of gold produced at the mine is deducted as a cost of refining.

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 has been applied to the Project.

22.2.4 Metal Prices

Estimated cash flows from revenue are based on a gold price of $1,200/oz as provided by Donlin Gold. The open pit was optimized at a gold price of $975/oz, which was the guidance in effect at the time the pit optimization work was completed.

22.2.5 Capital Costs

The initial capital costs for the Project are estimated at $6,679 million. Sunk costs are excluded from the cash flow calculation. Tax pools are zero at the end of 2010. Prior year expenses have been allocated to, and used by, the partners.

22.2.6 Operating Costs

Life-of-mine operating costs were estimated at $5.85/t mined, or $37.79/t milled.

22.2.7 Royalties

The Calista Corporation and The Kuskokwim Corporation (TKC) receive payments for land access and use. Over the life of the mine these payments amount to $1,580 million.

22.2.8 Working Capital

Inventory of consumables plus working capital are included in the cash flow. They are expenditures in the early years and are recovered in the final years.

First Fills Inventory

- Included in capital costs.

Initial Inventory

- Initial Inventory = 100.0 Days of Non-Labour Operating Costs.

Working Capital

- Accounts Payable = 30.0 Days of Non-Labour Operating Costs
- In-Process Inventory = 3.0 Days of Revenue
- Finished Products Inventory = 10.0 Days of Revenue
- Accounts Receivable = 10.0 Days of Revenue.

22.2.9 Taxes

AMEC is not an expert in taxation matters. The following taxation summary was prepared by Donlin Gold for the Project.

- Federal Income Tax = 35% subject to a Alternative Minimum Tax of 20%
- Alaska State Income Tax = 9.4% subject to a Alternative Minimum Tax of 18%
- Alaska State Mining License Tax = 7% of taxable mining income. There is a 3.0-year tax holiday on the mining license tax.

Income tax becomes payable after deductions for capital allowances.

22.2.10 Closure Costs and Salvage Value

The basis for the closure estimate for the site was a modified version of the Barrick Reclamation Cost Estimator (BRCE). BRCE costs are primarily incurred in the first five years after the mine closes (2044–2047), although some reclamation will be carried out immediately after construction and during operations. Some closure activities and expenditures will require ongoing post-closure operations and maintenance, such as water treatment, maintenance of surface water management facilities, and post-closure monitoring.

Total closure and reclamation costs were estimated at $674.1 M.

It is assumed that construction equipment will be sold at the end of the construction period when it is no longer required for Project-based work. Total recovered value from these sales is estimated at $23 million. These represent the only assets at closure. No salvage is assumed at the end of operations.

22.2.11 Financing

Financing has been assumed on a 100%, all equity, stand-alone basis.

22.2.12 Inflation

Escalation/inflation has been excluded. Escalation has been included in the determination of the funding requirements for the Trust Fund, but the Trust Fund values in the cash flow are expressed on an un-escalated (real) basis.

22.3 Financial Results

The financial analysis in the FSU2 report included discounted cash flow (DCF) metrics based on reference dates of 1 January 2012 and 1 January 2014. The DCF metrics for 1 January 2014 treat funds expended before that date as sunk, and are used by NovaGold as the Base Case for this Report.

Financial analysis of the Base Case (discount rate of 5%) showed the after-tax Project NPV to be $547 M and the internal rate of return (IRR) to be 6% (Table 22-1). The cashflow for the Project on an annualized basis is included as Table 22-2.

The cumulative, undiscounted, after-tax cash flow value for the Project is $6,197 M and the after-tax payback period is 9.2 years.

22.4 Sensitivity Analysis

Sensitivity analyses have been performed on the Project on a range of -20% to +20% on gold price, operating costs, and capital costs. For purposes of the sensitivity analysis, variations in the gold grade were assumed to mirror variations in the gold price. Sensitivities are as indicated in Figure 22-1.

The Project is particularly sensitive to changes in the gold price. The Project requires a gold price of approximately $902/oz to break even on a cash flow basis and a gold price of approximately $1,141/oz to achieve an IRR of 5% (Table 22-3).

Table 22-4 and Table 22-5 list the sensitivities of after-tax NCF, NPV (5%), and IRR to variations in operating cost and capital cost, respectively.

Table 22-6 is an analysis of the impact of changes in oil prices on operating costs. It was determined that a 10% change in the price of oil translates into an approximate 1.28% change in total operating costs (mining + processing + G&A).

Table 22-7 shows the impact of variations in the liquefied natural gas (LNG) price, assuming that 96% of LNG is used to produce electricity. Therefore, a ±18% in LNG prices translates into a ±10.42% change in power costs.

Table 22-1: Summary of Key Evaluation Metrics (Base Case is highlighted)

Item	Unit	Value
Total Mined	Mt	3,260
Ore Tonnes Treated	Mt	505
Strip Ratio	W/O	5.46
Gold Recovered	Moz	30.401
Gold Recovery	%	89.8
Gold Price	$/oz	1,200
Total Operating Costs	$/oz	584
Total Costs Before Taxes	$/oz	908
Total Costs Including Taxes	$/oz	998
EBITDA	$M	18,581
Total Cash Flow*	$M	6,197
Jan 2012 NPV @ 5%**	$M	337
Jan 2012 IRR	%	5.6
Jan 2014 NPV @ 5%**	$M	547
Jan 2014 IRR	%	6.0
Payback Period	Years	9.2
Operation Life	Years	27.0
Initial Capital	$M	6,679
Total LOM Capital	$M	8,184

Note: EBITDA = Earnings before interest, taxes, depreciation, and amortization

* Cash flow excludes sunk costs

** Reference dates for DCF metrics are 1 January 2012 and 1 January 2014. The DCF metrics for 1 January 2014 treat funds expended before that date as sunk.

During 2012 and 2013, Donlin Gold intends to complete basic engineering and commence detailed engineering, in tandem with, and in the case of detailed engineering, subject to, progress achieved on the Environmental Impact Statement and associated permitting process. Aggregate expenditures in these years are expected to be approximately $172 million, which if excluded from the discounted cash flow analysis would result in an increased project NPV at 5% and IRR from 2014 onwards of $210 million and 0.4%, respectively.

Table 22-2: Cashflow Analysis

Cash Flow	Units	Total	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Ore treated	Mt	504.811	0.000	0.000	0.000	0.000	0.000	7.539	18.696	19.205	19.412	19.557	19.533	18.719	18.621	19.582	19.662	19.455	19.321
Payable gold	Moz	30.371	0.000	0.000	0.000	0.000	0.000	0.522	1.392	1.385	1.490	1.531	1.509	1.572	1.294	1.366	1.510	1.240	1.464
Gross revenue	$M	36,481.103	0.000	0.000	0.000	0.000	0.000	626.623	1,671.682	1,663.701	1,789.646	1,839.041	1,813.185	1,888.842	1,554.035	1,641.111	1,813.824	1,489.843	1,758.005
Operating costs	$M	(17,752.172)	(2.983)	(6.118)	(12.096)	(13.697)	(12.387)	(249.850)	(618.568)	(581.478)	(556.425)	(573.802)	(655.451)	(604.062)	(710.008)	(785.572)	(683.788)	(695.074)	(692.595)
Applied depreciation	$M	(9,845.995)	0.000	0.000	0.000	0.000	0.000	(120.537)	(338.016)	(334.130)	(354.290)	(374.343)	(372.830)	(386.654)	(328.078)	(356.971)	(388.992)	(325.599)	(384.061)
Community & social development	$M	(137.671)	0.000	0.000	0.000	0.000	0.000	(2.598)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)
Total costs	$M	(27,735.838)	(2.983)	(6.118)	(12.096)	(13.697)	(12.387)	(372.984)	(961.779)	(920.804)	(915.911)	(953.340)	(1,033.477)	(995.911)	(1,043.281)	(1,147.738)	(1,077.976)	(1,025.868)	(1,081.851)
Income before tax	$M	8,745.265	(2.983)	(6.118)	(12.096)	(13.697)	(12.387)	253.639	709.903	742.897	873.735	885.701	779.709	892.930	510.753	493.373	735.848	463.975	676.154
Alaska state income tax	$M	(701.398)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	(23.910)	(25.540)	(90.449)	(68.861)	(100.781)
Alaska mining tax	$M	(536.904)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	(33.976)	(37.168)	(56.807)	(36.664)	(37.294)	(54.439)	(35.256)	(52.258)
Federal income tax	$M	(1,503.066)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	(132.834)	(141.891)	(180.573)	(119.539)	(170.034)
Total taxes	$M	(2,741.367)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	(33.976)	(37.168)	(56.807)	(193.409)	(204.725)	(325.461)	(223.657)	(323.072)
Net income after tax	$M	6,003.898	(2.983)	(6.118)	(12.096)	(13.697)	(12.387)	253.639	709.903	742.897	873.735	851.725	742.540	836.124	317.345	288.648	410.387	240.318	353.082
Stockpile Inventory Adjustment - Opex	$M	(0.000)	0.000	0.000	0.000	0.000	0.000	(122.647)	(31.148)	(85.361)	(132.722)	(140.635)	(136.629)	(181.699)	(92.282)	(38.525)	(117.183)	(94.979)	(104.603)
Depreciation add-back	$M	9,845.995	0.000	0.000	0.000	0.000	0.000	120.537	338.016	334.130	354.290	374.343	372.830	386.654	328.078	356.971	388.992	325.599	384.061
Operating cash flow	$M	15,849.893	(2.983)	(6.118)	(12.096)	(13.697)	(12.387)	251.528	1,016.770	991.667	1,095.304	1,085.433	978.741	1,041.079	553.142	607.094	682.197	470.939	632.540
Initial capital	$M	(6,511.411)	(230.989)	(659.453)	(1,659.770)	(1,927.742)	(1,708.416)	(325.043)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Sustaining capital	$M	(1,504.389)	0.000	0.000	0.000	0.000	0.000	(336.686)	(68.594)	(33.947)	(10.586)	(154.437)	(57.864)	(28.556)	(75.103)	(156.916)	(12.208)	(27.517)	(18.359)
IFRS Total Capitalized Opex (Sustaining Capital)	$M	(1,386.313)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Funding of Closure "Trust Fund"	$M	(273.730)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)
Add: Salvage Values	$M	23.118	0.000	0.000	0.000	0.000	0.000	23.118	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Initial Inventory	$M	0.000	0.000	0.000	0.000	0.000	0.000	(140.489)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Working Capital	$M	0.000	0.000	0.000	0.000	0.000	0.000	(16.333)	(48.509)	1.582	(6.733)	(1.514)	3.100	(5.318)	22.280	(3.488)	(12.537)	19.714	(16.239)
Net cash flow	$M	6,197.167	(242.526)	(674.125)	(1,680.420)	(1,949.993)	(1,729.356)	(552.458)	891.114	950.748	1,069.431	920.928	915.423	998.650	491.765	438.135	648.897	454.581	589.388
Cumulative cash flow	$M		(242.526)	(916.651)	(2,597.071)	(4,547.064)	(6,276.420)	(6,828.879)	(5,937.765)	(4,987.017)	(3,917.586)	(2,996.658)	(2,081.234)	(1,082.584)	(590.819)	(152.684)	496.213	950.794	1,540.182

