Donlin Creek Project
NI 43-101 Technical Report
Southwest Alaska, U.S.

Effective Date February 5, 2008

Prepared for:
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Vancouver, B.C.

Prepared by:
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CERTIFICATE OF AUTHOR

I, Kevin A. Francis, do hereby certify that:

1. I am a Professional Geologist, and Resource Manager of:

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2. I graduated from the University of Colorado, Boulder, Colorado, with a Master of Science degree in Geology from the University of Colorado in 1987.

3. I am registered as a Professional Geologist in the state of Utah (5311425-2250).

4. I have practiced my profession continuously since 1987 and have been involved in: mineral exploration for copper, gold and silver in the United States, Canada, Mexico, Russia and Southeast Asia; exploration data evaluation, geological modeling and resource modeling of gold, copper, iron, platinum, palladium, manganese and industrial mineral deposits in the United States, Canada, Ukraine, Mexico, Peru, Chile, Brazil, and South Africa.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Sections 1 through to 23 inclusive of the technical report titled *Donlin Creek Property NI 43-101 Technical Report Southwest Alaska U.S.*, dated 5 February 2008 (the "Technical Report") relating to the Donlin Creek property. I visited the Donlin Creek Project on July 10, 2006.

7. I am not independent of the issuer but do qualify to assist in the preparation of this report as outlined in Section 5.3 (3) of National Instrument 43-101.

8. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

9. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

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

Dated this 22 of February, 2008.

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Kevin A. Francis, P.Geo.

CONTENTS

FIGURES

TABLES

APPENDICES

1.0 SUMMARY

NovaGold Resources Inc. (NovaGold) has prepared an updated Technical Report that meets the guidelines as outlined by Canadian National Instrument 43-101 for its Donlin Creek property, an early Tertiary age gold-arsenic-antimony-mercury hydrothermal system located in southwest Alaska. NovaGold and Barrick Gold Corporation (Barrick) have 50/50 ownership in the property through their joint ownership of a limited liability company, the Donlin Creek LLC, which will oversee all aspects of project development. Kevin Francis, Manager of Resources for NovaGold and a qualified person as defined by NI 43-101, has reviewed sampling procedures, examined quality assurance/quality control (QA/QC) practices/results, and reviewed Barrick's estimate of Mineral Resources which includes drilling information through October 3, 2007. In addition to those reviews, the author performed a site visit in July 2006 for the purpose of observing drilling and sampling procedures and to examine drill core and geologic logging practices and, as at the effective date of the report, confirms that there has been no material change in the scientific and technical information about the property since that site visit.

The Donlin Creek project is located near the of the Kuskokwim River about 25 km north of the village of Crooked Creek and approximately 70 km northeast of Aniak, a regional hub. The latitude and longitude of the deposit is approximately 62^{o} 01' N and 158^{o} 12' W.

The gold-bearing deposits have been combined into two resource areas: ACMA (containing ACMA proper, 400 Zone, Aurora and Akivik deposits) and Lewis (comprising North and South Lewis, Vortex, Rochelieu and Queen deposits). The project is serviced by commercial air services out of Anchorage and Aniak and by a 25 km long winter road from the town of Crooked Creek.

The property consists of 109 km^{2} (42 mi^{2}, 10,900 ha) of privately owned Native Alaskan land. Calista Corporation (Calista), a regional Native corporation, owns the subsurface rights, and The Kuskokwim Corporation (TKC), a village corporation, owns the surface rights. Placer Dome acquired a 20-year lease from Calista effective May 1, 1995. Annual property payments are US$200,000 through the end of feasibility and increasing to US$500,000 per annum once a feasibility study is completed. Calista holds a retained net royalty of 1.5% until payback of capital, increasing to 4.5% thereafter. Placer Dome exercised its back-in right and assumed management of the continued development of the Donlin Creek project in 2003. Barrick acquired Placer Dome in February 2006. Barrick and NovaGold formed a joint venture to develop the property pursuant to a joint venture agreement dated November 13, 2002. The NovaGold and Barrick joint venture was converted to a limited liability company, the Donlin Creek LLC, with NovaGold and Barrick each holding a 50/50 interest therein.

Upon submission of a feasibility study, Calista retains a 90-day back-in right to participate in the project at a level of 5% to 15% by committing to contribute its share of capital. Their share would be divided pro rata from Barrick and NovaGold.

The Donlin Creek project geology consists of flysch sequence sedimentary units of the Cretaceous Kuskokwim Group intruded by Late Cretaceous to early Tertiary felsic intrusive rocks. The sediments consist of interbedded greywacke, shale and siltstone. Greywacke is dominant (Lewis resource area), but shale-rich areas also occur (ACMA resource area). The overall bedding strikes NW and dips 10° to 50° southwest. The intrusive units consist of porphyritic rhyodacite and rhyolite and lesser mafic dykes and sills. Sills are common in the ACMA and southern Lewis areas, whereas dykes dominate in the North Lewis area. The dykes and sills range from a few meters to more than 60 m in width.

Mineralization is best developed in the felsic intrusive rocks, with lesser mineralization in sediments (principally in the greywacke units). It is structurally and lithologically controlled along NNE-trending extensional fault/fracture zones and best developed where those zones intersect favourable host lithologies such as the felsic intrusive dykes and sills and greywacke. Gold mineralization is associated with quartz, carbonate and sulphide (pyrite, arsenopyrite and stibnite) vein and veinlet networks (dominant) as well as disseminated in favourable host rocks typically adjacent to veins (subdominant). The gold occurs primarily in the lattice structure of arsenopyrite. Realgar, native arsenic and stibnite can be found generally associated with the higher-grade gold mineralization.

Two sets of similar protocols were used for the samples that formed the basis of the Lewis and ACMA mineral resource model. Prior to 2002, most of the samples from Placer Dome's work were processed in their own laboratory. NovaGold's samples were processed by Bondar-Clegg (now ALS Chemex), a commercial laboratory. The 2006 and 2007 samples were assayed at ALS Chemex. The results can be evaluated together because the Standard Reference Material (SRM), the blank material and the duplicate protocol were the same. The performance of each SRM was within acceptable limits and showed that the overall assay process was in control for the work done. Good reproducibility of the gold values is demonstrated. The blank sample program worked well and demonstrated negligible contamination in the assay process.

The database used to estimate the mineral resources consists of samples and geological information from 1,578 drill holes and trenches. Since the last technical report in January 2006, Barrick has drilled 359 holes totalling 135,362 m (444,101 ft) of geologically logged and assayed core within the resource area. Assay data transfer to the resource database was validated from electronic assay certificates.

Metallurgical testwork, under the direction of Barrick, appears to have been completed in sufficient detail to support a feasibility level study.

Gold is mainly carried by arsenopyrite.

Variation is observed in processing behaviour between intrusives and sediments, but less so between the geographical sources. Concentration by flotation is efficient, being 91 to 97% for intrusives and 82 to 89% for sediments. Generally, direct cyanidation yields less than 10% gold recovery; whereas oxidation of the sulphides prior to cyanidation yields recoveries exceeding 90% for intrusives and 78 to 89% for the sediments.

Accordingly, process testing has been directed towards development of the following conceptual flowsheet:

- concentration by flotation using nitrogen
- high pressure oxidation in an autoclave
- carbon-in-leach (CIL) cyanidation of the concentrate
- carbon strip and regeneration circuits
- gold electrowinning, and
- refining and production of doré bars

This processing concept incorporates proven commercial unit operations. No issues have been identified to date that might lead to economic performance of this sequence that would be substantially different from similar processes in commercial operation today.

Presently there is a 160-person exploration camp on site. The exploration drilling for 2008 has commenced and, as of the date of this report, two core rigs are operating. Exploration is focussed on step out drilling in the East ACMA area testing for additional fault/fracture zones in the favourable intrusive rock type.

There is no history of significant development or hard rock production. Minor placer gold production has taken place within the project area.

1.1 Conclusions and Recommendations

Donlin Creek is a large-scale gold deposit with a long history of successful exploration. Presently the deposit is open in all directions and is still the subject of an extensive exploration drilling program. The resource estimate prepared by Barrick has been completed using generally-accepted methodology.

The mineral resource estimates for the Donlin Creek project were calculated by Barrick. The estimates were made from 3-dimensional (3D) block models utilizing Vulcan™ mine planning software. Industry-accepted methods were used to create interpolation domains based on mineralized geology, and grade estimation based on inverse distance to a power cubed (ID3). Acceptable mineralized envelopes were defined through probability-assisted domaining. This method limited the waste intervals of the intrusive units at ACMA from diluting the grades in the mineralized regions and honoured the significant contribution of greywacke-hosted mineralization together with mineralized felsic intrusive units at Lewis. Extremely high gold assays were capped prior to compositing into 6 m lengths.

Reasonableness of grade interpolation was reviewed by visual inspection of sections and plans displaying block model grades, drill hole composites and geology. Good agreement was observed. Global and local bias checks in block models, using nearest-neighbour estimated values versus the ID3 gold grade estimate, found no evidence of bias. Change of support measurements relative to a 6 m by 6 m by 6 m selective mining unit (SMU) indicate the grade estimate is not smooth enough to be used for mine planning without prior adjustment.

The logic for mineral resource classification of ACMA and Lewis was consistent with the CIM definitions referred to in National Instrument 43-101 (NI 43-101). The indicated mineral resource category is supported by the present drilling grids over the ACMA and Lewis deposits (nominal 25 m to 35 m). The measured mineral resource category is supported only in blocks pierced by exploration drill holes. Inferred mineralization is limited to a reasonable expectation of mining by a conceptual open pit shell using a metal price of US$650 per ounce of gold and recent estimates of mining, geotechnical and metallurgical parameters.

The mineralization of the Donlin Creek project effective date as of February 5, 2008, is classified as measured, indicated and inferred mineral resources. The classified mineral resources are shown in Table 1-1. The mineral resource has been constrained within a conceptual pit based on US$650 per ounce of gold and using recent estimates of mining, geotechnical and metallurgical parameters. Barrick's Technical Services Evaluations Group estimated a variable net smelter return (NSR) cut-off grade based on recent estimates of mining costs, processing costs (dependent upon sulphur content), selling costs and royalties, rather than gold grade alone. The NSR cut-off equates to approximately 0.8 g/t gold at the average estimated sulphur grade of 1.12%.

The mineral resource estimates for Donlin Creek project show an increase in resources over NovaGold's September 2006 mineral resource estimates. This increase is the result of additional drilling completed during 2006 and 2007.

Table 1-1: Donlin Creek Project Mineral Resource Summary(1)(2)(3)(4)
Effective Date February 5, 2008

	Tonnes (M)	Au (g/t)	Contained Au (Million oz)
US$0.01 NSR/t Cut-off			
Measured Mineral Resource	4.3	2.73	0.38
Indicated Mineral Resource	367.4	2.46	29.0
Measured + Indicated Mineral Resources	**371.7**	**2.46**	**29.38**
Inferred Mineral Resource	**46.5**	**2.31**	**3.46**

(1) Mineral resources that are not mineral reserves do not have demonstrated economic viability
(2) Rounding differences may occur.
(3) Resources are constrained within a Lerchs-Grossman (LG) open-pit shell using the long-term metal price assumption of US$650/oz of gold. Assumptions for the LG shell included pit slopes variable by sector and pit area: mining cost is variable with depth, averaging US$1.57/t mined; process cost is calculated as the percent sulfur grade x US$2.09 + US$10.91; general and administrative costs, gold selling cost and sustaining capital cost are reflected on a per tonne basis. Average sulphur content is 1.12%. Based on metallurgical testing, gold recovery is assumed to be 89.5%. Blocks with a cost margin of US$0.01/t or higher above the variable cut-off were reported.
(4) Waste blocks within the open-pit shell surrounded by blocks above cut-off are included in resource estimate. Blocks above cut-off within the open-pit shell surrounded by blocks of waste are excluded from resource estimate.