Table 22-2: Cashflow Analysis cont

Cash Flow	Units	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046
Ore treated	Mt	19.861	18.796	18.394	19.059	19.438	19.499	19.538	19.628	18.615	18.291	19.201	19.192	19.491	19.550	16.956	0.000
Payable gold	Moz	1.246	0.949	0.887	0.989	0.783	0.922	1.122	1.414	0.902	1.045	1.038	0.863	0.574	0.641	0.723	0.000
Gross revenue	$M	1,496.235	1,139.343	1,065.490	1,187.893	940.918	1,107.142	1,348.146	1,698.052	1,083.306	1,254.699	1,246.326	1,036.185	688.900	770.265	868.668	0.000
Operating costs	$M	(750.869)	(700.757)	(683.868)	(686.253)	(712.277)	(713.701)	(768.377)	(746.116)	(712.958)	(630.635)	(673.798)	(606.330)	(649.148)	(644.010)	(593.106)	(26.012)
Applied depreciation	$M	(350.810)	(284.267)	(286.724)	(343.102)	(290.167)	(365.919)	(456.179)	(602.635)	(406.833)	(479.645)	(502.897)	(413.521)	(295.530)	(332.838)	(370.426)	0.000
Community & social development	$M	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	(5.195)	0.000
Total costs	$M	(1,106.875)	(990.219)	(975.787)	(1,034.550)	(1,007.640)	(1,084.816)	(1,229.751)	(1,353.946)	(1,124.986)	(1,115.475)	(1,181.891)	(1,025.046)	(949.874)	(982.044)	(968.727)	(26.012)
Income before tax	$M	389.361	149.124	89.702	153.342	(66.722)	22.327	118.395	344.107	(41.681)	139.225	64.435	11.139	(260.974)	(211.779)	(100.060)	(26.012)
Alaska state income tax	$M	(62.761)	(29.505)	(20.706)	(27.710)	(3.743)	(14.640)	(29.940)	(70.679)	(17.182)	(38.725)	(36.401)	(26.972)	0.000	0.000	(12.894)	0.000
Alaska mining tax	$M	(31.715)	(14.568)	(9.663)	(12.357)	(1.907)	(6.271)	(13.499)	(35.309)	(9.714)	(18.545)	(17.500)	(13.099)	0.000	(2.153)	(6.741)	0.000
Federal income tax	$M	(112.068)	(55.235)	(42.943)	(61.753)	(6.567)	(34.550)	(65.628)	(129.243)	(22.984)	(76.784)	(71.670)	(52.116)	0.000	0.000	(26.653)	0.000
Total taxes	$M	(206.544)	(99.308)	(73.312)	(101.821)	(12.217)	(55.462)	(109.067)	(235.231)	(49.879)	(134.053)	(125.572)	(92.187)	0.000	(2.153)	(46.288)	0.000
Net income after tax	$M	182.816	49.816	16.391	51.522	(78.939)	(33.135)	9.328	108.876	(91.560)	5.171	(61.137)	(81.048)	(260.974)	(213.932)	(146.347)	(26.012)
Stockpile Inventory Adjustment - Opex	$M	(2.252)	42.240	50.982	88.612	53.728	96.838	61.180	50.997	(12.707)	26.784	132.447	36.361	191.867	247.055	214.280	0.000
Depreciation add-back	$M	350.810	284.267	286.724	343.102	290.167	365.919	456.179	602.635	406.833	479.645	502.897	413.521	295.530	332.838	370.426	0.000
Operating cash flow	$M	531.375	376.322	354.097	483.236	264.956	429.623	526.687	762.507	302.566	511.600	574.207	368.834	226.423	365.961	438.359	(26.012)
Initial capital	$M	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Sustaining capital	$M	(146.522)	(22.681)	(85.442)	(26.364)	(108.299)	(15.530)	(1.477)	(10.032)	(77.381)	(0.789)	(1.627)	(1.813)	(25.660)	0.000	0.000	0.000
IFRS Total Capitalized Opex (Sustaining Capital)	$M	(56.597)	(142.609)	(171.182)	(224.718)	(156.602)	(191.311)	(112.469)	(150.635)	(87.717)	(44.439)	(48.034)	0.000	0.000	0.000	0.000	0.000
Funding of Closure "Trust Fund"	$M	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	(8.554)	0.000
Add: Salvage Values	$M	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Initial Inventory	$M	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	140.489	0.000
Working Capital	$M	17.063	22.687	5.527	(6.566)	15.251	(10.567)	(15.053)	(21.148)	36.656	(20.164)	(3.370)	12.205	15.519	(8.355)	(8.770)	33.078
Net cash flow	$M	336.765	225.165	94.446	217.035	6.753	203.661	389.134	572.138	165.570	437.654	512.622	370.672	207.728	349.052	561.524	7.066
Cumulative cash flow	$M	1,876.947	2,102.112	2,196.558	2,413.593	2,420.346	2,624.007	3,013.141	3,585.279	3,750.849	4,188.503	4,701.125	5,071.797	5,279.525	5,628.577	6,190.101	6,197.167

Figure 22-1: After Tax LOM Total Cash Flow Sensitivity Spider Graph

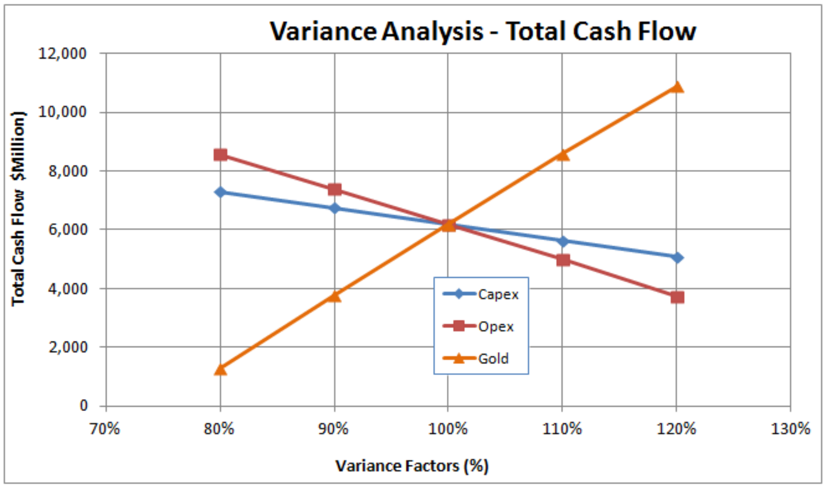


Table 22-3: Base Case Project Sensitivity to Gold Price (Base Case is highlighted)

Gold ($/oz)	LOM Cash Flow ($M)	Jan 2014 NPV @ 5% ($M)	Jan 2014 IRR (%)
700	(5,690)	(4,917)	—
800	(2,838)	(3,637)	—
900	(45)	(2,374)	—
1,000	2,143	(1,342)	2.3
1,100	4,191	(385)	4.3
1,200	6,197	547	6.0
1,300	8,187	1,465	7.5
1,400	10,166	2,375	8.9
1,500	11,631	3,147	10.2
1,600	13,092	3,862	11.2
1,700	14,616	4,581	12.3
1,800	16,156	5,296	13.2
1,900	17.699	6,010	14.2
2,000	19,248	6,722	15.1
2,100	20,793	7,429	15.9
2,200	22,343	8,138	16.8
2,300	23,882	8,838	17.6
2,400	25,429	9,541	18.3
2,500	26,975	10,243	19.1

Table 22-4: Project Sensitivity to Operating Cost (Base Case is highlighted)

Factor (%)	LOM Cash Cost ($/oz)	LOM Cash Flow ($M)	NPV @ 5% ($M)	IRR (%)	Opex ($/t ore)
80	478	8,565	1,627	7.8	25.66
90	532	7,387	1,089	6.9	28.86
100	585	6,197	547	6.0	32.05
110	638	4,985	(4)	5.0	35.24
120	691	3,726	(568)	3.9	38.43

Note: Opex = operating cost

Table 22-5: Project Sensitivity to Capital Cost (Base Case is highlighted)

Factor (%)	Total Capex ($M)	LOM Cash Flow ($M)	NPV @ 5% ($M)	IRR (%)	Initial Capex ($M)
80	7,933	7,306	1,530	8.2	5,343
90	8,752	6,751	1,040	7.0	6,011
100	9,570	6,197	547	6.0	6,679
110	10,388	5,643	48	5.1	7,347
120	11,207	5,091	456	4.3	8,015

Note: Capex = capital cost

Table 22-6: Project Sensitivity to Oil Price (Base Case is highlighted)

Factor (%)	Factor Total Opex (%)	LOM Cash Cost $/oz	LOM Cash Flow ($M)	NPV @ 5% ($M)	IRR (%)	Opex ($/t ore)
80	571	6,503	686	6.2	31.23	571
90	578	6,350	616	6.1	31.64	578
100	585	6,197	547	6.0	32.05	585
110	591	6,045	477	5.9	32.46	591
120	598	5,892	407	5.7	32.86	598

Note: 1.28% change in total Opex for 10% change in oil price; Opex = operating cost

Table 22-7: Project Sensitivity to LNG Price (Base Case is highlighted)

Factor (%)	Factor Total Opex (%)	LOM Cash Cost ($M)	LOM Cash Flow ($M)	NPV @5% ($M)	IRR (%)	Opex ($/t ore)
80	79.17	559	6,731	780	6.4	30.49
90	89.58	572	6,464	663	6.2	31.27
100	100.00	585	6,197	547	6.0	32.05
110	110.42	598	5,931	429	5.8	32.83
120	120.83	611	5,663	312	5.6	33.61

Note: 10.42% change in power Opex for 18% change in LNG price based on power being 96% of LNG use; LNG = liquefied natural gas, Opex = operating cost

22.5 Comment on Section 22

In the opinion of the QPs, using the financial parameters and assumptions set out in this Report, the after-tax Project NPV at a discount rate of 5% from 1 January 2014 is $547 M and the internal rate of return (IRR) is 6%. The cumulative, undiscounted, after-tax cash flow value for the Project is $6,197M and the after-tax payback period is 9.2 years. The Project is most sensitive to variations in the gold price, and is less sensitive to variation in operating cost or capital cost.

NovaGold has advised AMEC that NovaGold has opening tax pools of approximately $102 million that can be applied against NovaGold's share of income from the Project which would increase NovaGold's prorate share of Project NPV.

23.0 ADJACENT PROPERTIES

There are no adjacent properties that are relevant to this Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Preliminary Development Schedule

As part of FSU2 activities, a preliminary development schedule was compiled. This schedule envisages the following scopes of work to be undertaken, in order, over an envisaged seven year construction period. Dates in this subsection are for illustrative purposes only, as no Project permits and approvals have been received, and Project development and construction has not been approved by the respective Boards of Donlin Gold, NovaGold and Barrick.

The proposed work program is as follows:

- Award of EPCM and commencement of engineering from approximately the end of 2012 to 2015. To meet the Project's engineering and procurement requirements, a compressed, workforce-loaded basic engineering phase will commence January 2013 and last ten months. During this phase, the operating and contracting strategy for barges and tugs must be finalized to ensure that the barging requirements for the initial 2015 freight campaign are in place. Actual specifications for the tow and barges and for equipment procurement would come later in 2013 and 2014.
- Vendor negotiations for the long-lead equipment and contracts that must be procured in 2013 would be held the second and third quarter of 2013. Water management and any other mining and process engineering optimization work would also be completed during this time. Deliverables will include finalizing the process flow diagrams, procuring long-lead items, and refining the capital cost estimate as engineering proceeds. These tasks will be integrated with the early front-end needs of the permitting process
- Detailed engineering would follow directly on basic engineering. This phase would include a three-month overlap with basic engineering to support the preparation of procurement and contract work packages for the tendering process. Detailed engineering would last for a total of 18 months
- Receipt of environmental approval to commence construction may be received toward the end of 2015; for schedule development purposes, the end of October 2015 was assumed. No site work can proceed before environmental approvals are granted.
- To provide access for construction as soon as possible, 2015 earthworks construction equipment and a 500-bed portion of the construction camp will be staged at Crooked Creek during the 2015 barge season and then be moved to site during the winter of 2015 along an established winter road route. During the 2015

barge season, road-building equipment, three 49-bed mobile construction camps, platework for two fuel storage tanks, and some erection equipment will be staged at Crooked Creek for transport to site over the winter road during the winter of 2015/2016.

- An all-weather access road from Jungjuk to the plant site will be in place for the 2016 barge season, permitting the site earthworks construction fleet to be mobilized.
- Given the water management issues associated with construction, pit development, and the waste dump area, the contact water pond structures in the American Creek area will be given early priority in the construction program. The nature of the soils in this area is such that material excavation must be done in the winter months from December to April. The construction water treatment plant will be installed by March 2016 for use during early construction and pit development. Construction and mining equipment mobilized in 2016 will be used to construct the contact water pond, freshwater ponds, and tailings dam. During the summer of 2016, the construction infrastructure (fuel storage, construction camp, site rough grading, explosives storage areas) will be completed and ready for the winter earthworks program. The airstrip is currently scheduled to be ready for use 15 November 2016.
- The construction camp at site will be expanded from the initial 500 beds to an ultimate 2,500 beds by February 2017. The additional camp modules will be delivered to site during the 2016 barge season via Jungjuk. The permanent camp would be delivered to site during the 2017 barge season and will be complete and ready for occupancy by March 2018.
- Construction of the process facilities will commence with concrete work in May 2017. The concrete batch plant will be delivered to site during the 2016 barge season. Work on the primary crushing, coarse ore storage, and process facilities will proceed through 2017 into the last half of 2018. Commissioning of the process plant and associated facilities will commence in December 2018, with start-up scheduled for the end of March 2019. The power plant will be completed by 30 October 2018 and will be used during commissioning.
- The truckshop service bays will be completed in January 2018 to support the ramp-up of mining activities. The administration section of the truckshop will be ready for occupancy in early 2018. Construction of the tailings storage facility is seasonally constrained, with areas below the floodplain scheduled for winter construction and higher areas scheduled for year-round construction. The construction and small mining fleets will be used to build the dam foundations and seepage collection system, beginning in December 2016. The dams and impoundment liner will be finished in October 2018 to support commissioning and start-up activities.

24.2 Project Opportunities

Opportunities exist for refinements within the Mineral Resource modelling to address:

- Potential for increase in recoveries from material within the upper part of the currently-estimated Mineral Resource
- Definition of additional mineralization within known gold-bearing overburden material
- Improved pit slope designs
- Modelling of SWIR (short-wave infrared reflectance) alteration mineral suites could help break out ore types based on clay and carbonate content
- Use of sulphur rather than gold values may produce more-accurate default regression equations for arsenic, mercury, and antimony
- Development of an independent variogram model for NP
- Infill drilling that could support potential upgrade of the Inferred Mineral Resources within the design pit and explore areas within the sedimentary rock packages that are currently under-drilled
- Step-out drilling to evaluate extensions to known gold zones outside the current pit shell, in particular in the Akivik and East ACMA areas.

Opportunities within the mine plan include:

- The implementation of a waste crushing and conveying system could result in lower operating costs, as well as reduced labour, emissions, and barging requirements; however, power cost increases may balance out any cost savings
- The mine plan assumes a high degree of blending to satisfy plant feed constraints. If greater variation in plant feed is permissible, then material movement and cost will decrease. There is further potential to reduce the amount of assumed stockpile re-handle from the currently assumed 45%.

Opportunities within the metallurgical and process areas include:

- Better definition of the oxide zones could reduce the tonnage of oxide ore that reports to the mill, resulting in higher calculated flotation gold recovery
- Reagent costs in flotation could be decreased by substituting the selected products with stronger, alternative reagents that could still achieve the same, or similar,

metallurgical performance. This would reduce the amount of reagents that need to be purchased, transported, and stored on site

- Further work is warranted to determine the optimal recovery of heat from site sources for use in the process plant, allowing for higher process efficiencies.

25.0 INTERPRETATION AND CONCLUSIONS

Following review of the key parameters and assumptions contained in the FSU2 report, the QPs have reached the following interpretations and conclusions.

25.1 Agreements, Mineral Tenure, Surface Rights, and Royalties

The Project is a 50:50 partnership between NovaGold Resources Alaska, Inc, a wholly-owned subsidiary of NovaGold) and Barrick Gold U.S. Inc, (a wholly-owned subsidiary of Barrick). The partners use an operating company, Donlin Gold LLC (Donlin Gold) to manage the Project.

Information from legal and Donlin Gold experts support that the mining tenure held is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves.

The Donlin exploration and mining lease currently includes a total of 72 sections in the vicinity of the deposit, and additional partial sections associated with the Project infrastructure, leased from Calista Corporation, an Alaska Native Corporation that holds the subsurface (mineral) estate for Native-owned lands in the region. Calista owns the surface estate on 27 of these 72 sections.

In addition to the 49,261 acres (20,081 hectares) leased from Calista, Donlin Gold holds 242 Alaska State mining claims comprising 31,740 acres (12,845 hectares), bringing the total land package to 81,361 acres (32,926 hectares).

A separate Surface Use Agreement with The Kuskokwim Corporation (TKC), an Alaska Native Village Corporation that owns the majority of the private surface estate in the area, grants non-exclusive surface use rights to Donlin Gold on at least 34 sections overlying the mineral deposit, with provisions allowing for adjusting that area in conjunction with adjustments to the subsurface included in the Calista lease.

The Lyman family owns a small (13 acre) private parcel in the vicinity of the deposit and holds a placer mining lease from Calista that covers approximately four sections.

Donlin Gold, through native lease agreements, holds a significant portion of the surface rights that will be required to support mining operations in the proposed mining area. Negotiations with TKC will be required for surface rights for additional lands supporting mining and access infrastructure.

The currently identified Mineral Resource and the bulk of the primary infrastructure (mill and waste rock facilities) are located on the leased lands. Additional lands required for the Jungjuk port site, road to the port site, gas pipeline, and tailings storage facility in Anaconda Creek are located on a combination of Native, State of Alaska, and Federal (Bureau of Land Management, BLM) lands. Rights-of-Way will be required from the State and BLM for the road and pipeline alignments where they cross state and federal lands, respectively. Discussions regarding the extension and expansion of the TKC Surface Use Agreement and the disposition of the Lyman family land parcel and lease are ongoing.

The Calista Corporation and The Kuskokwim Corporation (TKC) receive payments for land access and use.

There are no Government royalty obligations. Royalties are payable to the Calista Corporation that vary depending on the stage Project development.

Donlin Gold advised AMEC that Barrick has maintained all of the necessary permits for exploration and camp facilities.

25.2 Geology and Mineralization

The Lewis–ACMA part of the district is a low sulphidation, reduced intrusion related, epizonal system with both vein and disseminated mineral zones. Other zones of the deposit are more akin to a high-level, reduced intrusion-related vein system.

Knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization, and the mineralization style and setting is sufficient to support Mineral Resource and Mineral Reserve estimation.

25.3 Exploration, Drilling, and Data Analysis

The exploration programs completed to date are appropriate to the style of the deposits and prospects within the Project. The exploration and research work supports the orogenesis interpretations.

Approximately 1,834 exploration and development diamond core (90%) and reverse circulation (RC) (10%) drill holes, totalling 1,337,321 ft (407,720 m), were completed from 1988 through 2010. All but about 20% (district exploration, carbonate resource, facilities condemnation, hydrology, infrastructure engineering) of this drilling was utilized for the current resource model.

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

Sampling methods are acceptable, meet industry-standard practice, and are acceptable for Mineral Resource and Mineral Reserve estimation.

The quality of the gold analytical data are sufficiently reliable to support Mineral Resource and Mineral Reserve estimation and sample preparation, analysis, and security are generally performed in accordance with exploration best practices and industry standards.

The data verification programs undertaken on the data collected from the Project adequately support the geological interpretations, the analytical and database quality, and therefore support the use of the data in Mineral Resource and Mineral Reserve estimation.

25.4 Metallurgical Testwork

Process development work has determined that pressure oxidation is the preferred method of pre-treatment. Extensive testwork on composites has shown that acceptable gold recoveries can be produced through a combination of flotation pre-concentration, POX, and CIL cyanidation.

Air flotation using the MCF2 flowsheet provides an estimated life-of-mine (LOM) average of 93.0% recovery, with CIL recoveries after POX at approximately 96.6% for an estimated combined plant total gold recovery of 89.8%.

Process selection is supported by extensive testwork. Placer Dome undertook an initial phase of testwork from 1995 to 1999 to define the basic process. Early on, it became apparent that direct cyanidation or CIL of ore or flotation concentrate returned very low recoveries. Pre-treatment by oxidation was considered necessary. Placer Dome testwork included grinding, gravity concentration, flotation, POX, cyanidation, and neutralization.

Subsequently from 2002 to 2005, Placer Dome also explored HPGR comminution, arsenopyrite/pyrite separation, nitrogen aerated flotation, and oxidation both by bio-oxidation and pressure autoclave. At the end of 2005, another round of work began with some testing at G&T, but this was interrupted by the acquisition of Placer Dome by Barrick Gold, which subsequently assumed management of remaining testwork.

Major programs at the bench-scale level were initiated in 2006 to test grinding, flotation, POX, and neutralization. In addition to bench-scale work, major pilot-plant runs were performed in flotation, POX, and neutralization at the Barrick Technology Centre, SGS-Lakefield, G&T, and Hazen Research (Golden, U.S.A.). Both bench-level and pilot-plant scale testwork were conducted to develop process parameters and expand engineering information for use in FSU2.

25.5 Mineral Resource and Mineral Reserve Estimation

Estimations of Mineral Resources and Mineral Reserves for the Project conform to industry best practices, and meet the requirements of CIM (2010). An open pit extraction scenario is appropriate to the style of mineralization and LG shells have been used to constrain the estimates. Assumptions used in the shells are appropriate to the envisaged process route and mine plan.