NovaGold recommends that Donlin Creek LLC continue to advance the project toward a feasibility study.

2.0 INTRODUCTION

NovaGold Resources Inc. (NovaGold) is completing this Technical Report to update the resource estimate for the Donlin Creek project in Alaska. Barrick has provided NovaGold with a new resource estimate containing drilling data through October 2007. Resource estimates were previously reported in Technical Reports dated February and March 2002 (Juras, 2002 and Juras and Hodgson, 2002), January 2006 (Dodd, 2006) and September 2006 (Dodd et al., 2006). Kevin Francis, P.Geo., an employee of NovaGold, serves as the non-independent Qualified Person (QP) responsible for the preparation of this Technical Report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1 (the Technical Report). A non-independent qualified person is permitted under NI 43-101 because the Donlin Creek project is a joint venture between a producer (Barrick) and a non-producer (NovaGold).

NovaGold relied on scientific and technical information prepared by or under the supervision of a qualified person of Barrick relating to the preparation of resource estimates in this Technical Report.

The work entailed review of pertinent geological data in sufficient detail to prepare the Technical Report.

2.1 Qualified Person

Kevin Francis, Manager of Resources and an employee of NovaGold Resources Inc., served as the QP responsible for the preparation of the report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1 (the Technical Report).

The QP visited the Donlin Creek project on July 10, 2006 during which time additional background data were reviewed and, as at the effective date of the report, confirms that there has been no material change in the scientific and technical information about the property since that site visit.

2.2 Previous Technical Studies

Technical reports completed for the project and on file at www.sedar.com include:

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

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 Creek Gold Project Alaska, USA, unpublished NI43-101Fi Technical Report to NovaGold Resources Inc. by SRK Consulting (US), Inc., effective date September 20, 2006

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

Table 2-1 relates the sections as shown in the contents page of this report to the Prescribed Items Contents Page of NI 43-101. The main differences are that Item 25 “Additional Requirements for Technical Reports on Development Properties and Production Properties” is incorporated into the main body of the report, immediately following Item 18, “Other Relevant Data and Information”, and that all illustrations (Item 26, “Illustrations”) are included in the body of the report immediately following the text citation of the appropriate illustration.

Table 2-1: Contents Page Headings in Relation to NI 43-101 Prescribed Items—Contents

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

3.0 RELIANCE ON OTHER EXPERTS

Placer Dome assumed project management in 2003, which was subsequently transferred to Barrick as a result of their takeover of Placer Dome in February 2006. NovaGold relies upon Barrick to have collected and analysed drill hole samples, directed metallurgical testing and constructed the resource estimate using accepted industry practice by, or under the supervision of, qualified persons.

3.1 Mineral Tenure

The QP has not reviewed the mineral tenure, nor independently verified the legal status or ownership of the project area or underlying property agreements. The QP has relied upon Barrick for this information.

3.2 Surface Rights, Access and Permitting

The QP has relied upon Barrick for information regarding Surface Rights, Road Access and Permits, including the status of the granting of surface rights for land designated for mining, milling, dumps and tailings impoundments.

3.3 Environmental

The QP has relied relied upon Barrick to provide the environmental status for the project.

4.0 PROPERTY DESCRIPTION AND LOCATION

Donlin Creek is located in southwest Alaska in the United States of America, approximately 70 km (44 mi) northeast of Aniak, a regional hub (see Figure 4-1). The property consists of 109 km^2 (42 mi^2, 10,858 hectares) of privately owned Native Alaskan land. Calista Corporation (Calista), a regional Native corporation, owns the subsurface rights, and The Kuskokwim Corporation (TKC), a village corporation, owns the surface rights. The resource areas are within T. 23 N., R. 49. W. (see Figure 4-2), Seward Meridian, Kuskokwim and Mt. McKinley Recording Districts, Crook Creek Mining District, Iditarod A-5 USGS 1:63,360 topography map. These areas consist of the ACMA and 400 Zone, Aurora and Akivik prospects (grouped as ACMA) and the Lewis, South Lewis, Vortex, Rochelieu and Queen prospects (grouped as Lewis) (see Figure 4-3).

4.1 Coordinate System

The Donlin Creek project uses Universal Transverse Mercator (UTM) Zone 4 (metres). The map datum is NAD83.

4.2 Mineral Tenure

The land status of the Donlin Creek area is shown in Figure 4-2. 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 according to a formula based on original village corporation enrolments.

The Donlin Creek exploration and mining lease currently includes a total of 42 contiguous sections leased from Calista (Table 4-1), which holds the subsurface (mineral) estate for Native-owned lands in the region. The leased land is believed to contain 10,858 hectares (26,830 acres). Title to all of these sections has been conveyed to Calista by the Federal Government. Calista owns the surface estate on 9 of these 42 sections. A separate Surface Use Agreement with TKC, which owns the surface estate of the remaining 33 sections, grants non-exclusive surface use rights to the Donlin Creek LLC. All of these sections have now been conveyed to Calista/TKC by the Federal Government. Figure 4-2 shows the lease block. Figure 4-3 shows the resource area and prospect names.

Table 4-1: Donlin Creek Land Leased from Calista

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

In addition to the leased land, Donlin Creek LLC holds 176 unpatented mineral claims (Table 4-2) comprising 8,968 hectares (22,160 acres) primarily surrounding the leased land in the Kuskokwim and Mt. McKinley recording districts. Of these, 32 claims are tentatively approved (T.A.) for conveyance from the Federal to State government subject to official surveying. The remaining 144 claims are state-selected (S.S.). These claims have not been legally surveyed. All claims are either 16.2 hectares (40 acres), 32.4 hectares (80 acres) or 64.8 hectares (160 acres) in size.

Table 4-2: Donlin Creek LLC Mineral Claims

ADL #	Claim Name	Status	Located/ T.A.'ed	Size in Hectares	Twp	Rng	Sec
578768	DNC # 1	T.A.	08/14/07	16.2	24N	47W	31
578769	DNC # 2	T.A.	08/14/07	16.2	24N	47W	31
578770	DNC # 3	T.A.	07/10/07	16.2	24N	48W	36
578771	DNC # 4	T.A.	07/10/07	16.2	24N	48W	36
578772	DNC # 5	T.A.	07/10/07	16.2	24N	48W	36
578773	DNC # 6	T.A.	07/10/07	16.2	24N	48W	36
578774	DNC # 7	T.A.	07/10/07	16.2	24N	48W	35
578775	DNC # 8	T.A.	07/10/07	16.2	24N	48W	35
578776	DNC # 9	T.A.	07/10/07	16.2	24N	48W	35
578777	DNC # 10	T.A.	07/10/07	16.2	24N	48W	35
578778	DNC # 11	T.A.	07/10/07	16.2	24N	48W	36
578779	DNC # 12	T.A.	07/10/07	16.2	24N	48W	36
578780	DNC # 13	T.A.	07/10/07	16.2	24N	48W	36
578781	DNC # 14	T.A.	07/10/07	16.2	24N	48W	36
578782	DNC # 15	T.A.	08/14/07	16.2	24N	47W	31
578783	DNC # 16	T.A.	08/14/07	16.2	24N	47W	31
578784	DNC # 17	T.A.	08/14/07	16.2	24N	47W	31
578785	DNC # 18	T.A.	08/14/07	16.2	24N	47W	31
578786	DNC # 19	T.A.	07/10/07	16.2	24N	48W	36
578787	DNC # 20	T.A.	07/10/07	16.2	24N	48W	36
578788	DNC # 21	T.A.	07/10/07	16.2	24N	48W	36

ADL #	Claim Name	Status	Located/ T.A.'ed	Size in Hectares	Twp	Rng	Sec
578789	DNC # 22	T.A.	07/10/07	16.2	24N	48W	36
578790	DNC # 23	T.A.	07/10/07	16.2	24N	48W	35
578791	DNC # 24	T.A.	07/10/07	16.2	24N	48W	35
578792	DNC # 25	T.A.	07/10/07	16.2	24N	48W	35
578793	DNC # 26	T.A.	07/10/07	16.2	24N	48W	35
578794	DNC # 27	T.A.	07/10/07	16.2	24N	48W	36
578795	DNC # 28	T.A.	07/10/07	16.2	24N	48W	36
578796	DNC # 29	T.A.	07/10/07	16.2	24N	48W	36
578797	DNC # 30	T.A.	07/10/07	16.2	24N	48W	36
578798	DNC # 31	T.A.	08/14/07	16.2	24N	47W	31
578799	DNC # 32	T.A.	08/14/07	16.2	24N	47W	31
578800	DNC # 33	S.S.	09/27/96	16.2	23N	48W	1
578801	DNC # 34	S.S.	09/27/96	16.2	23N	48W	1
578802	DNC # 35	S.S.	09/27/96	16.2	23N	48W	2
578803	DNC # 36	S.S.	09/27/96	16.2	23N	48W	2
578804	DNC # 37	S.S.	09/27/96	16.2	23N	48W	2
578805	DNC # 38	S.S.	09/27/96	16.2	23N	48W	2
578806	DNC # 39	S.S.	09/27/96	16.2	23N	48W	2
578807	DNC # 40	S.S.	09/27/96	16.2	23N	48W	2
578808	DNC # 41	S.S.	09/27/96	16.2	23N	48W	1
578809	DNC # 42	S.S.	09/27/96	16.2	23N	48W	1
578810	DNC # 43	S.S.	09/27/96	16.2	23N	48W	2
578811	DNC # 44	S.S.	09/27/96	16.2	23N	48W	2
578812	DNC # 45	S.S.	09/27/96	16.2	23N	48W	2
578813	DNC # 46	S.S.	09/27/96	16.2	23N	48W	2
578814	DNC # 47	S.S.	09/27/96	16.2	23N	48W	2
578815	DNC # 48	S.S.	09/27/96	16.2	23N	48W	2
578816	DNC # 49	S.S.	09/27/96	16.2	23N	48W	11
578817	DNC # 50	S.S.	09/27/96	16.2	23N	48W	11
644952	GROUSE 1	S.S.	05/04/04	64.8	23N	50W	35
644952	GROUSE 1	S.S.	05/04/04	64.8	23N	50W	35
644953	GROUSE 2	S.S.	05/04/04	64.8	23N	50W	35
644954	GROUSE 3	S.S.	05/04/04	64.8	23N	50W	36
644955	GROUSE 4	S.S.	05/04/04	64.8	23N	50W	36
644956	GROUSE 5	S.S.	05/04/04	64.8	23N	49W	31
644957	GROUSE 6	S.S.	05/04/04	64.8	23N	49W	31
644958	GROUSE 7	S.S.	05/04/04	64.8	23N	49W	32
644959	GROUSE 8	S.S.	05/03/04	64.8	23N	49W	32
644960	GROUSE 9	S.S.	05/04/04	64.8	23N	50W	35
644960	GROUSE 9	S.S.	05/04/04	64.8	23N	50W	35