Measured and Indicated Mineral Resources total 541 Mt grading 2.24 g/t Au, and are inclusive of Mineral Reserves. Inferred Mineral Resources total 92 Mt grading 2.02 g/t Au.

Mineral Resources are contained within a conceptual Measured, Indicated and Inferred optimized pit shell shell and reported using a constant NSR cut-off of $0.001/t calculated using the following parameters: gold price of US$1,200/oz; variable process cost based on 2.1874 * (S%) + 10.65; administration cost of $2.29/t; refining, freight & marketing (selling costs) of $1.85/oz recovered; stockpile rehandle costs of 0.20/t processed assuming that 45% of mill feed is rehandled; and variable royalty rate, based on royalty of 4.5% * (Au price – selling cost). The net sales return (NSR) was calculated using the formula: NSR = Au grade * Recovery * (Price of Au – (1.85 + (Price of Au – 1.85) * 0.045))) - (10.654 + 2.1874 * (S%) + 2.29 + 0.20) and reported in US$/tonne.

Factors which may affect the Mineral Resource estimate include the commodity price; changes to the assumptions used to generate the NSR cut-off; changes in interpretations of fault geometry; changes to the 0.25 g/t threshold used for defining the mineralized indicator shells; changes to the search orientations used for grade estimation in the ACMA area, results of a review of the Measured classification criteria; and changes to the assumptions used to generate the LG pit constraining the estimate, in particular slope design assumptions.

Proven and Probable Mineral Reserves total 505 Mt grading 2.09 g/t Au. Mineral Reserves are contained within Measured and Indicated pit designs, and supported by a mine plan, featuring variable throughput rates, stockpiling and cut-off optimization. The pit designs and mine plan were optimized on diluted grades using the following economic and technical parameters: Metal price for gold of US$975/oz; reference mining cost of $1.67/t incremented $0.0031/t/m with depth from the 220m elevation (equates to an average mining cost of $2.14/t), variable processing cost based on the formula 2.1874 x (S%) + 10.65 for each $/t processed; general and administrative cost of US$2.27/t processed; stockpile rehandle costs of 0.19/t processed assuming that 45% of mill feed is rehandled; variable recoveries by rocktype, ranging from 86.66% in shale to 94.17% in intrusive rocks in the Akivik domain; refining and freight charges of US$1.78/oz gold; royalty considerations of 4.5%; and variable pit slope angles, ranging from 23º to 43º. Mineral Reserves are reported using an optimized net sales return value based on the following equation: NSR = Au grade * Recovery * (Price of Au – (1.78 + ((Price of Au – 1.78) * 0.045))) - (10.654 + 2.1874 * (S%) + 2.27 + 0.19) and reported in US$/tonne. The life of mine strip ratio is 5.48. The assumed life-of-mine throughput rate is 53.5 kt/d.

Factors which may affect assumptions used in estimating Mineral Reserves include the commodity price; unrecognized structural complications in areas with relatively low drill hole density that could introduce unfavourable pit slope stability conditions; changes in interpretation of the fault orientiations, in particular the Vortex and Lo Faults; changes in orientations of the bedding or ash layer orientations which may necessitate flatter slope angles than currently assumed; in-pit and pit wall water management if water inflows are higher than predicted; and the likelihood of obtaining required permits and social licenses to construct the gas pipeline and operate the planned mine in the timeframe envisaged in the study.

AMEC notes that based on an assessment of qualitative, non-technical factors, Barrick treats mineralization at Donlin as Measured and Indicated Mineral Resources, rather than Proven and Probable Mineral Reserves for securities reporting, accounting, and other public disclosure purposes.

25.6 Mine Plan

Two pits are planned, one at Lewis (nine phases), the other at ACMA (six phases); the pits will partly coalesce toward the end of the mine life. The pits have similar mineralization characteristics, with ore-grade gold hosted in both intrusive and sedimentary rock units. The grade of the gold mineralization in ACMA is higher than in the Lewis area, and because of the higher grades, the ACMA pit will be the first developed.

The operating mine life is estimated to be 25 years based on a nominal processing rate of 59,000 stpd (53,500 t/d). 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.

Mineralization will be mined by a combination of bulk and selective mining on 12 m and 6 m benches, respectively. Ramps in final walls have a design width of 131 ft

(40 m) and a gradient of 10%. A nominal minimum mining width of 492 ft (150 m) was used for phase design. The mine plan assumes the ACMA pit will be backfilled.

Mining will use a conventional truck-and-shovel fleet. Large electric hydraulic shovels mining the full 40 ft (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 20 ft (6 m) split benches, thereby avoiding the need for a mixed fleet of hydraulic shovels. All shovels will be equipped with GPS positioning to allow real-time updates of the digging face and delineation of the ore–waste contacts. Large front-end loaders (FELs) will be used as a backup to the shovels and to work on less-productive faces and muckpile cleanup. Large 400 st (360 t) capacity haul trucks will be used for transporting both ore and waste out of the pit. All trucks will be equipped with a GPS dispatch system for tracking ore and waste from the mine face to the designated stockpiles or dumps. The mine and fleet design is appropriate for the Mineral Reserves defined.

It is expected that any future mining operations will be able to be conducted year-round, and will include appropriate provisions for winter operating conditions.

25.7 Process Design

Run-of-mine (ROM) ore at 59,000 stpd (53,500 t/d) from the Donlin deposit will be crushed in a gyratory crusher followed by a semi-autogenous grinding (SAG) mill and two-stage ball milling, addition of chemicals, and a flotation circuit (MCF2). The primary ball milling circuit will produce a P_{80} particle size of 120 to 150 µm as feed to the primary rougher flotation section. The secondary ball milling circuit will produce a P_{80} particle size of 50 µm as feed to the secondary rougher flotation section.

Gold-bearing sulphides, recovered by flotation, generate a concentrate containing 7% sulphur. The concentrate is refractory and will be treated in a pressure oxidation circuit prior to cyanidation. Overall gold recovery from flotation, pressure oxidation and cyanidation is estimated to be in the order of 89.83%. Excess acid from the autoclave circuit will be neutralized with flotation tailings and slaked lime. Tailings from the process will be impounded in a zero-discharge tailings storage facility; water reclaimed from here will be re-used in the process plant.

Mineralogical studies have shown that the gold is not visible. Testwork analysis indicates a high level of association of gold with arsenopyrite. Other sulphides such as pyrite and marcasite are also present, with reduced tenors of gold. Organic carbon, a potential preg robber, is present in the sedimentary ore. It is also present at lower levels in the intrusive ores, believed to be in the form of well-ordered graphite. This form of organic carbon is possibly less likely to preg-rob.

The average Bond work index for the ore is in the range of 15 kWh/t. Flotation work has shown that kinetics are initially rapid, but to achieve high recoveries, a combined primary and secondary rougher residence time over 100 minutes, together with a high reagent loading in the system, is required. Clay-like minerals will affect slurry viscosity and settling. Slurry density in the underflow will be less than 50% solids for the concentrate thickeners.

Partially geologically oxidized (altered) ore in the deposit, up to 7% of the mill feed, is the key non-performing ore type in the flotation circuit. Degradation of the sulphide ore via oxidation in the stockpile will also affect the flotation recovery, applied as 5% recovery loss within flotation on all ores stockpiled for longer than one year.

Pressure oxidation (POX) has been shown to be successful in releasing the valuable constituents, under certain conditions. To optimize oxidation conditions, the water systems design has been modified to use the highest-quality water in the oxidation circuit. The autoclave design incorporates variable level control to provide better control over operating residence time.

The oxidation circuit discharge will be washed to reduce lime load in carbon-in-leach (CIL); the washed solution will be neutralized by the use of high-carbonate flotation tails to further reduce plant lime consumption prior to tailings disposal.

Gold recovery by CIL has proven successful in treating Donlin ores and is estimated to be 96.6%. Rheological investigation and CIL testing results have determined that a relatively low CIL feed density of 35% solids should be adopted. In addition, to control lime usage, the CIL circuit will be operated at a pH of approximately 11.0.

Given the plan to use stockpiles to manage the ore blend into the process from the perspective of gold, sulphur, carbonate, and hardness, allowances were made for ore aging or stockpile degradation for the life-of-mine feed. Ore oxidized through weathering will have a slower flotation response than fresh rock. In general, ore at Donlin does not contain highly reactive sulphide species, and testwork has shown no statistical deviation over a one-year period. While data from a longer timeframe are not presently available, the testwork results for oxidized material show some degradation. Consequently, there is no effect on recovery for material stockpiled for less than one year (sulphide “fresh” material), and a recovery deduction of 5% has been applied to gold and sulphur recoveries for sulphide material stockpiled for longer than one year.

Alternative flowsheets to flotation-POX-CIL were considered, including whole ore pressure oxidation, roasting a flotation concentrate, and bio-oxidation (BIOX). None of these proved to be a viable economic alternative to the flotation-POX-CIL route.

25.8 Infrastructure Considerations

The Project will require construction of significant infrastructure to support the planned producing facilities. Key infrastructure will include:

- Access road
- Airstrip
- Barge cargo terminal
- Marine cargo terminal
- Two open pit mines
- Process plant site
- Primary crusher area
- Fuel storage compound
- Mining and road fleet truckshops
- Water management dams
- Water management pumping systems
- Power plant
- Tailings storage facility (TSF)
- Waste rock storage facility (WRF)
- Gas pipeline
- Construction and permanent accommodation camps.

The mine access road will traverse varied terrain from the mine site to the planned Kuskokwim River dock site at Jungjuk. Jungjuk is eight nautical miles (13 km) downstream from the village of Crooked Creek; there is no road connection between the two locations. To the mine site battery limits, the Jungjuk access road will be 27 miles (44 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.

A new airstrip is required to support long-term development and mine operations at the site. The airstrip will be approximately 9 miles (14 km) by road west of the mine site. DHC Dash 8 and the Hercules C-130 aircraft will be able to land on the gravel strip.

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. The cargo will be unloaded and either placed into storage or reloaded onto river barges to be towed upriver to the Jungjuk port site.

The plant site and fuel storage compound will be in the Anaconda valley, above the tailings storage facility. The elevation of the plant site platform has been designed to allow for a gravity tailings system over most of the mine life, with the possibility of pumping at the tail end of the Project. An extensive area adjacent to the fuel farm will be provided for container storage, including a covered structure for the storage of cyanide and other reagents.

The primary crusher and mine fleet truckshop will be on a ridge on the south side of American Creek. Other facilities, including an assay laboratory, administration offices, and mill change-rooms, have been located on three floors within the concentrator building.

The main objectives of the water management plan for the Donlin Gold Project are to avoid discharge of contact, tailings, or process water during construction and operations; 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. Water systems to service site requirements include remote pump stations and barges and many miles of overland pipelines that extend from the far eastern end of the Anaconda TSF to the far western limits of the two open pits. Included in the management systems are a lower contact water dam (CWD) and a lower contact water pond (CWP). Water in the CWP will be used in the process plant. Freshwater supply will come from a series of wells to be located south of Omega Gulch, near Crooked Creek.