ADL #	Claim Name	Status	Located/ T.A.'ed	Size in Hectares	Twp	Rng	Sec
644961	GROUSE 10	S.S.	05/04/04	64.8	23N	50W	35
644962	GROUSE 11	S.S.	05/04/04	64.8	23N	50W	36
644963	GROUSE 12	S.S.	05/04/04	64.8	23N	50W	36
644964	GROUSE 13	S.S.	05/04/04	64.8	23N	49W	31
644965	GROUSE 14	S.S.	05/04/04	64.8	23N	49W	31
644966	GROUSE 15	S.S.	05/04/04	64.8	23N	49W	32
644967	GROUSE 16	S.S.	05/03/04	64.8	23N	49W	32
644968	GROUSE 17	S.S.	05/04/04	64.8	23N	50W	26
644968	GROUSE 17	S.S.	05/04/04	64.8	23N	50W	26
644969	GROUSE 18	S.S.	05/04/04	64.8	23N	50W	26
644970	GROUSE 19	S.S.	05/04/04	64.8	23N	50W	25
644971	GROUSE 20	S.S.	05/04/04	64.8	23N	50W	25
644972	GROUSE 21	S.S.	05/04/04	64.8	23N	49W	30
644973	GROUSE 22	S.S.	05/04/04	64.8	23N	49W	30
644974	GROUSE 23	S.S.	05/04/04	64.8	23N	49W	29
644975	GROUSE 24	S.S.	05/03/04	64.8	23N	49W	29
644976	GROUSE 25	S.S.	05/04/04	64.8	23N	50W	26
644976	GROUSE 25	S.S.	05/04/04	64.8	23N	50W	26
644977	GROUSE 26	S.S.	05/04/04	64.8	23N	50W	26
644978	GROUSE 27	S.S.	05/04/04	64.8	23N	50W	25
644979	GROUSE 28	S.S.	05/04/04	64.8	23N	50W	25
644980	GROUSE 29	S.S.	05/04/04	64.8	23N	49W	30
644981	GROUSE 30	S.S.	05/04/04	64.8	23N	49W	30
644982	GROUSE 31	S.S.	05/04/04	64.8	23N	49W	29
644983	GROUSE 32	S.S.	05/03/04	64.8	23N	49W	29
644984	GROUSE 33	S.S.	05/04/04	64.8	23N	50W	23
644984	GROUSE 33	S.S.	05/04/04	64.8	23N	50W	23
644985	GROUSE 34	S.S.	05/04/04	64.8	23N	50W	23
644985	GROUSE 34	S.S.	05/04/04	64.8	23N	50W	23
644986	GROUSE 35	S.S.	05/04/04	64.8	23N	50W	24
644987	GROUSE 36	S.S.	05/04/04	64.8	23N	50W	24
644988	GROUSE 37	S.S.	05/04/04	64.8	23N	49W	19
644989	GROUSE 38	S.S.	05/04/04	64.8	23N	49W	19
644990	GROUSE 39	S.S.	05/04/04	64.8	23N	49W	20
644991	GROUSE 40	S.S.	05/03/04	64.8	23N	49W	20
644992	GROUSE 41	S.S.	05/04/04	64.8	23N	50W	23
644993	GROUSE 42	S.S.	05/04/04	64.8	23N	50W	23

ADL #	Claim Name	Status	Located/ T.A.'ed	Size in Hectares	Twp	Rng	Sec
644993	GROUSE 42	S.S.	05/04/04	64.8	23N	50W	23
644994	GROUSE 43	S.S.	05/04/04	64.8	23N	50W	24
644994	GROUSE 43	S.S.	05/04/04	64.8	23N	50W	24
644995	GROUSE 44	S.S.	05/04/04	64.8	23N	50W	24
644996	GROUSE 45	S.S.	05/04/04	64.8	23N	49W	19
644997	GROUSE 46	S.S.	05/04/04	64.8	23N	49W	19
644998	GROUSE 47	S.S.	05/04/04	64.8	23N	49W	20
644999	GROUSE 48	S.S.	05/03/04	64.8	23N	49W	20
645000	GROUSE 49	S.S.	05/04/04	64.8	23N	50W	14
645001	GROUSE 50	S.S.	05/04/04	64.8	23N	50W	14
645002	GROUSE 51	S.S.	05/04/04	64.8	23N	50W	13
645002	GROUSE 51	S.S.	05/04/04	64.8	23N	50W	13
645003	GROUSE 52	S.S.	05/04/04	64.8	23N	50W	13
645003	GROUSE 52	S.S.	05/04/04	64.8	23N	50W	13
645004	GROUSE 53	S.S.	05/04/04	64.8	23N	49W	18
645004	GROUSE 53	S.S.	05/04/04	64.8	23N	49W	18
645005	GROUSE 54	S.S.	05/04/04	64.8	23N	49W	18
645006	GROUSE 55	S.S.	05/04/04	64.8	23N	49W	17
645007	GROUSE 56	S.S.	05/03/04	64.8	23N	49W	17
645008	GROUSE 57	S.S.	05/03/04	64.8	23N	49W	16
645009	GROUSE 58	S.S.	05/03/04	64.8	23N	49W	16
645010	GROUSE 59	S.S.	05/04/04	64.8	23N	50W	14
645011	GROUSE 60	S.S.	05/04/04	64.8	23N	50W	14
645012	GROUSE 61	S.S.	05/04/04	64.8	23N	50W	13
645013	GROUSE 62	S.S.	05/04/04	64.8	23N	50W	13
645014	GROUSE 63	S.S.	05/04/04	64.8	23N	49W	18
645014	GROUSE 63	S.S.	05/04/04	64.8	23N	49W	18
645015	GROUSE 64	S.S.	05/04/04	64.8	23N	49W	18
645016	GROUSE 65	S.S.	05/04/04	64.8	23N	49W	17
645017	GROUSE 66	S.S.	05/03/04	64.8	23N	49W	17
645018	GROUSE 67	S.S.	05/03/04	64.8	23N	49W	16
645019	GROUSE 68	S.S.	05/03/04	64.8	23N	49W	16
645020	GROUSE 69	S.S.	05/04/04	64.8	23N	50W	11
645021	GROUSE 70	S.S.	05/04/04	64.8	23N	50W	11
645022	GROUSE 71	S.S.	05/04/04	64.8	23N	50W	12
645023	GROUSE 72	S.S.	05/04/04	64.8	23N	50W	12
645024	GROUSE 73	S.S.	05/04/04	64.8	23N	49W	7

ADL #	Claim Name	Status	Located/ T.A.'ed	Size in Hectares	Twp	Rng	Sec
645024	GROUSE 73	S.S.	05/04/04	64.8	23N	49W	7
645025	GROUSE 74	S.S.	05/04/04	64.8	23N	49W	7
645025	GROUSE 74	S.S.	05/04/04	64.8	23N	49W	7
645026	GROUSE 75	S.S.	05/04/04	64.8	23N	49W	8
645027	GROUSE 76	S.S.	05/03/04	64.8	23N	49W	8
645028	GROUSE 77	S.S.	05/03/04	64.8	23N	49W	9
645029	GROUSE 78	S.S.	05/03/04	64.8	23N	49W	9
645030	GROUSE 79	S.S.	05/04/04	64.8	23N	50W	11
645031	GROUSE 80	S.S.	05/04/04	64.8	23N	50W	11
645032	GROUSE 81	S.S.	05/04/04	64.8	23N	50W	12
645033	GROUSE 82	S.S.	05/04/04	64.8	23N	50W	12
645034	GROUSE 83	S.S.	05/04/04	64.8	23N	49W	7
645035	GROUSE 84	S.S.	05/04/04	64.8	23N	49W	7
645035	GROUSE 84	S.S.	05/04/04	64.8	23N	49W	7
645036	GROUSE 85	S.S.	05/04/04	64.8	23N	49W	8
645036	GROUSE 85	S.S.	05/04/04	64.8	23N	49W	8
645037	GROUSE 86	S.S.	05/03/04	64.8	23N	49W	8
645038	GROUSE 87	S.S.	05/03/04	64.8	23N	49W	9
645039	GROUSE 88	S.S.	05/03/04	64.8	23N	49W	9
645040	GROUSE 89	S.S.	05/04/04	64.8	23N	50W	2
645041	GROUSE 90	S.S.	05/04/04	64.8	23N	50W	2
645042	GROUSE 91	S.S.	05/04/04	64.8	23N	50W	1
645043	GROUSE 92	S.S.	05/04/04	64.8	23N	50W	1
645044	GROUSE 93	S.S.	05/04/04	64.8	23N	49W	6
645045	GROUSE 94	S.S.	05/04/04	64.8	23N	49W	6
645046	GROUSE 95	S.S.	05/04/04	64.8	23N	49W	5
645046	GROUSE 95	S.S.	05/04/04	64.8	23N	49W	5
645047	GROUSE 96	S.S.	05/03/04	64.8	23N	49W	5
645048	GROUSE 97	S.S.	05/03/04	64.8	23N	49W	4
645049	GROUSE 98	S.S.	05/03/04	64.8	23N	49W	4
645050	GROUSE 99	S.S.	05/04/04	64.8	23N	50W	2
645051	GROUSE 100	S.S.	05/04/04	64.8	23N	50W	2
645052	GROUSE 101	S.S.	05/04/04	64.8	23N	50W	1
645053	GROUSE 102	S.S.	05/04/04	64.8	23N	50W	1
645054	GROUSE 103	S.S.	05/04/04	64.8	23N	49W	6
645055	GROUSE 104	S.S.	05/04/04	64.8	23N	49W	6
645056	GROUSE 105	S.S.	05/04/04	64.8	23N	49W	5

ADL #	Claim Name	Status	Located/ T.A.'ed	Size in Hectares	Twp	Rng	Sec
645056	GROUSE 105	S.S.	05/04/04	64.8	23N	49W	5
645057	GROUSE 106	S.S.	05/03/04	64.8	23N	49W	5
645058	GROUSE 107	S.S.	05/03/04	64.8	23N	49W	4
645059	GROUSE 108	S.S.	05/03/04	64.8	23N	49W	4

Donlin Creek LLC also holds 28 claims comprising 1,813 hectares (4,480 acres) on a limestone resource in the vicinity of the Donlin Creek project (Table 4-3). The Donlin Creek LLC will need to demonstrate that the limestone is of high quality in order to secure the rights to these claims. These claims have not been legally surveyed.

Table 4-3: Claims located on Limestone Deposit

ADL #	Claim Name	Status	Located/ T.A.'ed	Size in Hectares	Twp	Rng	SEC.
641011	TUMS 1	T.A.	04/30/03	64.8	15N	45W	10
641012	TUMS 2	T.A.	04/30/03	64.8	15N	45W	10
641013	TUMS 3	T.A.	04/30/03	64.8	15N	45W	16
641014	TUMS 4	T.A.	04/30/03	64.8	15N	45W	16
641015	TUMS 5	T.A.	04/30/03	64.8	15N	45W	15
641016	TUMS 6	T.A.	04/30/03	64.8	15N	45W	16
641017	TUMS 7	T.A.	04/30/03	64.8	15N	45W	16
641018	TUMS 8	T.A.	04/30/03	64.8	15N	45W	15
641019	TUMS 9	T.A.	04/30/03	64.8	15N	45W	20
641020	TUMS 10	T.A.	04/30/03	64.8	15N	45W	21
641021	TUMS 11	T.A.	04/30/03	64.8	15N	45W	21
641022	TUMS 12	T.A.	04/30/03	64.8	15N	45W	20
641023	TUMS 13	T.A.	04/30/03	64.8	15N	45W	21
641024	TUMS 14	T.A.	04/30/03	64.8	15N	45W	21
641025	TUMS 15	T.A.	04/30/03	64.8	15N	44W	33
641026	TUMS 16	T.A.	04/30/03	64.8	15N	44W	33
641027	TUMS 17	T.A.	04/30/03	64.8	14N	44W	5
641028	TUMS 18	T.A.	04/30/03	64.8	14N	44W	4
641029	TUMS 19	T.A.	04/30/03	64.8	14N	44W	4
641030	TUMS 20	T.A.	04/30/03	64.8	14N	44W	5
641031	TUMS 21	T.A.	04/30/03	64.8	14N	44W	4
641032	TUMS 22	T.A.	04/30/03	64.8	14N	44W	4
641033	TUMS 23	T.A.	04/30/03	64.8	14N	44W	8
641034	TUMS 24	T.A.	04/30/03	64.8	14N	44W	9
641035	TUMS 25	T.A.	04/30/03	64.8	14N	44W	9
641036	TUMS 26	T.A.	04/30/03	64.8	14N	44W	8

ADL #	Claim Name	Status	Located/ T.A.'ed	Size in Hectares	Twp	Rng	SEC.
641037	TUMS 27	T.A.	04/30/03	64.8	14N	44W	9
641038	TUMS 28	T.A.	04/30/03	64.8	14N	44W	9

Additionally, Donlin Creek LLC has three prospecting sites comprising 97 hectares (240 acres) (Table 4-4). These claims have not been legally surveyed.