Two contact water dams designed to collect runoff and seepage water from the WRF will be constructed in the American Creek valley. The Upper CWD will be located in the upper reaches of American Creek, upstream of the WRF, and the lower CWD downstream of the WRF, upstream of the confluence of Rob's Gulch with American Creek. The CWDs will be constructed of compacted rockfill and lined. A freshwater dam (FWD) in Snow Gulch to the north of American Creek will store fresh water required by the mill during periods of drought.

The power plant will consist of two combined-cycle gas turbine (CCGT) systems and two simple-cycle gas turbine (SCGT) backup units. The total connected electric load at the Donlin site is estimated to be 227 MW, the average running load approximately 153 MW, and the peak load 184 MW.

The tailings storage facility in the Anaconda Creek basin will be a fully-lined impoundment with a cross-valley dam at the downstream end of the valley. The tailings dam will be constructed of compacted rockfill using the downstream method. The TSF will have an ultimate capacity of 334,298 acre-ft (412.35 Mm^3), corresponding to an ultimate impoundment surface area of 1,357 acres (549 ha). The total catchment area of the TSF will be 3,755 acres (1,520 ha). The TSF inflow design flood is the 200-year return period snowmelt and 24-hour probable maximum precipitation (PMP). The stability of the tailings dam yields static and pseudo-static factors of safety of at least 1.5 and 1.15, respectively. The TSF has been designed to withstand the maximum credible earthquake (MCE)

The waste rock facility (WRF) will be constructed in the American Creek valley, immediately east of the open pit. The ultimate footprint of the facility covers an area of approximately 3.5 mi^2 (9 km^2). Approximately 2,449 Mst (2,222 Mt) of waste rock will be placed in the WRF, plus 46 Mst (42 Mt) of overburden not required for reclamation purposes. The elevation of the top lift of the WRF will be approximately 1,705 ft (520 m) asl, resulting in a maximum dump height of about 1,115 ft (340 m) and a maximum dump thickness about 985 ft (300 m). Early in the mine life, potentially acid generating (PAG) waste rock will be placed into the WRF. A dumping plan has been developed that separates PAG materials from non-acid generating (NAG) materials within the dump.

The construction camp will be built on a bench near the process plant. The camp will accommodate 2,560 people. The permanent accommodation complex will be located just off the main access road, approximately four miles (6 km) west of the plant site. The complex will accommodate a maximum of 638 people during operations.

25.9 Markets and Contracts

NovaGold will be able to market the company's share of the gold produced from the Donlin Gold Project. Sales contracts that could be negotiated would be expected to be within industry norms. However, the majority of production would be expected to be spot marketed.

25.10 Environmental, Social Issues and Permitting

Key areas of environmental concern are likely to be the use of mercury and cyanide in the process plant, water management issues, transportation of cargoes, and the impact of any proposed development on flora and fauna. Donlin Gold is of the opinion that these issues have been, or can be, addressed and mitigated through a combination of good baseline data collection, diligent engineering and Project design, and thorough public consultation.

Any mining operation at Donlin will require a considerable number of permits and authorizations from both Federal and State agencies. To support successful application for the more than 80 permits, extensive baseline environmental information, supporting scientific analysis, and detailed engineering have been compiled over a 15-year period.

Baseline data were collected to support Project design; to determine and implement environmental controls to mitigate impacts; and to sufficiently characterize the environment in support of permit applications and environmental impact assessments. The environmental baseline data will also provide a reference point against which environmental conditions can be evaluated during operations to facilitate early detection of potential changes that may occur during Project development and future operation.

Permitting timelines will be controlled by the requirements of the Federal National Environmental Policy Act (NEPA) review and State requirements for meaningful public and agency participation to determine if the Project is in the State‘s best interest. Upon completion of the NEPA process, a Record of Decision (ROD) will be prepared that approves the preferred alternative for the Project, describes the conditions of the approval, and explains the basis for the decision. The State permitting process typically is not finalized until the NEPA process is completed. Each Federal and State permit will have compliance stipulations requiring review and possibly negotiation by the applicant and appropriate agency. Upon final issuance of permits and authorizations, the Environmental Management System (EMS) for the Donlin Gold Project will be fully implemented.

The region has a complex political and social structure, represented by a diverse group of social, business, and governmental entities. Relationships between these various entities are often complex and are influenced by competing political and economic interests. To successfully acquire the support and "social license" required to develop and operate this Project, a process of ongoing engagement and consultation will be continued with all of these entities throughout the permitting process, construction, operation, and closure of the Project. Calista Corporation and The Kuskokwim Corporation, the two primary Native business entities of the region, each have a financial interest in the Project. Donlin Gold is focusing on sustainable development to benefit local communities over the long term by providing opportunities for direct employment, local procurement, and community development projects.

A closure plan has been developed. In addition to the basic goal of reclaiming 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 areas no longer required for active mining.

Area and component-specific reclamation plans governing actual reclamation activities will be developed further as Project designs become more refined and alternatives are identified during permitting.

Surface water and groundwater monitoring systems for process components will remain in place up to and possibly beyond 30 years, depending on compliance history and until each specific facility has been stabilized, physically and chemically, to the satisfaction of the applicable state and federal regulatory agencies.

A post-closure water treatment plant (WTP) will be built at the site. More testwork will be conducted before permitting to confirm the final design and flowsheet selection for the WTP. The results of ongoing bench- and pilot-testing will also be used to update the water treatment process design for the post-closure WTP.

Long-term or permanent diversions, water treatment, physical barriers, and signage will be monitored and maintained as needed until all closure standards are met, reclamation surety has been released, and property management reverts to the landowner. The decision on what constitutes final closure and the release of any outstanding financial surety will require the concurrence of State and applicable Federal agencies.

The final reclamation cost estimate for the FSU2 is estimated at $131.3 million. The additional cost of closure/abandonment of the natural gas pipeline is estimated at $9.6 million. The funding amount is estimated at $8.6 million provided annually over 32 years, including the construction period and 27-year LOM, for a total of $273.7 million.

25.11 Capital and Operating Cost Estimates

The FSU2 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% +30% of estimated final costs, per AACE Class 3 definition. All costs are expressed in second quarter (Q2) 2011 U.S. dollars. No allowances were included for escalation, interest during construction, taxes, or duties.

The total estimated capital cost to design and build the Donlin Gold Project described in this Report is $6,679 million, including an Owner-provided mining fleet and self-performed pre-development.

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 have been prepared in second quarter (Q2) 2011 U.S. dollars with no allowances for escalation, sales tax, import duties, or contingency.

The estimated life-of-mine operating costs are $5.42/t mined or $34.99/t milled, or $581/oz.

25.12 Financial Analysis

Using the financial parameters and assumptions set out in this Report, the after-tax Project NPV at a discount rate of 5% is $547 M and the internal rate of return (IRR) is 6%. The cumulative, undiscounted, after-tax cash flow value for the Project is $6,019 M and the after-tax payback period is 9.5 years.

The Project is most sensitive to variations in the gold price, and is less sensitive to variation in operating cost or capital cost.

25.13 Preliminary Development Schedule

As part of FSU2 activities, a preliminary development schedule was compiled. This schedule envisages a seven-year construction period. No Project permits and approvals have been received, and Project development and construction has not been approved by the respective Boards of Donlin Gold, NovaGold and Barrick.

25.14 Conclusions

AMEC considers that the scientific and technical information available on the Project can support proceeding with additional data collection, trade-off and engineering work and preparation of more detailed studies. However, the decision to proceed with a mining operation on the Project is at the discretion of Donlin Gold, NovaGold and Barrick.

26.0 RECOMMENDATIONS

Donlin Gold has completed a feasibility study and two updates on the study. A decision to proceed with any mine development plans would be made by the partners.

As a consequence, AMEC's recommendations are restricted to activities that would support permitting and detailed engineering studies. These activities are envisaged as a single phase of work, with no item or area dependent on results of another. The estimated total cost of the proposed work is in the range of $135,000 to $200,000.

The work program is outlined in the following sub-sections.

26.1 Geology and Modelling

- Complete deposit- and district-scale outcrop mapping during the permitting phase to advance geologic understanding of the deposit to support step-out and exploration programs
- Update faults and sedimentary rock structural models to help reduce pit slope design uncertainties and support step-out and exploration programs
- Conduct an independent review of the NAG/PAG classifications and models prior to mine startup.

These work programs are estimated to cost approximately $10,000 to $15,000, including logistical considerations.

26.2 Data

- In the bottom of the ACMA pit, the location of the intrusive and hence the orebody, is subjective with the information available, which could lead to non-optimized location of the highwall with significant economic consequences. AMEC recommends that additional northwest-trending drill holes be drilled to confirm the location of the orebody at the bottom of the ACMA pit before mining commits to the final highwall design.
- A comparison between the trench samples and core/rotary samples should be performed to determine if there is any bias that may affect the resource estimation.

These work programs are estimated to cost approximately $10,000 to $15,000, including logistical considerations.

26.3 Geotechnical

- Recommended open pit slope angles are primarily based on the geological characteristics of the rocks in the Donlin area. To reduce uncertainty in the current design criteria and potentially further optimize the criteria at the next stage of design, additional geological work should be undertaken during the next phase of engineering, as follows:
  - conduct additional laboratory testing to estimate the intact strength of the main greywacke (mnGWK) unit
  - optimize the footwall slope designs to account for local (bench-scale) variation in bedding dips
  - update the structural model, geological interpretation, and geotechnical evaluation beyond the current pit limits to expand on the analysis of oriented core data, the modelling, and the comprehensive compilation of GIS geo-data that began in 2008
  - review the mine-scale fold and fault models, including the potential for a fault zone in the lower American Creek area
  - align the local geology with regional geological interpretations.
- As updated geological information becomes available, the slope design criteria should be optimized. Additional design charts for use in developing footwall geometries for mining are recommended. The estimates of in-situ block size should also be updated to assist in blast design as well as for the rock drain design for the waste rock facility.

These work programs are estimated to cost approximately $15,000 to $20,000, depending on the complexities that may be encountered for representation in the updated models.

26.4 Pit Slope Dewatering

- The pit slope dewatering plan incorporates 80 in-pit wells during the life of the pit. Some of the wells have a relatively short life span and may be impractically located. Therefore, it is recommended that the dewatering plan continue to be reviewed in future studies with input from the mine planners to optimize the locations of in-pit wells. To collect sufficient information to support detailed design of the pit dewatering system at the next stage of Project design, it is recommended that two additional deep test wells be installed for long-term pumping tests.
  - One of the wells should be located adjacent to American Creek and should intersect lower-hydraulic conductivity bedrock at depth in the south ACMA pit

wall. This will allow further evaluation of the ability of vertical wells to depressurize bedrock at depth in a more complex geologic and structural setting than was considered in the previous prototype dewatering well test.
- The second well should be located near Crooked Creek at the confluence with American Creek in the north ACMA pit wall. The well testing program should be designed to investigate the potential for significant inflows to the pit through alluvium and shallow bedrock that potentially could be exposed in the pit wall.
- In addition to individual tests at these locations, both new wells and the existing prototype dewatering well (MW07-11) should be pumped in unison at the highest sustainable extraction rate possible for as long as possible to investigate large-scale hydraulic connectivity and/or compartmentalization of the rockmass.
- As more information is acquired, a detailed schedule of the mine dewatering plan should be developed showing the installation of in-pit wells and pumping infrastructure on the period plans to confirm access and practicality as mine phases progress over time.