Table 4-4: Prospecting Sites

ADL #	Claim Name	Status	Located/ T.A.'ed	Size in Hectares	Twp	Rng	Sec
582920	PDD 9	S.S.	10/06/96	32.4	23N	48W	2
582923	PDD 12	S.S.	10/06/96	32.4	23N	48W	2
582925	PDD 15	S.S.	10/06/96	32.4	23N	48W	11

Figure 4-1: Location Map

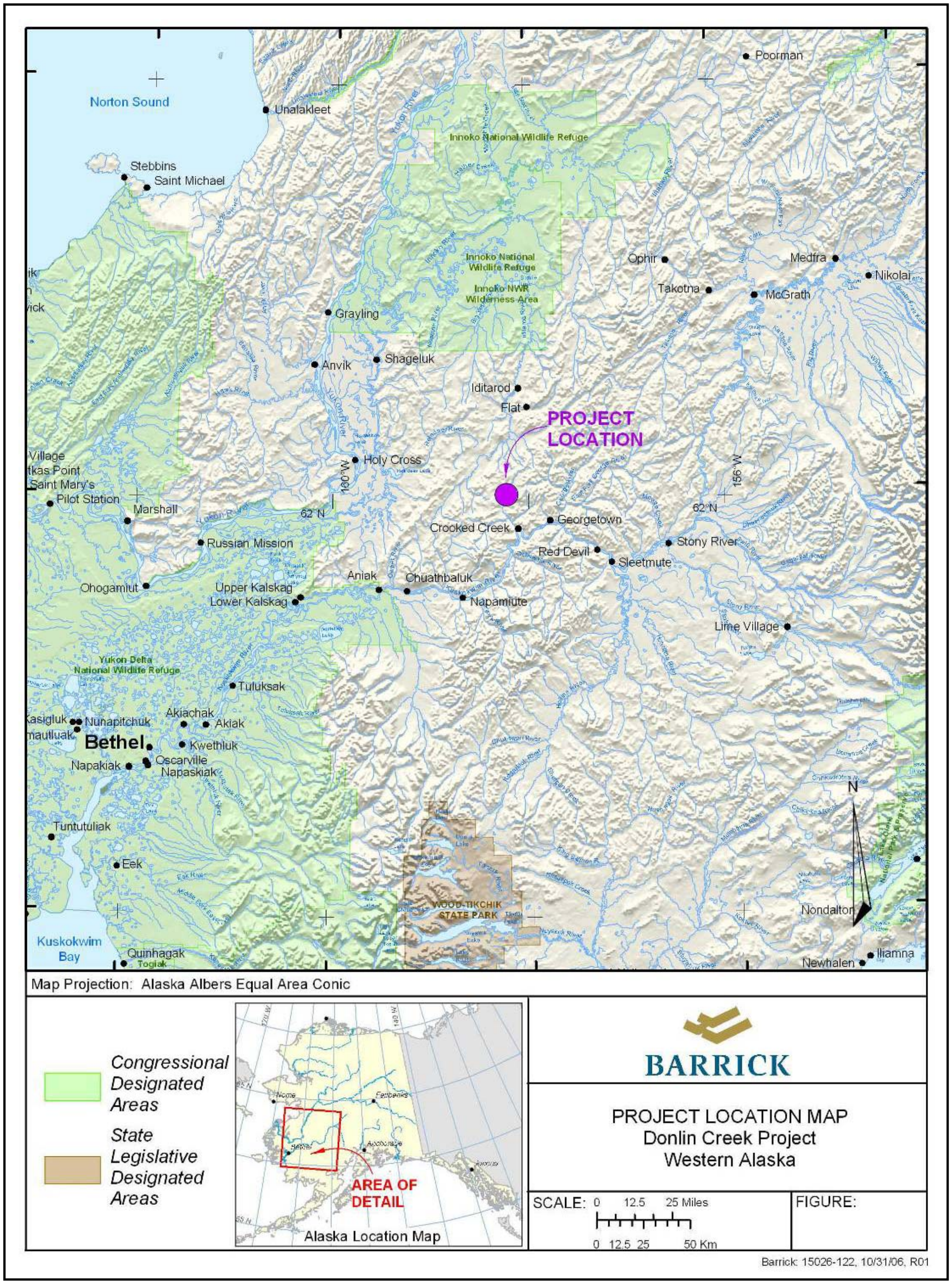


Figure 4-2: Lease Block Map

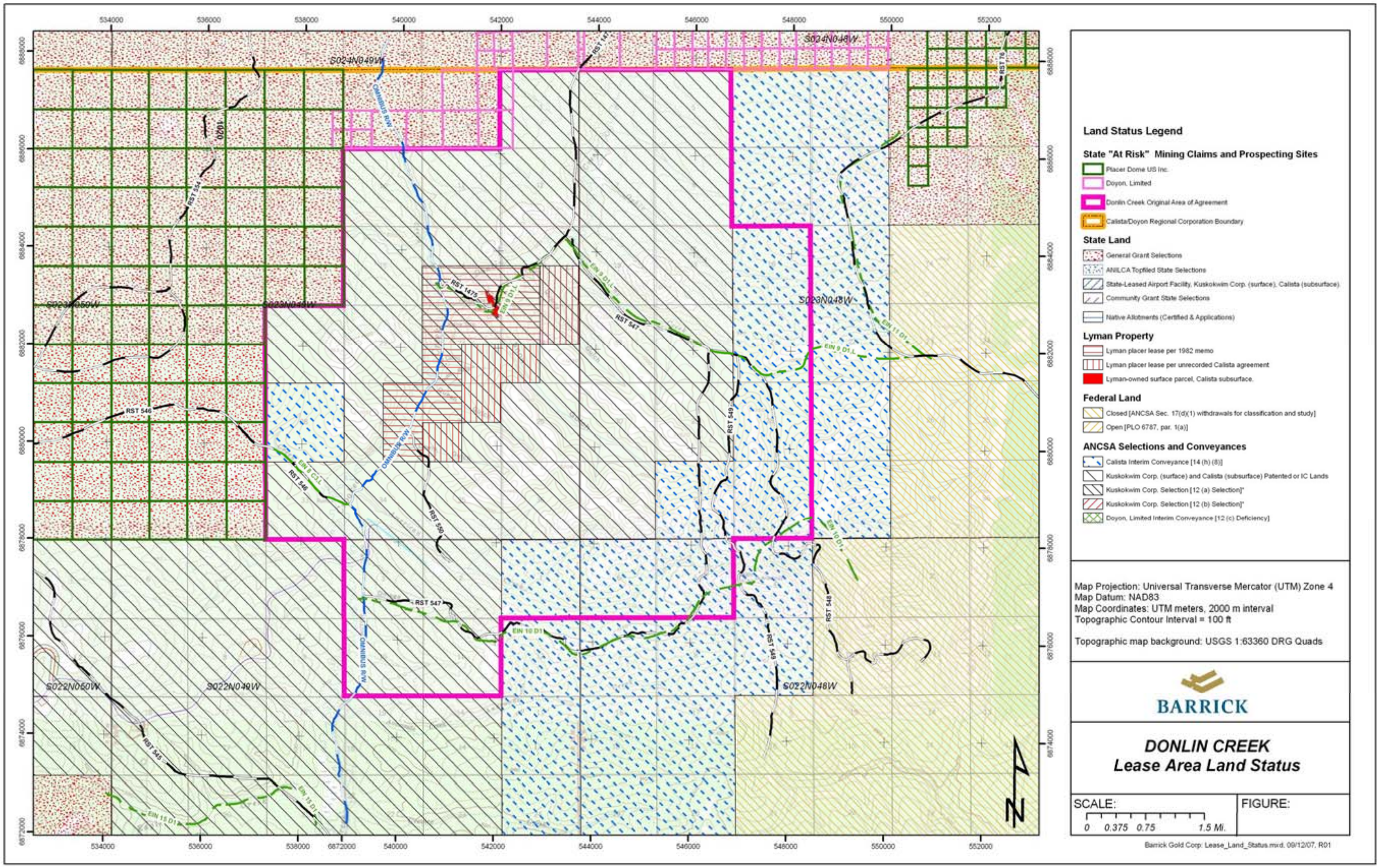
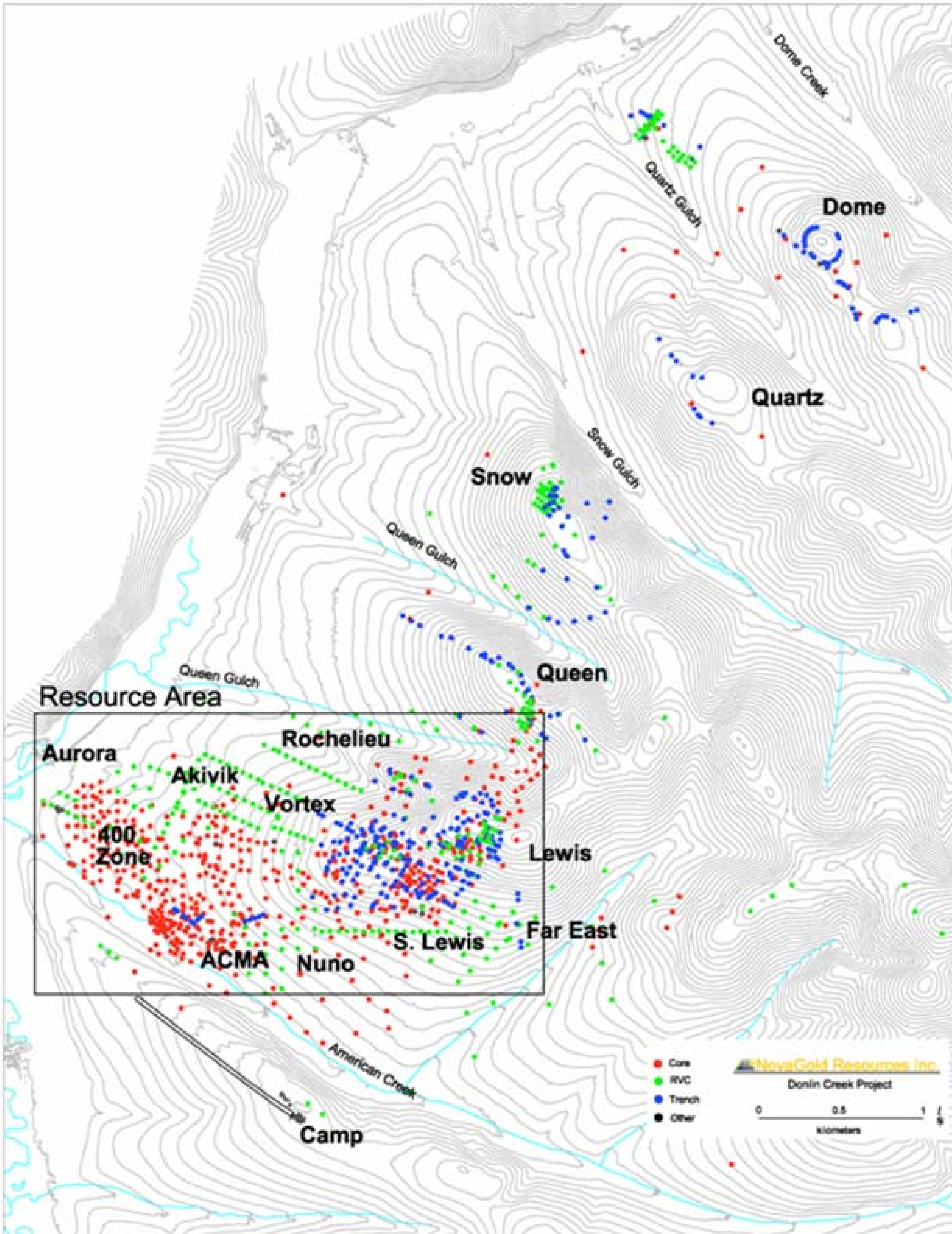


Figure 4-3: Project Location Map



4.3 Agreements and Permits

Placer Dome acquired a 20-year lease from Calista effective May 1, 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 (as discussed below). On November 13, 2002, NovaGold Resources Alaska, Inc., a wholly-owned subsidiary of NovaGold Resources Inc., earned a 70% interest in the project by expending US$10 million on exploration and development of the project. Once the financial commitment was fulfilled, Placer Dome had 90 days to decide on one of three options: a) to remain at 30% interest and participate as a minority partner; b) to convert to a 5% Net Profits Interest (NPI); or c) to exercise a back-in right to re-acquire a majority interest in the project (70%) by expending three times the amount expended by NovaGold at the time the back-in is exercised, completing a feasibility study, and making a decision to construct a mine at a production rate of not less than 600,000 ounces of gold per year within a five-year period from the exercise back-in. On February 11, 2003, Placer Dome exercised its back-in right and assumed management of the continued development of the Donlin Creek project. In January 2006, Barrick acquired Placer Dome and assumed Placer Dome's joint venture responsibilities with regard to Donlin Creek.

On December 1, 2007, NovaGold entered into a limited liability company agreement with Barrick that provided for the conversion of the Donlin Creek project into a new limited liability company, the Donlin Creek LLC, which is jointly owned by the Company and Barrick on a 50/50 basis. As part of the Donlin Creek LLC, NovaGold has agreed to reimburse Barrick over time for approximately US$63.5 million, representing 50 percent of Barrick's approximately US$127 million expenditures at the Donlin Creek project from April 1, 2006 to November 30, 2007. NovaGold's reimbursement will be made following the effective date of the agreement, by the Company paying the next approximately US$12.7 million of Barrick's share of project development costs, and the remaining approximately US$50.8 million will be paid out of future mine production cash flow. These amounts were agreed to subject to adjustment upon audit of the US$127 million expenditure. After the Company's initial contribution, all funding will be shared by both parties on a 50/50 basis. Upon submission of a feasibility study, Calista retains a 90 day back-in right to participate in the project at a level of 5% to 15% by committing to contribute its share of capital. Their share would be divided pro rata from Barrick and NovaGold.

An advance minimum royalty ("AMR") on the Donlin Creek property of US$200,000 is payable by the joint venture to Calista annually until a feasibility study is completed, after which the AMR will increase to US$500,000 per year. Upon commencement of production, a net smelter return royalty on production equal to the greater of 1.5% of the revenues from valuable minerals production and US$500,000 is payable to Calista, until the earlier of the expiry of five years or the payback of all pre-production expenses incurred by Barrick and NovaGold. Thereafter, the annual net smelter return royalty on production will be increased to the greater of 4.5% of the revenues from valuable minerals production and US$500,000.

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 should not result in any significant conflicts with the pit shell or envisioned infrastructure layout. The Calista Exploration and Lode Mining Lease grants priority to extraction of the lode resource in the event of a conflict of use between lode and placer mining operations, provided that a two-year notice period is provided to Lyman Resources. Negotiations regarding the future of the Lyman holdings are ongoing.

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).

4.4 Environmental

The environmental baseline study program was initiated in 1996 and has run continuously at varying levels of activity, except for a hiatus in 2001 and the first half of 2002. Initial work focused on studies needed to support the exploration program, largely independent of the ultimate project design. These studies included surface water quality in the general area, meteorology, aquatic studies in the main drainages, wetlands delineation in the vicinity of the known resource and some waste rock characterization.

In 2003, as project concepts began to be defined, the baseline program was expanded to include studies such as ambient air monitoring, terrestrial wildlife and avian surveys, groundwater monitoring, detailed aquatic studies, cultural site surveys, detailed waste rock characterization and additional wetlands delineation in the areas of the facilities and supporting infrastructure. Evaluation of initial baseline studies and design concepts was then used as a basis for the expansion of initial studies, the initiation of new studies and the discontinuation of older studies. As the project continued to evolve, and based on feedback from regulatory and public consultation, additional studies were initiated to address issues of concern such as mercury and the impact of barge traffic on subsistence fishing and river erosion.

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

Permits issued by federal agencies constitute "federal actions." Any major federal action requires review under NEPA. All elements of a project and their cumulative effects are considered and evaluated in a NEPA review. In addition, alternatives to the proposed action are evaluated and potential mitigation measures are identified. For Donlin Creek, NEPA will require the preparation of an environmental impact statement (EIS). Typically, under NEPA the federal agency with the predominant permit is designated the lead agency. The lead agency for this project has not yet been selected.

Over the nearly 12 years since exploration and environmental baseline data collection began, considerable effort has been spent developing support for the project by fostering local relationships, developing a strong local workforce, educating stakeholders about the project and mining in general and providing stakeholders with regular project updates and site visits. This activity has enabled the Donlin Creek LLC to better understand and address the perspectives and concerns of the project stakeholders and has resulted in broad public support for the project, especially in the upriver region surrounding the immediate project area. This support has taken the form of resolutions from tribal councils and organizations, participation by individuals and tribal groups in various project-related forums and permissions granted to conduct environmental baseline studies on tribal lands.

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

To support successful application for the more than 60 permits, this project will likely require extensive baseline environmental information, supporting scientific analysis, and detailed engineering design. Donlin Creek LLC and predecessors have invested significant money, resources, and time acquiring this information over the last 5 years, and in some cases the last 12 years. Designing in line with baseline data in advance of filing permit applications has resulted in a project that affords due consideration to all environmental concerns and is designed to mitigate potential impacts on the environment wherever practicable.

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

1. Exploration stage permitting – required to obtain approval for exploration drilling, environmental baseline studies, and feasibility engineering studies.
2. Pre-application phase – conducted in parallel with feasibility engineering studies. This stage includes the collection of environmental baseline data and interaction with stakeholders and regulators to facilitate the development of a project that can be successfully permitted.
3. The NEPA process and formal permit applications – formal agency review and analysis of the project, resulting in the issuance or denial of permits.

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

Permit review timelines are controlled by the requirements of the federal NEPA review and State requirements for meaningful public and agency participation to determine if the project is in the State's best interest. Having engaged in comprehensive dialogue with stakeholders and regulatory agencies, and by moving forward with a well-defined project description, the Donlin Creek LLC has positioned itself well for an optimal permit review timeline.

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

4.5 Permits and Process

The Donlin Creek project will require multiple State and Federal permits, approvals, licenses, and authorization from Federal, State, Local and Tribal governments. Table 4-1 provides a brief summary of the permits required for the Donlin Creek project.

Table 4-5: Federal Agency Permit and Authorizations

Agency	Authorization
Federal	
Bureau of Land Management (BLM)	Surface Estate Lease (facilities managed lands) Land Use Permit (activities on BLM managed lands) Access Right-of-Way (BLM managed lands)
Environmental Protection Agency (EPA)	CWA Section 402 NPDES Permit (discharges to waters of the U.S.) Spill Prevention Containment and Contingency (SPCC) Plan Storm Water Pollution Prevention Plan – Construction and Operations
U.S. Army Corps of Engineers (USACE)	CWA Section 404 Permit (wetlands dredge and fill) River and Harbors Act (RHA) Section 10 (structures in navigable waters) RHA Section 9 (dams and 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 requires a permit)
Bureau of Alcohol, Tobacco, and Firearms	License to Transport Explosives Permit and License for Use of Explosives
Federal Aviation Administration	Notice of Landing Area Proposal (existing airstrip) Notice of Controlled Firing Area for Blasting
U.S. Department of Transportation	Hazardous Materials Registration
National Marine Fisheries Service	Marine Mammal Protection Act authorization (IHA/LOA)
U.S. Fish and Wildlife Service	Section 7 of the Endangered Species Act, Consultation requiring a Biological Assessment or Biological Opinion
State	
Office of Project Management and Permitting	Alaska Coastal Management Program Consistency Applicability Determination
Division of Mining, Land, and Water	Plan of Operations Reclamation Plan Approval Mining License Land Use Permits and Leases Right-of-Ways, Easements, Material Sales, etc. Certificate of Approval to Construct a Dam Certificate of Approval to Operate a Dam Temporary Water Use Permit Water Rights Permit/Certificate to Appropriate Water Tidelands Permit
Office of History and Archaeology/State Historic Preservation Office	Section 106 Historical and Cultural Resources Protection Act clearance
Office of Habitat Management and Permitting	Fish Habitat Permit Culvert/Bridge Installation Permit

Agency	Authorization
Division of Water	Section 401 Water Quality Certification (CWA 404 permit) Section 401 Water Quality Certification (CWA 402 permit) Wastewater Disposal Permits Non-Domestic Wastewater Disposal Permit Storm Water Discharge Pollution Prevention Plan Domestic Wastewater Disposal Permit Approval to Construct and Operate a Public Water Supply System
Division of Environmental Health	Solid Waste Disposal Permits Food Sanitation Permit
Division of Air Quality	Air Quality Construction Permit (first 12 months) Air Quality PSD Title V Operating Permit (after 12 months) Air Quality permit to Open Burn

Each Federal and State permit will have compliance stipulations that require scrutiny and negotiation that can typically be resolved within 60 days of the ROD.

Project delays could occur as public opposition, inefficiencies in regulator review or project changes made by the owner.

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

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Donlin Creek property is located in southwest Alaska, approximately 19 km (12 mi) north of the village of Crooked Creek on the Kuskokwim River (see Figure 4.1). The Kuskokwim River is a regional transportation route and is serviced by commercial barge lines. A 25 km (15 mi) long winter road, designated as an Alaska State Highway route and transportation corridor, accesses the property from the barge site at the village of Crooked Creek (Figure 4.2). The project has an all-season camp with facilities to house up to 160 people and an adjacent 1,500 m (5,000 ft) long airstrip capable of handling aircraft as large as C-130 Hercules (19,050 kg or 42,000 lb capacity), allowing efficient shipment of personnel, large equipment and supplies. The project is directly serviced by commercial air services out of both Anchorage, 450 km (280 mi) to the east, and Aniak, 80 km (50 mi) to the west.

The area has a relatively dry interior continental climate with typically less than 50 mm (20") total annual precipitation. Summer temperatures are relatively warm and may reach nearly 30°C (83°F). Minimum temperatures may fall to well below -20°C (0°F) during the winter months.

The project area is one of low topographic relief on the western flank of the Kuskokwim Mountains. Elevations range from 150 m to 640 m (500 ft to 2,100 ft). 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.

The Maximum Design Earthquake for the “High” hazard potential water and tailings dams is characterized by a peak horizontal ground acceleration of 0.4g from a Magnitude 7.8 event. For the design of plant site buildings and other structures, the seismic design provisions of the 2006 International Building Code result in a mapped short-period spectral response acceleration, Ss, of 0.5g, and a mapped 1s period spectral response acceleration, S1, of 0.14g. Site Class B conditions are anticipated for the plant site, pending future confirmation.

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. In regard to 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. Ample water supply is available from surface and subsurface sources. Additional diesel generators or alternative power sources would need to be developed for mining operations.

6.0 HISTORY

Prior to 2003, operators have undertaken significant work on the property. Table 6-1 summarizes the work history at Donlin Creek.