These work programs are estimated to cost approximately $5,000 to $10,000, depending on the duration and number of pumping tests required.

26.5 Mine Plan/ROM Stockpiling

- The mine plan provides a maximized NPV based on multiple plant feed constraints. As the schedule is optimized further, there may be opportunities to review or relax some of these constraints by incorporating additional capital items. This requires further evaluation.

This work program is estimated to cost approximately $5,000 to $7,000, depending on the duration and number of alternative schedule considerations required.

26.6 Process, Metallurgy, and Water Treatment

- Ongoing refinements in the process metallurgical area could lead to savings in capital costs, operating costs, and improved NPV. The following work is recommended:
  - refine the oxide mineralization model
  - complete variability flotation testwork on the core drill material obtained during in-fill drilling programs since the second half of 2007
  - conduct additional POX pilot testing under design conditions to increase the test result database and decrease the variance in the confidence limits.

These work programs are estimated to cost approximately $70,000 to $100,000, depending on the number of pilot and variability tests required.

26.7 River Surveys

- The bathymetric survey by TerraSond of the navigation channel in the Kuskokwim River alongside Nelson Island indicates that for most of its length, the river is only about 100 ft (30.5 m) wide, or just slightly wider than the proposed barge tow, and that channel width varies little between low and high water. As a consequence only a single tow will be able to use this stretch of the river. If barge tows heading downstream to Bethel are given priority, then barges moving upstream approaching Nelson Island would have to wait until the Bethel-bound tow clears the channel and it is once again safe to proceed to Jungjuk. It is recommended that additional survey work, include the following, be undertaken in the next phase of the Project:
  - extend the bathymetric surveys to Crooked Creek, if it is to be used as a temporary landing site during the early phase of construction
  - install streamflow measurement gauges on the Kuskokwim river at Bethel and Jungjuk and monitor river levels to confirm design assumptions.

These work programs are estimated to cost approximately $20,000 to $30,000.

26.8 Third-Party Logistics Service Providers

- As currently conceived, third-party logistics service providers will be contracted to design, supply, and operate the fuel and general cargo supply chains during both the construction and operations phases of the Project. The negotiations for these contracts, construction of infrastructure, and acquisition of tugs and barges could take up to three years. To ensure that the facilities, infrastructure, and support services are in place to support the build-up in construction activities in Year -2, when the total volume of cargo is estimated to exceed 210,000 st (190,000 t), negotiations with potential service providers should commence immediately upon the Project receiving approval to proceed to detail design in Year -5.

26.9 Permitting, Environment, and Social

- Detailed Spill Prevention and Response Strategies – The potential for, and consequences of, a large fuel or chemical spill resulting during barge transportation of consumables should be addressed through the continuing development of detailed spill response strategies and fuel handling procedures based around the established procedures, logistics, and supply strategy. An Oil Discharge Prevention and Contingency Plan (ODPCP), which details spill scenarios and specific response strategies, should be developed specifically for water-based activities. This plan should be reviewed by the permitting agencies before submitting the permit applications.

- Mercury Data Collection and Baseline Studies – Ongoing, proactive baseline studies should continue to evaluate current levels of mercury in ambient air, stream sediment, vegetation, soils, water, and fish tissue. This will assist in apprising local stakeholders of existing conditions and allow the company to account for mercury (Hg) emission controls from the basic engineering phase through to detailed design. Design of the plant thermal units include state of the art Hg controls, which consider the Environmental Protection Agency's Maximum Achievable Control Technology (MACT) regulations for gold processing facilities, published in 2011. Ongoing work should continue to evaluate available and emerging technologies to ensure that the best available control technology is considered during final engineering.
- Monitor Alaska Water Quality Regulations – Developments in the State of Alaska with regard to mixing zone regulations should continue to be monitored closely. Advances in treatment technologies for dealing with post-closure water discharge should continue to be monitored, specifically with regard to selenium, arsenic, sulphate, and emerging passive treatment technologies.
- Develop Local and Regional Support for the Project – Local outreach and local hire efforts should continue through the permitting process to assist in developing people's understanding of the Project and to generate local support. Community scoping meetings, organized tours of operating mines, and mining education workshops have all proved to be successful tools for developing understanding and support in other mining projects and are recommended strategies for Donlin Gold.
- Local Workforce Development and Training Strategy – Work should continue on implementing a strategy for training of the local workforce. This will assist in meeting obligations for local hire, generate support for the Project, and help to off-set workforce shortages seen on other projects. Strategies may include partnering with trade school programs, project "share" programs, participating in scholarship programs, and tracking the ongoing development and education of key individuals.

These work programs are estimated to cost approximately $10,000 to $15,000.

27.0 REFERENCES

27.1 Bibliography

AMEC, 2003: Prefeasibility Study – Status Report, Donlin Creek Project: unpublished report draft prepared for for Placer Dome Inc

Associated Mining Consultants, 2005: Donlin Creek Geophysical Survey, Alaska: unpublished report prepared for BGC, 17 June 2005

Buchanan, L.J., 2009: Donlin Creek, Alaska, Estimation of Potential Lode Ounces Gold: unpublished internal NovaGold memorandum, August 10, 2009, 17 p

Bundtzen, T.K., and Miller, M.L., 1997: Precious Metals Associated with Late Cretaceous-Early Tertiary Igneous Rocks of Southwestern Alaska: Econ. Geol., Monograph 9, pp 242–286

Cady, W.M., Wallace, R.E., Hoare, J.M., and Webber, E.J., 1955: The Central Kuskokwim Region, Alaska: USGS Professional Paper 268, 132 p

Canadian Dam Association (CDA), 2007: Dam Safety Guidelines: Edmonton, Alberta, 2007

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2003: Estimation of Mineral Resources and Mineral Reserves, Best Practice Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, November 23, 2003, http://www.cim.org/committees/estimation2003.pdf.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2010: CIM Standards for Mineral Resources and Mineral Reserves, Definitions and Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, November 2010, http://www.cim.org/UserFiles/File/CIM_DEFINITON_STANDARDS_Nov_2010.pdf

Canadian Securities Administrators (CSA), 2005: National Instrument 43-101, Standards of Disclosure for Mineral Projects, Canadian Securities Administrators.

Chamois, P., 2009: Donlin Creek Exploration Potential: unpublished report prepared by Scott Wilson Roscoe Postle Associates Inc. for NovaGold, September 2, 2009, 6 p.

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

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

Ebert, S.W., Miller, L., Petsel, S., Kowalczyk, P., Tucker, T.L., and Smith, M.T., 2000: Geology, Mineralization, and Exploration at the Donlin Creek Project, Southwestern Alaska: British Columbia and Yukon Chamber of Mines, v. Special 2, p. 99–114

Ebert S.W., Tosdal, R., Goldfarb, R., Dodd, S., Petsel, S., Mortensen, J., and Gabites, J., 2003a: The 25 Million Ounce Donlin Creek Gold Deposit, Southwest Alaska: A Possible Link Between Reduced Porphyry Au And Sub-Epithermal Au-As-Sb-Hg Mineralization: *in* Regional Geologic Framework and Deposit Specific Exploration Models For Intrusion Related Gold Mineralization, Yukon and Alaska, *ed*. S. Ebert: MDRU Special Publication 3, the Mineral Deposit Research Group, University of British Columbia

Ebert, S.W., Dodd, S., Miller, L., and Petsel, S., 2003b: The Donlin Creek Au-As-Sb-Hg Deposit, Southwestern Mineralization, Yukon and Alaska: *in* Regional Geologic Framework and Deposit Specific Exploration Models For Intrusion Related Gold Mineralization, Yukon and Alaska, *ed*. S. Ebert: MDRU Special Publication 3, the Mineral Deposit Research Group, University of British Columbia.

Ebert, S.W., Baker, T., and Spencer, R.J., 2003: Fluid Inclusion Studies at the Donlin Creek Gold Deposit, Alaska, Possible Evidence for Reduced Porphyry-Au To Sub Epithermal Transition: *in* Mineral Exploration and Sustainable Development, , Proceedings of the Seventh Biennial SGA Meeting August 24-28, 2003: Athens, Greece, p. 263–266

Francis, K., 2008: Donlin Creek Project, NI 43-101 Technical Report, Southwest Alaska, U.S.: unpublished technical report prepared for NovaGold Resources Inc., effective date 5 February 2008'

Francis, K., 2011: Donlin Creek – Geologic Framework, Resources and Exploration Upside: unpublished powerpoint presentation prepared for NovaGold Resources Inc., dated 21 September 2011

Goldfarb, R. J., Ayuso, R., Miller, M.L., Ebert, S.W., Marsh, E.E., Petsel, S.A., Miller, L.D., Bradley, D., Johnson, C., and McClelland, W., 2004: The Late Cretaceous Donlin Creek Gold Deposit, Southwestern Alaska: Controls on Epizonal Ore Formation: Economic Geology, v. 99, p. 643–671

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

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, report date March 2002.

Jutras, M., 2006: Donlin Creek Project – Exploratory Data Analysis – Drill Hole Orientations: unpublished internal memorandum for DCLLC, June 5 2006, 5 pp and figures

MacNeil, D., 2009: The Timing and Structural Evolution of the Donlin Creek Gold Deposit, Southwest Alaska: unpublished M.Sc. thesis, University of British Columbia

O'Dea, M., and Bartch, M., 1997: Structural Controls on Gold Mineralization at The Donlin Creek Deposit, Southwest Alaska: unpublished report prepared by Etheridge Henley Williams consultants for Placer Dome Exploration Inc., September 1997

Pacific Rim Geological Consulting Inc., 2004: Assessment of Calcium Carbonate Resource Potential near Donlin Creek Project, Iditarod A-5 Quadrangle, Alaska: Final report prepared for Placer Dome U.S., dated 24 September 2004

Piekenbrock, J.R., and Petsel, S.A., 2003: Geology and Interpretation of the Donlin Creek Gold Deposit: NovaGold internal report, April 2003, 58 pp with appendices

Szumigala, D.J., Dodd, S.P., and Arribas, A., Jr., 2000: Geology and gold mineralization of the Donlin Creek prospects, southwestern Alaska: *in* Short Notes on Alaska Geology, Professional Report 119, *ed*. M.A. Wiltse: State of Alaska Department of Natural Resources Division of Geological and Geophysical Surveys, pp 91–115

Water Management Consultants, 2004: Summary of Field Activities: draft technical memorandum prepared for Placer Dome US, 21 May 2004