Table 6-1: Work History Summary

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 1996	Robert Lyman and heirs	Resumed sluice mining in Donlin area and placer mined Snow Gulch.	800 oz Au recovered in the first year of operation in Snow Gulch.
1974, 1975	Resource Associates of Alaska (RAA)	Regional mineral potential evaluation for Calista Corporation. Soil grid and 3 bulldozer trenches dug in Snow area.	Soil, rock and vein samples return anomalous gold values. Trench rock sample results range from 2 ppm Au to 20 ppm Au.
1984 to 1987	Calista Corporation	Minor work. Various mining company geologists visit property.	
1986	Lyman Resources	Placer drilling finds abundant gray, sulphide-rich clay near Quartz Gulch.	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	Western Gold Exploration and Mining Co. (WestGold)	Airborne geophysics, geologic mapping and soil sampling over most of project area. Total of 13,525 m of D-9 Cat trenching at all prospects. Over 15,000 soil, rock chip and auger samples collected. 947 m of AX core drilling, 404 m (239 holes) of auger drilling and 10,423 m of RC drilling (125 holes). First metallurgical tests and petrographic work.	Initial work identified eight prospects with encouraging geology ± Au values (Snow, Dome, Quartz, Carolyn, Queen, Upper Lewis, Lower Lewis and Rochelieu). Drilling at most of these prospects led to identification of the Lewis areas as having the best bulk-minable 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 Exploration Ltd.	1,400 m of D-9 Cat trenching and two 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 M t at average grade of 3.15 g Au/t (393,000 oz Au).

Year	Company	Work Performed	Results
1995 to 2000	Placer Dome	87,383 m of core, 11,909 m of RC drilling and 8,493 m of trenching. Environmental work.	Discovery of American Creek Magnetic Anomaly (ACMA) when testing an aeromag anomaly. Numerous mineral resource calculations.
2001, 2002	NovaGold	39,092 m of core, 11,589 m of RC drilling, 89.5 m of geotechnical drilling and 268 m of water monitoring holes. Updated resource estimate.	43-101 Preliminary Assessment Measured and indicated: 117.4 million tonnes at 2.91 g/t (1.5 g/t cut-off), Inferred: 142.4 million tonnes at 3.1 g/t (1.5 g/t cut-off)
2003 through 2005	Placer Dome	25,448 m of core and 5,979 m of RC drilling	Infill drilled throughout the resource area. Discovered a calcium carbonate resource.
2006 through 2007	Barrick		Infill drilled throughout the resource area. Completion of DC7a resource estimation model.

6.1 1996 Activities

Major activities included:

- building a 75-person Weatherhaven tent camp
- constructing a 1,500 m (5,000 ft) airstrip on American Ridge
- constructing more than 4 km (2.5 mi) of new road between camp and mineral prospects
- drilling a total of 34,995 m (144 holes, both core and reverse circulation)
- assaying more than 21,000 drill, rock and soil samples
- excavating more than 2,500 m of trenches for sampling and mapping purposes in southeast Lewis area.

Most core drilling was on Lewis and Queen ridges, but eight core holes were drilled on the Dome, Far Side (formerly Carolyn) and Snow prospects. Seven RC drill holes were located at the southern end of an aeromagnetic anomaly southwest and west of Lewis Ridge. Four water wells were drilled for camp and drilling purposes.

Metallurgical studies were conducted on both sedimentary- and igneous-hosted mill feed from the Lewis area.

6.2 1997 Activities

The goal of the 1997 exploration program was to develop a structural/geologic model of the Lewis/Queen area that would assist in determining mineralization controls. The following tasks were completed during 1997:

- 8,129 m of reverse circulation (RC) drilling in 52 holes concentrated in wetlands and environmentally sensitive areas
- 15,771 m of HQ core drilled in 67 holes across the property
- 4,222 m of trenches excavated and a detailed geologic and mineralization map completed in the Lewis area
- Air photos taken of the Donlin Creek project area
- 25 line km of max-min (electromagnetic) geophysical survey completed in the ACMA, 400 and southern Lewis areas
- 1,800 line km of aeromagnetic survey completed at 50 m line spacing and 50 m elevation over the property
- more than 600 soil samples collected in the ACMA and 400 areas
- 2,100 m of 1996 and 1997 trenches reclaimed in the Lewis area
- continuation of baseline environmental studies.

6.3 1998 Activities

The main tasks completed in 1998 include:

- 24,131 m of HQ core drilled in 96 holes, mainly in the Lewis, Queen and ACMA areas (ACMA discovered when testing a magnetic anomaly)
- 1,904 m of trenching and mapping in the Lewis/Vortex areas and 150 m of trenching and mapping in the ACMA area (includes re-trenching and re-mapping of older trenches)
- Air photos taken of the Donlin Creek project area from the airstrip to Dome at 1:20,000 scale
- geological reconnaissance within the Donlin Creek property boundary
- ongoing reclamation of trenches throughout the property
- continuation of baseline environmental studies.

6.4 1999 Activities

Two programs were completed during 1999: an exploration drilling program focused in the ACMA/400 area, and a property-wide exploration program to locate other higher-grade prospects. Results were:

- 9,189 total m of core drilled in 33 holes
- 646 soil samples and 92 rock samples collected
- 17.7 km of IP and resistivity lines completed
- 2,237 m of trenching and mapping (Dome, Queen, Far Side and Vortex)
- property-wide 1:10,000 geological mapping
- ongoing reclamation of trenches throughout the property (900 m reclaimed in 1999)
- continued baseline environmental studies.

6.5 2000 Placer Dome

Work during 2000 included an IP/resistivity survey and a drill program to test IP/resistivity anomalies coincident with soil geochemistry anomalies generated in the Dome-Quartz area. Results included:

- 41.6 km of IP/resistivity lines
- 1,403 m of core drilled in 7 holes from the Dome and Quartz areas
- completion of a supplemental resource economic study
- continued baseline environmental studies.

6.6 2001 NovaGold

NovaGold began field work on the project in 2001 after finalization of a joint venture agreement with Placer Dome. Work in 2001 included the following:

- 7,403 m of HQ core drilled in 42 holes from the ACMA area
- 822 m of trenching in the Lewis area

6.7 2002 NovaGold

NovaGold continued work on the property in 2002 focusing on expanding both the ACMA resource and defining mineralization and new resources in adjacent prospect areas (Aurora, 400, Akivik as well as Vortex). Work in 2002 included the following:

- 39,092 m of HQ core in 194 holes from the ACMA, Aurora, 400, Akivik and Vortex areas
- 89.5 m of HQ core in 2 geotechnical holes from Anaconda Creek
- 11,589 m of exploration RC drilling and sampling in 147 holes from the ACMA, Akivik, Aurora and Nuno areas
- 268 m of RC drilling in 5 water monitoring wells
- resource estimation and preliminary assessment (AMEC, 2002a and 2002b)

- Measured and indicated: 104.1 million tonnes at 3.00 g/t (1.5 g/t cut-off)
- Inferred: 129.1 million tonnes at 3.11 g/t (1.5 g/t cut-off)
- contracted an updated economic study
- continued baseline environmental studies.

Stephen Juras, P.Geo., of MRDI Canada, a division of AMEC E&C Services Limited (MRDI), estimated the mineral resources for the project effective January 24, 2002, and issued a report in February 2002 entitled "Technical Report, Donlin Creek Project, Alaska". The report is filed on SEDAR under NovaGold Resources Inc.

MRDI reports the estimates were made from 3D block models utilizing commercial mine planning software (MineSight®). Mineralized domains were interpolated based on mineralized geology and grade estimation based on ordinary kriging. Mineralized envelopes were defined through PACK on gold thresholds of 0.7 g/t Au and 0.5 g/t Au for the ACMA and Lewis zones, respectively. Gold values, in the 2 m composites, were capped at 30 g/t Au and 20 g/t Au for the ACMA and Lewis zones, respectively.

MRDI reported mineral resources at gold prices ranging from US$250 to US$350 per ounce of Au and corresponding cut-off grades of 3.5 to 1.5 g/t Au respectively. The capped mineral resource estimate at US$350 per ounce Au is summarized in Table 6-2. MRDI reported that the mineral resource classification was consistent with CIM definitions referenced in NI43-101. The resource estimate is reliable but is irrelevant given the material amount of drilling completed since 2002.

Table 6-2: Mineral Resources - MRDI - January 24, 2002

Classification	Tonnes (t 000's)	Grade (g/t Au)	Cont. Au (oz 000's)
Measured			
ACMA	2,262	4.48	326
Lewis	4,331	2.76	384
Total Measured	**6,593**	**3.35**	**710**
Indicated			
ACMA	32,327	3.54	3,678
Lewis	65,203	2.70	5,652
Total Indicated	**97,530**	**2.98**	**9,330**
Measured + Indicated			
ACMA	34,589	3.60	4,004
Lewis	69,534	2.70	6,036
Total Measured + Indicated	**104,123**	**3.00**	**10,040**
Inferred			
ACMA	48,852	3.53	5,550
Lewis	80,291	2.86	7,371
Total Inferred	**129,144**	**3.11**	**12,921**

Notes: 1. Mineral resources estimated at gold price of US$350/oz
2. Mineral resources estimated at cut-off grade of 1.5 g/t Au
3. Columns may not total exactly due to rounding

6.8 2003 Placer Dome

Placer Dome elected to return as operator in 2003 as per the joint venture agreement. Work in 2003 included the following:

- updated the resource estimation based on NovaGold's 2002 and previous drill programs (AMEC, 2003)
    - Measured and indicated: 117.4 million tonnes at 2.9 g/t (1.5 g/t cutoff)
    - Inferred: 142.4 million tonnes at 3.1 g/t (1.5 g/t cutoff)
- calcareous sandstone investigations
- economic studies

6.9 2004 Placer Dome

Placer Dome focussed on environmental and geotechnical studies in 2004. Work included the following:

- 2,335 m of RC drilling and sampling in 17 condemnation holes
- 852 m of HQ core in 3 geotechnical holes
- geologic mapping and sampling for carbonate-rich material
- continued environmental baseline studies

6.10 2005 Placer Dome

Placer Dome focussed on resource conversion, geotechnical investigation and environmental studies in 2005. Work included the following:

- 24,596 m of HQ core (resource infill, geotechnical, condemnation) in 90 holes from the ACMA, Akivik, 400 Vortex, Lewis and Far East areas
- 3,644 m of RC drilling and sampling in 30 condemnation, water well and calcium carbonate exploration holes
- 154 m in 28 auger holes for geotechnical purposes
- 22 test pits for geotechnical purposes
- continued environmental baseline studies
- continued economic studies

6.11 2006 NovaGold

NovaGold reviewed and validated a mineral resource model constructed by Placer Dome with an effective date of January 19, 2006. Stanton Dodd authored a report entitled “Donlin Creek Project, 43-101 Technical Report”, dated January 20, 2006. The report is available on SEDAR under NovaGold Resources Inc.

The resource estimates were made from 3D block models using proprietary Placer Dome mine modelling software (OP). Table 6-3 summarizes the resource estimate. The resource estimate is reliable but is irrelevant given the material amount of drilling completed since January 2006.

Table 6-3: Mineral Resources - NovaGold - January 19, 2006

Classification	Tonnes (t 000's)	Grade (g/t Au)	Cont. Au (oz 000's)
Measured	16,100	2.84	1,469
Indicated	151,100	2.75	13,360
Measured + Indicated	**167,200**	**2.76**	**14,829**
Inferred	**156,000**	**2.72**	**13,643**

Notes: 1. Mineral resources estimated at gold price of US$450/oz
2. Mineral resources estimated at cut-off grade of 1.2 g/t Au
3. Columns may not total exactly due to rounding

A September 2006 preliminary economic assessment ("PEA") prepared for NovaGold by SRK Consulting (US), Inc. confirmed the economics of a conventional open-pit mining operation at a production rate of 60,000 t/d with the potential to produce on average 1.4 million ounces of gold per year over the estimated 22-year life of the project in the report titled "Preliminary Assessment Donlin Creek Gold Project Alaska, USA" dated September 20, 2006. The report is available on SEDAR under NovaGold Resources Inc.

Costs, appropriate with this level of the study, were estimated and formed the foundation of the economic analysis of the project on a 100% basis. The study was prepared based on a technical and economic review by a team of consultants who are specialists in the fields of mineral exploration, mineral resource estimation and classification, open-pit mining, mineral processing and mineral economics. The study was completed under the direction of Gordon Doerksen, P.E., an independent Qualified Person as defined by NI 43-101. The resource estimate with an effective date of January 19, 2006 was the basis of the PEA.

SRK also completed a sensitivity analysis to determine the economic effects of changes to the capital and operating costs and the gold price, to determine the economic potential of the Donlin Creek project. In the first 7 years, the study projects average annual production of approximately 1.885 million ounces of gold at an average cash cost of US$223/oz of gold. The project would generate an average annual pre-tax cash flow of approximately US$482 million for the first 7 years using a long-term gold price of US$500/oz, resulting in rapid payback of all mine capital in less than 5 years.

SRK's analysis indicated that using a gold price of US$500/oz, Donlin Creek could generate a pre-tax rate of return of 12.1% and a net present value (NPV) at a 5%

discount rate ("NPV5%") of US$1,001 million, resulting in a capital cost payback period of less than 5 years. A sensitivity analysis on the project shows that the NPV is most sensitive to changes in the gold price, followed by changes to operating costs and capital costs. For example, a gold price of US$550/oz increases the NPV5% to US$1,453 million, and a gold price of US$450/oz decreases the NPV5% to US$554 million.

This financial analysis includes capital costs to construct a powerline connecting the Donlin Creek project site to the existing Anchorage/Fairbanks power grid. The study was preliminary in nature, and included inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would allow them to be categorized as mineral reserves, and there is no certainty that the conclusions reached in this PEA will be realized. The PEA is reliable but is not current given the material amount of drilling and technical studies completed since January 2006 and possible capital cost price escalation.

6.12 2006–2007 Barrick

Barrick focussed on resource conversion, geotechnical investigation, metallurgical and environmental studies from 2006 through 2007. Work included the following:

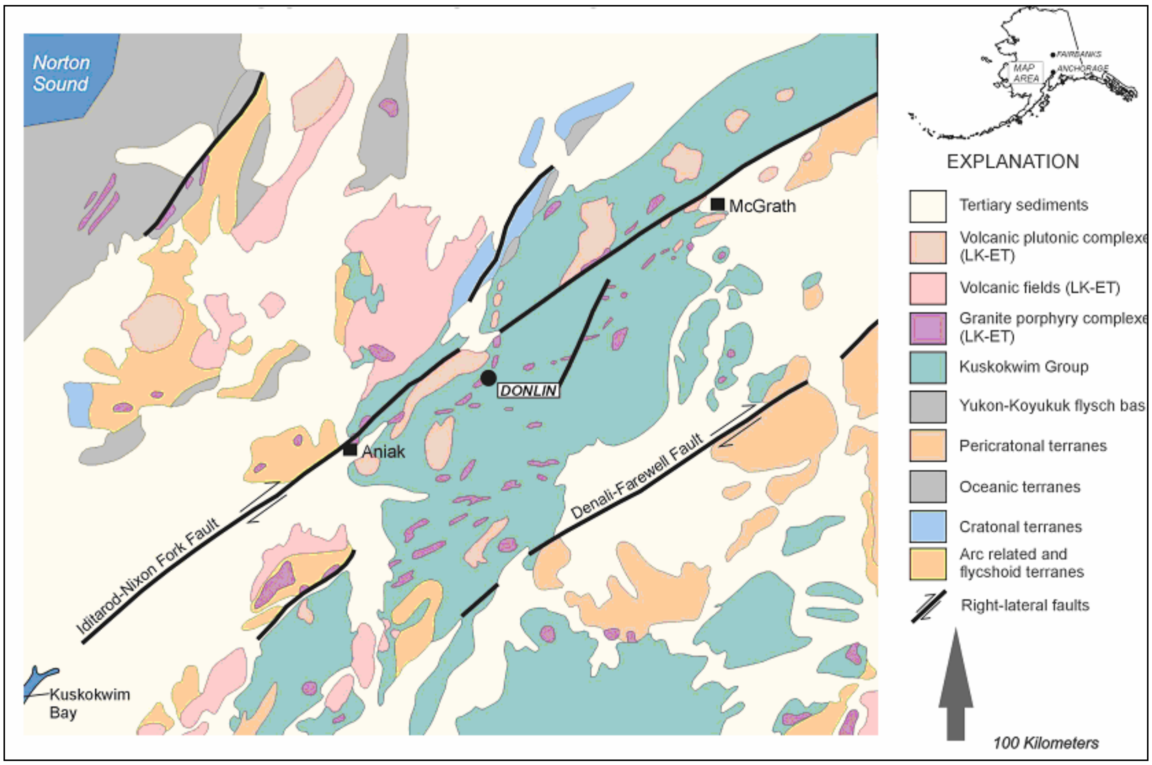
- 135,362 m (444,101 ft) of primarily HQ core (resource infill, geotechnical, metallurgical, condemnation) in 359 holes in Lewis and ACMA.
- continued environmental baseline studies
- water geochemistry
- peat exploration
- wind power generation studies
- metallurgical studies

7.0 GEOLOGICAL SETTING

7.1 Regional Geology

The Kuskokwim region of southwestern Alaska is predominately underlain by rocks of the Upper Cretaceous Kuskokwim group (Figure 7-1). These include coarse- through fine-grained clastic rocks that reach an estimated thickness of 12 km (7.5 miles). Minor basin margin andesitic tuff and flows are also present near the top of the sequence and may represent an initiation of volcanism that later culminated in widespread Late Cretaceous and early Tertiary igneous activity. These basin margin volcanic rocks also suggest that deep penetrating structures controlled basin subsidence.

Figure 7-1: Regional Geology of Donlin Creek Area



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

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

The Donlin Creek area lies between two regional, northeast-trending, right lateral faults: the Denali–Farewell fault system to the south, and the Iditarod–Nixon Fork fault system to the north. The region contains numerous northeast to east-northeast- and northwest to west-northwest-trending lineaments that probably represent steeply dipping strike-slip faults. Fault movement in the Donlin Creek region appears to be right lateral on northeast structures and left lateral on northwest structures. Folding in the region probably occurred soon after sedimentation, since folds are truncated by the volcano-plutonic complexes. East-trending open folds are prominent east of the Donlin Creek area, but appear truncated to the west by the Donlin Creek fault, a splay of the Iditarod–Nixon Fork fault.

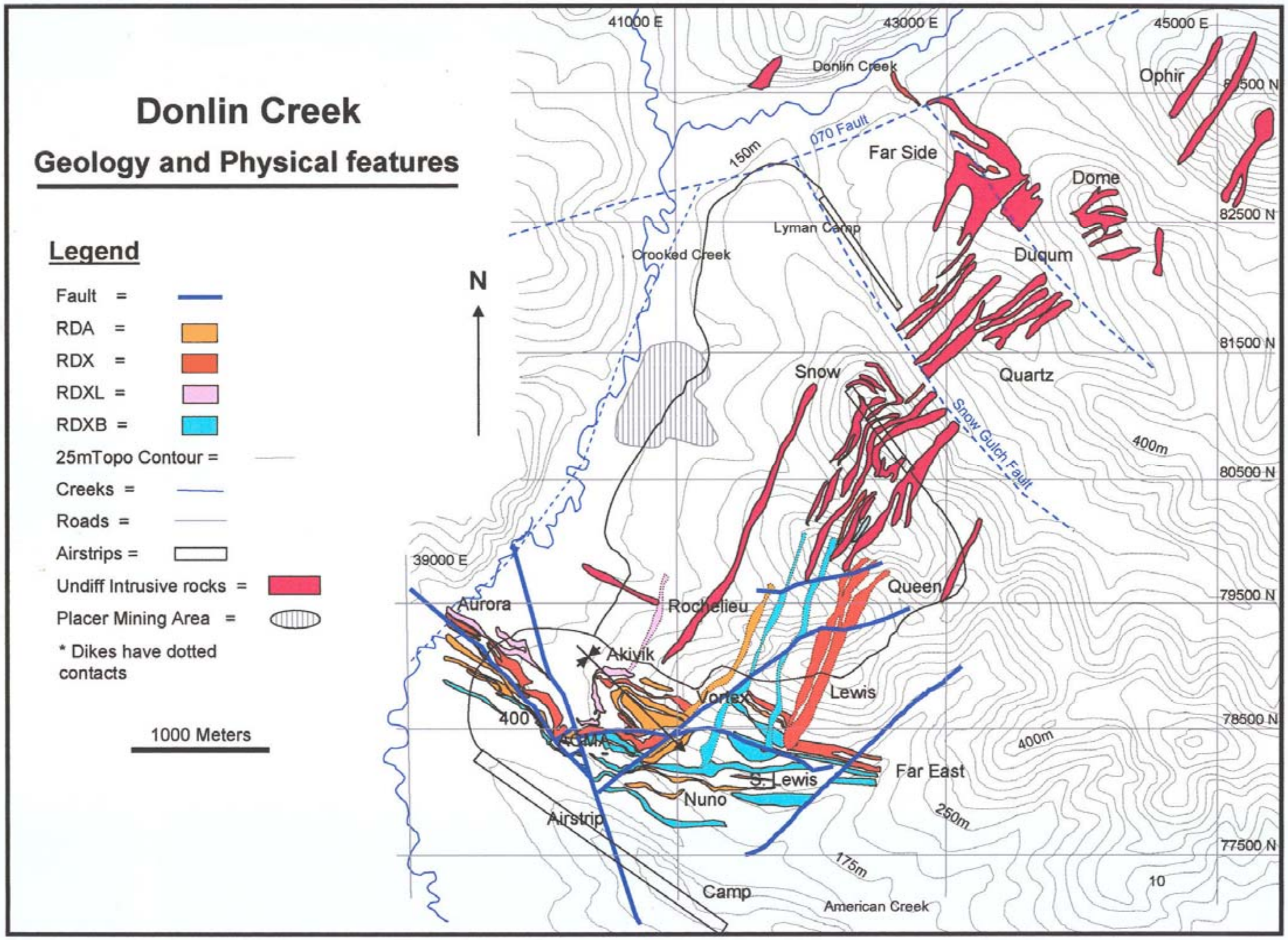
7.2 Property Geology

Simplified property scale igneous geology is shown in Figure 7-2. Undivided Kuskokwim Group sedimentary rocks (uncoloured) and felsic intrusive rocks associated with the 70 to 65 Ma igneous event are the main rock units. Greywacke is dominant in the northern part of the resource area (Lewis, Queen, Rochelieu, Akivik), while shale-rich units are common in the southern part of the resource area (South Lewis, ACMA). Sedimentary bedding strikes northwest and dips moderately to the southwest. Overall, sedimentary structure in the northern resource area is monoclinal, while sedimentary rocks in the southern resource display open easterly-trending folds.

The earliest intrusive rocks at Donlin Creek are 74 to 72 Ma intermediate to mafic dykes and sills. They are not abundant, but occur widely throughout the property as generally thin and discontinuous bodies. The later and much more voluminous 70 to 65 Ma felsic dykes and sills vary from a few metres to 60 m (200 ft) wide and intrude the sedimentary rocks along a 8 km long by 3 km wide (5 mile x 2 mile), northeast-trending corridor. Sills are common in the southern resource area (shale-dominant), while dikes dominate in the north (greywacke-dominant). The felsic dykes and sills have similar mineralogy and generally display a porphyry texture indicative of relatively shallow emplacement. Although these rocks belong to the regionally important granite porphyry igneous event, geologists working on the property classify them into five textural varieties of rhyodacite. Rhyodacite is a term normally reserved for volcanic to sub-volcanic rock types, but it is also used informally for igneous rocks emplaced at a shallow depth. These units are chemically similar, temporally and spatially related, and probably reflect textural variations of related intrusive events. Differences include phenocryst size and abundance, groundmass texture, and overall colour.