Appendix A

Process List

Process Design Criteria

Design Input Codes	
Source of Information	**Basis**
CL = Client	A = Assumed
AA = AMEC Americas Limited	C = Calculated
P = Published Data / Information	TBD = To Be Determined
R = Regulatory Requirement	TBC = To Be Confirmed
V = Vendor Data / Information	P = Published Data / Information
D = Drawings / Archived Data	T = Testwork Data
H = Hatch Ltd.	P(M) = Latest mine plan

Parameter	Unit	Value	Source	Basis
THROUGHPUT				
Life of Plant	years	27.5	CL	P (M)
Throughput (Design)				
Annual	t/a (t = tonnes)	19,527,500		
Daily	t/d	53,500		
Hourly	t/h	2,397		
Operating Days/Year	d	365		
Overall Plant Availability	%	93.0	CL	
Recovery				
Au recovery (overall)	%	89.83		C
Au per day	kg	100.3		C
Au per year	kg	36,592		C
Au per day	oz	3,224		C
Au per year	oz	1,176,581		C
RUN-OF-MINE ORE				
Moisture Content	%wt	1.8	AA	
Run-of-Mine Top Size	mm	1,200	P	
Specific Gravity	sg	2.68		T
Broken Ore Bulk Density	kg/m^3	1,600	P	
Nominal Feed Grade				
Au	g/t	2.086	CL	P (M)
Ag	g/t	1.65	CL	P (M)
As	g/t	2,346	CL	P (M)
S (total)	%	1.06	CL	P (M)
% Sulphur as Sulphide	%	100	CL	P (M)
CO_2	%	2.393	CL	P (M)
Hg	g/t	2.4	CL	P (M)
CRUSHING & COARSE ORE STORAGE				
Crushing				
Run-of-Mine Top Size	mm	1,200		A
Feed F_{80}	mm	635		A
Broken Ore Bulk Density	kg/m^3	1,600	P	
Ore Abrasion Index	-	0.133	AA	C

Parameter	Unit	Value	Source	Basis
Minnovex Crushing Work Index	-	14.7		T
Availability	%	65.0	CL	
Throughput (design)				
Annual	t/a	19,527,500		
Daily	t/d	53,500		C
Hourly (operating)	t/h	3,430		C
Haul Truck Capacity (797B Cat)	t	345	AA	
Crusher Type	-	gyratory	AA	
No. of Units	-	1	CL	
Crusher Size	in x in	60x89	V	
Discharge Top Size	mm	160	P	
Discharge P_{80}	mm	125	P	
Discharge Feeder Type	-	apron	AA	
Coarse Ore Storage				
No. of Storage Units	-	1	AA	
Type of Storage Unit	stockpile, bin	covered stockpile	CL	
Stacking Conveyor Type	fixed, stacker	fixed	AA	
Live Storage	h	16	AA	
Live Storage	t	38,000		C
Total Capacity	t	174,000	AA	
Coarse Ore Reclaim				
No. of Draw Points	-	4	AA	
No. of Discharge Feeders	-	4	AA	
Discharge Feeder Type	-	Apron	CL	
GRINDING				
Feed Top Size	mm	160	P	
F_{80}	mm	122	P	
Ore Specific Gravity	sg	2.68		T
Broken Ore Bulk Density	kg/m^3	1,600	P	
Throughput (Design)				
Annual	t/a	19,527,500	CL	
Daily	t/d	53,500		C
Hourly	t/ op hour	2,397		C
Circuit Availability	%	93.0	AA	
Circuit Sizing Basis	%	93.0	AA	
Operating Days/Year	d	365	AA	
MinnovEX Grinding Parameters				
Crusher Index – Average	dimensionless	14.7		T
SPI – Average	min	89.7		T
Bond Work Index – Average	kWh/t	15.4		T
JKTech Ore Breakage Test Results				
A	-	75.3		T
B	-	0.58		T
A x b	-	42.3		T
t_a	-	0.71		T
DWI	-	6.6		T

Parameter	Unit	Value	Source	Basis
Ore Abrasion Index	-	0.133	AA	C
SAG Ball Transfer Size, T_{80}	µm	1,700	AA	
Primary Ball Mill Circuit Discharge, P_{80}	µm	142	CL	T
Secondary Ball Mill Circuit Discharge, P_{80}	µm	50	CL	T
SAG Mill Grinding				
Circuit Configuration	open, closed	closed	AA	
No. of Units	-	1	AA	
Mill Type	-	SAG	AA	
F_{80}	mm	122	P	
Throughput				
Nominal Fresh Feed	t/h	2,397		C
Discharge				
P_{80}	µm	1,700	AA	
Slurry Density	%wt (solids)	70	AA	
Mill Dimensions	ft dia x ft EGL	38 x 24.8	AA	P
Mill Motor Size	MW	20	AA	
Mill Volumetric Steel Loading				
Nominal	%	12	AA	
Volumetric Loading	%	29	AA	
Critical Speed				
Process Maximum	% of	75	AA	
Grinding Media				
Type	ball, rod, none	ball, steel		
Size	mm	125		
Media Consumption	kg/t	0.289	AA	C
Mill Liners				
Liner Material	steel, rubber	steel		
Consumption	kg/t	0.023	AA	C
Discharge Sizing Method 1	-	trommel	AA	
Slot Size	mm	12.7	AA	
Discharge Sizing Method 2	-	vibrating screen	AA	
Slot Size	mm	12.7	AA	
Pebble Crushing				
No. of Units	-	2	AA	
Top Size	mm	77	AA	
Ore Abrasion Index	-	0.133	AA	C
Crushing Work Index	-	18.4	AA	
Uplift for Pebble Crushing	-	1.25	AA	
Circuit Availability	%	85.0	AA	
Circuit Sizing Basis	%	93.0	AA	
Capacity (Design)				
Circulating Load	% of mill feed	25	AA	
Feed Bin Capacity	t	500		C
Crusher Type	cone	cone	AA	
Discharge Top Size	mm	25	AA	
Discharge P_{80}	mm	11	AA	

Parameter	Unit	Value	Source	Basis
Primary Ball Mill Grinding				
Circuit Configuration	-	closed	AA	
No. of Units	-	1	AA	
Mill Type	-	ball – overflow	AA	
Feed F_{80}	µm	1,700	AA	
Throughput				
Nominal	t/h	2,397		C
Discharge				
Slurry Density	%wt (solids)	70	AA	
Ball Mill Circulating Load	%	210	AA	
Mill Dimensions	ft dia x ft EGL	26 x 42.5	AA	
Mill Motor Size	MW	18	AA	
Mill Volumetric Steel Loading				
Nominal	%	29	AA	
Volumetric Loading	%	45	AA	
Critical Speed				
Nominal	%	75	AA	
Grinding Media				
Type	ball, rod, none	ball, high chrome	AA	C
Size	mm	63.5	AA	
Media Consumption	kg/t	0.221	AA	C
Mill Liners				
Liner Material	steel, rubber	steel	AA	
Consumption	kg/t	0.023	AA	C
Scats Rejection	-	trommel	AA	
Slot Cut Size	mm	25		
Discharge Sizing Method	-	cyclones	AA	
Cyclone Feed				
Density	%wt (solids)	56.4		C
Cyclone Overflow				
Density	%wt (solids)	40		C
Size (P_{80})	µm	121	P	
OVERALL FLOTATION				
Circuit Availability	%	98.0	AA	
Circuit Sizing Basis	%	93.0	AA	
Feed	t/h	2,397		C
Feed Sulphur Grade, Design Average	%	1.060		C
Recovery				
Mass	% of mill feed	15.2	CL	C
Gold	% of mill feed	93.0	AA	C
Sulphur	% of mill feed	97.6	CL	C
Grade				
Sulphur	%	7.0		T
Total Retention Time (Primary and Secondary Rougher)	min	114		T

Parameter	Unit	Value	Source	Basis
Reagents				
Total PAX Addition	g/t	200		T
Total F549 Addition	g/t	5		P
Total MIBC Addition	g/t	95		P
Total Copper Sulphate Addition	g/t	100		
Total Dispersant (Cytec E40)	g/t	50		T
Total Soda Ash	g/t	50		T
Flotation pH	-	5 to 7		T
Conditioning Tanks				
Feed Density	%wt (solids)	40		C
Tank 1 Residence Time	min	3	AA	
Tank 2 Residence Time	min	3		C
No. of Tanks	-	2	AA	
Primary Rougher Flotation				
Feed				
Feed Density	%wt (solids)	30		C
Specific Gravity (solids)	sg	2.7		C
Concentrate Recovery Mass	% of mill feed	5.28	CL	C
Specific Gravity (solids)	sg	3.54	H	C
Density	%wt (solids)	17.68	H	C
Flotation Cells				
Residence Time	min	57	AA	T
Secondary Ball Mill Grinding				
Circuit Configuration	-	closed	AA	
No. of Units	-	1	AA	
Mill Type	-	ball – overflow	AA	
Feed F_{80}	µm	142	AA	
Throughput (Circuit) – Nominal	t/h	2,270		C
Mill Discharge – Slurry Density	%wt (solids)	72	AA	
Ball Mill Circulating Load	%	210	AA	
Mill Dimensions	ft dia x ft EGL	26 x 42.5	AA	
Mill Motor Size	MW	18	AA	
Mill Volumetric Steel Loading – Nominal	%	30	AA	
Volumetric Loading	%	45	AA	
Critical Speed – Nominal	% of	75	AA	
Grinding Media				
Type	ball, rod, none	ball, high chrome	AA	
Size	mm	37	AA	
Media Consumption	kg/t	0.35	CL	P
Mill Liners				
Liner Material	steel, rubber	steel	AA	
Consumption	kg/t	0.025	AA	C
Discharge Sizing Method		cyclones	AA	
Cyclone Feed – Density	%wt (solids)	48.5	H	C
Cyclone Overflow – Density	%wt (solids)	28.7	H	C
Size (P_{80})	µm	50	CL	T

Parameter	Unit	Value	Source	Basis
Secondary Rougher Flotation				
Feed Density	%wt (solids)	27		T
Concentrate Recovery Mass	% of mill feed	14.88	CL	C
Specific Gravity (solids)	sg	2.80	H	C
Density	%wt (solids)	15.9	H	C
Flotation Cells				
Residence Time	min	57	AA	
Cleaner Flotation				
Feed				
Feed Density	%wt (solids)	15.8	H	C
Concentrate				
Recovery Mass	% of mill feed	9.97	CL	C
Specific Gravity (solids)	sg	2.80	H	C
Density	%wt (solids)	11.3	H	C
Flotation Cells				
Residence Time	min	100		
Cleaner Scavenger Flotation				
Feed Density	%wt (solids)	26.4	H	C
Concentrate Recovery Mass	% of mill feed	3.76	CL	C
Specific Gravity (solids)	sg	2.79	H	C
Density	%wt (solids)	14.8	H	C
Flotation Cells				
Residence Time	min	150		C
CONCENTRATE THICKENING				
Number of Thickener Stages	-	1	CL	
Solids SG	sg	3.1		T
Feed Slurry Density (with auto dilution)	%wt (solids)	7		T
Solids Loading	$t/h/m^2$	0.27		T
Underflow Density	%wt (solids)	46	V	T
Flocculant Dosage	g/t Conc	70	V	T
CONCENTRATE STORAGE				
Feed Density	%wt (solids)	46		T
Circuit Live Capacity	h	36	CL	
ACIDULATION				
Type of Circuit	-	two tanks in series		
Operating Density	%wt (solids)	16	C	
Circuit Capacity	h	5	C	
CCD CHLORIDE WASH				
Assumed Feed Chloride (average)	ppm	30		C
Overall Wash Efficiency	%	>80		C
POX Feed Chloride (max)	ppm	15		C
Number of Thickener Stages	-	3	CL	
Solids SG	sg	3.1		C
Feed Slurry Density (with auto dilution)	%wt (solids)	7		T
Solids Loading	$t/h/m^2$	0.33		T
Underflow Density	%wt (solids)	48		T

Parameter	Unit	Value	Source	Basis
Flocculant Dosage No. 1 Thickener	g/t flot conc	250	V	T
Flocculant Dosage No. 2 Thickener	g/t conc	62.5	AA	A
Flocculant Dosage No. 3 Thickener	g/t conc	62.5	AA	A
FLOTATION TAILINGS THICKENING				
Number of Thickener Stages	-	1	CL	
Solids SG	sg	2.8	H	C
Feed Slurry Density (before auto dilution)	%wt (solids)	30	H	C
Feed Slurry Density (with auto dilution)	%wt (solids)	10		T
Solids Loading	$t/h/m^2$	0.28		T
Underflow Density	%wt (solids)	55	V	T
Flocculant Dosage	g/t tailings	40	V	T
POX CCD WASH				
Target Wash Efficiency	%	≥97	CL	
Wash Water Ratio	-	4.0	AA	
Number of Thickener Stages	-	4	CL	
Type of Thickener	-	High Rate	AA	
Solids SG	sg	2.75	H	C
Feed Slurry Density (with auto dilution)	%wt (solids)	7		T
Solids Loading	$t/h/m^2$	0.18		T
Slurry temp	°C	50		A
Underflow Density	%wt (solids)	41		T
Flocculant Dosage No. 1 Thickener	g/t POX Disch	250		T
Flocculant Dosage No. 2 Thickener	g/t POX Disch	62.5	AA	A
Flocculant Dosage No. 3 Thickener	g/t POX Disch	62.5	AA	A
Flocculant Dosage No. 4 Thickener	g/t POX Disch	62.5	AA	A
POX CCD CLARIFIER				
Type of Circuit	-	single stage hopper clarifier	AA	
Feed Solids	mg/L	380		T
Overflow Solids	mg/L	<50	AA	A
Flocculant Dosage	mg/L	2	AA	A
Underflow Density	% wt (solids)	>5	AA	A
NEUTRALIZATION (CCD O/F)				
Temperature (combined)	°C	≥55		T
Mix Tank Stages				
Tailings Neutralization	-	4		T
Lime Neutralization	-	1		T
Tailings Neutralization Retention	h	6		T
Aeration	m^3/min/ 1,000 m^3 tank	5	AA	
Lime Neutralization Retention	h	1		T
Lime Dosage, average	kg/t mill feed	0.1	CL	T
FT Neutralization Tanks				
Slurry pH (tailings neutralization)	-	1 - 6		T
Slurry pH (lime neutralization)	-	7		T
NEUTRALIZATION (CIL CIRCUIT FEED)				
Lime Neutralization Tanks				
Feed Density	%wt (solids)	40	CL	

Parameter	Unit	Value	Source	Basis
Residence Time, Circuit	h	8	AA	
Oxidant	-	oxygen	AA	
Slurry pH				
Tank No. 1 Feed pH	-	2		C
Tank No. 2 Discharge pH	-	9		C
Lime Consumption	kg/t of conc	5.9		T
CONCENTRATE LEACHING				
Type of Circuit	-	CIL		T
Gold Extraction	%	96.6	H	
Feed Density	%wt (solids)	35	AA	
Residence Time, Circuit	h	24		C
Oxidant	-	oxygen	CL	
Total Oxygen Addition to CIL	t/h	0.208	H	
Slurry pH	-	11	AA	
NaCN Consumption	kg/t (feed)	0.8	AA	T
NaCN Concentration (CIL Tank #2)	g/l	0.2	AA	
Carbon Concentration	g/l	15		C
Total Carbon Inventory (in CIL)	t	290		V
Carbon Gold Loading	g/t	4,365		C
Loaded Carbon Generated	t/d	23	AA	C
Carbon Advance Schedule	h/d per stage	2.0	AA	C
Number of Strips per Day	-	3	AA	
Carbon Batch Size Recommended	t of carbon	12	AA	
Stripped Carbon	g/t	100	AA	
Carbon Gold Adsorption Efficiency	%	99.5		T
Carbon Loss to CIL Tailings	% of daily use	1	AA	
Loss Caught on Safety Screen	% of loss	80	AA	
Recovered Carbon Moisture	%	50	AA	
Carbon Collection Tank Discharge Solids	%	10	AA	
Carbon				
Carbon Size	mesh	6 x 16	AA	
Carbon Screening Aperture				
Interstage	mesh	24		
Safety Screen	mesh	28		
CARBON ELUTION				
Stripping Method	-	Pressure Zadra	AA	
Number of Strip Vessels	-	2		
Carbon Batch Size	t	12.0		C
Loaded Carbon Mmoisture	%	50	AA	
Vessel Batch Size (Design)	t	12	AA	
No of Batches to Strip per Day	-	3	AA	
Number of Strips per Week	-	21		C
Acid Washing				
Wash Duration	h	3.5	AA	
Temperature	°C	ambient	AA	
Number of Tanks	-	2	AA	

Parameter	Unit	Value	Source	Basis
Acid Wash Solution	-	HNO_3	AA	
Acid Concentration	%	3	AA	
Acid Recirculation	h	1.5	AA	
Solution Velocity	BV/h	1.5	AA	
Pressure Stripping				
Strip Total Cycle Duration	h	9	AA	
Temperature	°C	145	AA	
Solution Strip Volume	BV	16	AA	
Strip Duration	h	8	AA	
Strip Solution Volume	m^3	384	AA	
Stripping Efficiency	%	97.7	AA	
Strip Solution, NaCN Conc.	%	0.2	AA	
Strip Solution, NaOH Conc.	%	1.0	AA	
Rinse Duration	h	1	AA	
Capacity of Tanks	t of carbon	12	AA	
Solution Velocity	BV/h	2	AA	
CARBON REGENERATION				
Kiln Type	-	indirect heated rotary kiln	AA	
Method of Heating	-	electric	AA	
No. of Kilns	-	1	AA	
Maximum Capacity/Unit	t/h	1.5	AA	
Operating Time/Batch (design)	h	8.0	AA	
Regeneration Temperature – Normal	°C	750	AA	
Quench Tank Target Temperature	°C	60		
Screen Fines Tank Discharge Solids Density	%	5		
Recovery from Filter Press	%	98		
Filter Cake Moisture Content	%	35		
Off-gas Scrubbing System Components	-	Wet scrubber	CL	
	-	Venturi scrubber	CL	
	-	Wet gas condenser	CL	
	-	Coalescer	CL	
	-	Carbon columns	CL	
New Carbon Sizing				
Screen Aperture	Mesh	20	AA	
REFINING				
Electrowinning				
Electrowinning Efficiency per Pass	%	98	AA	
Cell Voltage (Rectifier)	V (DC)	0 to 9	V	
Cell Solution Temperature	°C	60 to 80	AA	
Current Density	A/m^2	0.95	AA	
No. of Units	-	4	AA	
Feed Solution Flow/Unit	m^3/h	24.0	AA	
Target Barren Solution Tenor	g/t	2	AA	
Off-gas Scrubbing System Components	-	Electrowinning scrubber	CL	
	-	Caustic scrubber	CL	
	-	Venturi scrubber	CL	

Parameter	Unit	Value	Source	Basis
	-	Coalescer	CL	
	-	Carbon columns	CL	
Filter Presses				
Electrowinning Sludge				
Type of Filters	-	Pressure	AA	
No. of Units	-	2	AA	
Precipitate Drying/Rretort				
Operating Temperature	°C	600	AA	
Residence Time	h	16	AA	
Smelting Furnace				
Furnace Type	-	Induction		
Off gas scrubbing system components		Dust cyclone	CL	
		Carbon columns	CL	
CYANIDE DESTRUCTION				
Feed Density	%wt (solids)	31%	H	C
Residence Time, Circuit	h	1		T
Number of Tanks	-	1	AA	
Oxidant	-	SO_2/Air	AA	
Slurry pH	-	8.5		T
Lime Consumption	kg/t CIL Feed	0.194		T
CN(T) Feed to CN Destruction	mg/L	100	AA	
Residual CN(T)	mg/L	<=10	CL	
SO_2 Addition Rate	g/t of CIL Feed	880		C
Cu^{++} Requirement	g/t of solution	10		T
TAILINGS				
Circuit Availability	%	100.0	AA	
Circuit Sizing Basis	%	93.0	AA	
Capacity – Normal	t/h	2,510	H	C
Specific Gravity (solids)	sg	2.80	H	C
P_{80}	µm	50	AA	
Slurry Density	%wt (solids)	30	H	C
Slurry Specific Gravity	sg	1.250	H	C
Number of Tailings Pipelines		1	CL	
REAGENTS HANDLING				
PAX				
Usage Rate	g/t mill feed	200		C
F549				
Usage Rate	g/t mill feed	5		C
MIBC				
Usage Rate	g/t mill feed	95		C
$CuSO_4.5H_2O$ (Flotation & Cyanide Destruction)				
Usage Rate (Flotation)	g/t mill feed	100		C
Usage Rate (CN Destruction)	g/t mill feed	11.1		C
Dispersant (Cytec E40)				
Usage Rate	g/t mill feed	50		C

Parameter	Unit	Value	Source	Basis
Soda Ash				
Usage Rate	g/t mill feed	50		C
Flocculant 1 (POX CCD)				
Usage Rate	g/t mill feed	65		C
Flocculant 2 (Flotation Tails)				
Usage Rate	g/t mill feed	36.2		C
Flocculant 2 (Flotation Concentrate)				
Usage Rate	g/t mill feed	10.9		C
Flocculant 3 (CCD)				
Usage Rate	g/t mill feed	58		C
NaCN				
Usage Rate	kg/t mill feed	0.12		C
Lime				
Usage Rate				
CIL Neutralization	g/t solids CIL feed	5,900		C
CN Destruction	g/t solids CIL feed	194		T
Flotation Tailings Neutralization, Average	g/t mill feed	100	CL	T
Sulphur Dioxide				
Usage Rate	g/t CIL feed solids	880		C
$CuSO_4.5H_2O$ (Cyanide Destruction)				
Usage Rate	g/t CIL feed solids	13.2		C
HNO_3				
Usage Rate	g/t CIL feed solids	31		C
NaOH				
Usage Rate	g/t CIL feed solids	14		C
UNR 829				
Usage Rate	g/t CIL feed solids	7.3		C
Borax	kg/y	54,900		C
Nitre	kg/y	43,910		C
Silica	kg/y	43,910		C