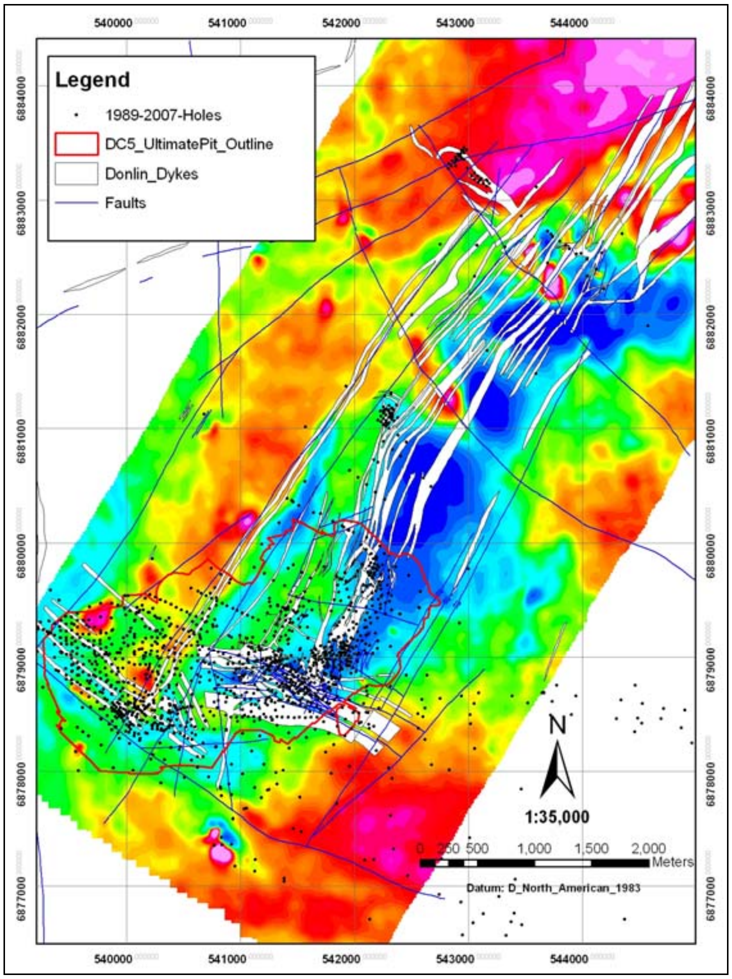
In chronological order, the oldest documented structures are easterly-trending open folds and southerly-directed thrusts or low to moderate north-dipping reverse faults. The ca. 70 Ma north-northeast-striking rhyodacite porphyry dikes and west-northwest-striking sills were emplaced in multiple pulses and post-date the minor mafic dyke and sill event. Cross-cutting high- and low-angle northeast- and northwest-striking faults developed after the thrusts, and may have been active during, as well as after, igneous and hydrothermal activity. These structural trends are clearly evident in the surface geology and as topographical and aeromagnetic linears. Finally, gold-bearing north-northeast-striking extensional fractures formed and cut igneous rocks and faults. The fractures are best developed in the relatively competent igneous rocks and coarser greywacke-dominant sedimentary sequences.

Figure 7-2: Main Trend Geology (Piekenbrock and Petsel 2003)



As shown in Figure 7-3, the geophysical (magnetic) expression of the hydrothermal system is a pronounced northeast-trending aeromagnetic low (blue shades) related to the low magnetic susceptibility of the intrusive rocks, fracture-controlled, magnetite-destructive hydrothermal alteration, and possible weak thermal metamorphism of the enclosing sedimentary rocks.

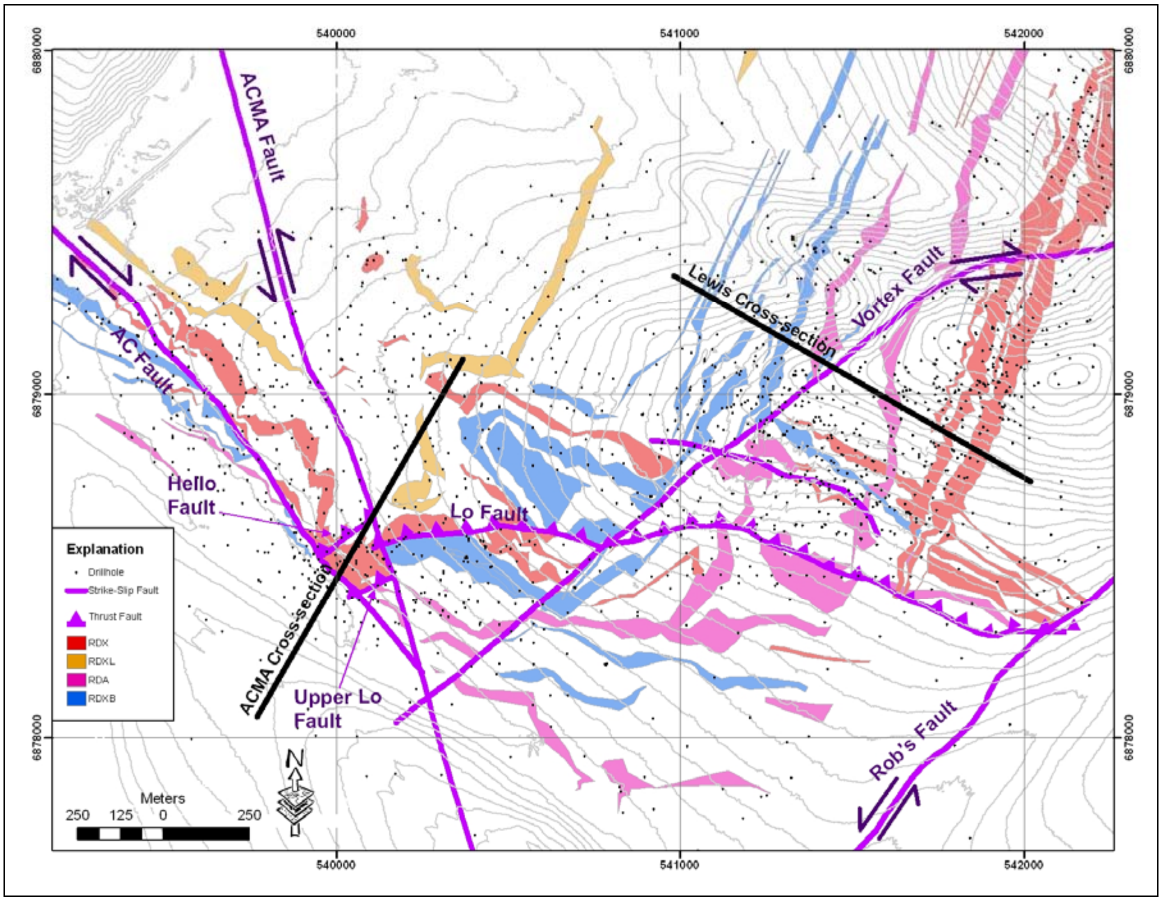
Figure 7-3: Aeromagnetic Image showing Interpreted Faults and Intrusive Rocks (source NovaGold)



7.3 Deposit Geology

General geology of the resource area is shown in Figure 7-4. The 100 m (328 ft) level of the geological model is projected to the topographic surface.

Figure 7-4: General Geology of the Resource Area Showing Intrusive Rock Units and Faults (100 m level projected to surface) (source Barrick)



7.3.1 Sedimentary Stratigraphy

Preliminary stratigraphy for sedimentary rocks in the immediate deposit area is shown in Table 7-1.

The stratigraphy in the deposit area consists of multiple turbidite sequences and is therefore very complex. Transition zones of rhythmically interbedded shales and lithic sandstones are common. Marker beds are not yet recognized, so absolute stratigraphic breaks are difficult to identify.

Table 7-1: Donlin Creek 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/argillite	up to 459 (with sills)	up to 140 (with sills)
Basal Greywacke	greywacke	656	200+

Thicknesses presented for each unit in Table 7-1 are from the southern resource area also known as the American Creek magnetic anomaly, or ACMA area. In general, the Main Shale appears to thin to the west, whereas the Upper Siltstone appears to thicken in the same direction. The northern part of the resource area is mostly greywacke, while the southern area is shale rich. The coarse-grained greywacke contains abundant metamorphic lithic fragments and locally abundant igneous and sedimentary clasts. Shale-rich sedimentary rocks contain minor syngenetic pyrite, minor coaly plant debris to thin coal seams, and rare thin 10 cm (0 to 4") volcanic ashfall beds. The ash beds are restricted to low-energy shale and argillite intervals.

7.3.2 Igneous Rocks

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

Table 7-2: Donlin Creek Intrusive Rocks

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

MD – Mafic Dykes

The earliest intrusive rocks at Donlin Creek are a series of intermediate to mafic dykes and sills (MD). These dykes and sills are thin 1 to 3 m (3 to 10 ft) and are normally characterized by intense pervasive carbonate and bright green clay + fuchsite (?) alteration. The mafic rocks are compositionally variable, typically porphyritic, and have been compared to lamprophyres. In the transition area between Akivik and ACMA, an area of extremely abundant mafic sills occurs within the Lower Greywacke immediately below the Main Shale. The mafic sills locally host high-grade gold.

RDF – Fine-Grained Porphyry

The RDF dykes are the earliest rhyodacite intrusions recognized at Donlin. They are typically fine-grained, felsic porphyries with distinctive small feldspar phenocrysts set in a grey fine-grained matrix. RDF intrusives occur as dykes 5 to 10 m (16.5 to 32.8 ft) wide and appear to fill the north-northeast extension fracture zones and the east-striking compressional (e.g., Lo fault) faults.

RDX – Crowded Porphyry

The RDX rocks are volumetrically the most significant intrusive phase on the property. The unit is characterized by a homogenous crowded porphyry texture and sharp intrusive contacts with little (<2" or <5 cm) to no chill margins. The unit occurs as two 50 to 100 m (164 to 328 ft) wide dike zones in the eastern edge of the north to north-northeast Lewis/South Lewis mineralized trend. RDX also occurs as sills throughout the southern portion of the property as the lowest part of the stratigraphy. The sills begin as sub-horizontal units in the South Lewis area and follow the syncline/anticline structure as they dip from sub-horizontal to near-overturned at depth in the ACMA area.

RDXL – Lath-Rich Porphyry

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

RDA – Aphanitic Porphyry

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

RDXB – Blue Porphyry

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

7.3.3 Structural Geology

The sedimentary rocks in the northern part of the resource area are monoclinal with average dips to the southwest of about 10° to 50°. Southeasterly-plunging open folds are evident in the southern resource area near the Lewis–ACMA transition (note intrusive rock map, Figure 7-4). Recent drilling shows that the south limb of the prominent syncline is vertical to overturned toward American Creek. Structural data from deep oriented core holes show an abrupt change from vertical to nearly horizontal bedding attitudes at depth. This vertical fold limb and the deep nearly flat bedding may suggest drag on a major low-angle fault at depth beneath ACMA.

North-dipping low-angle reverse or thrust faults (?), including the Lo Fault and similar structures (Figure 7-4), are believed to have formed along stratigraphic competency breaks during north-northeast directed compression. There is some suggestion that the Lo Fault and the related upward transition from more massive greywacke to shale may have influenced the change from dykes in the Lewis area (Figure 7-5) to sills in the ACMA area (Figure 7-6). Relative displacements and an oblique sense of movement on later northwest and northeast high-angle faults are portrayed in Figure 7-4. Rob's Fault, southeast of the known resource area, is very poorly understood, except where intersected in a few widely spaced shallow drill holes. None of these faults hosts significant mineralized material, but they may have locally enhanced ground preparation and influenced the circulation of gold-bearing hydrothermal fluids (fluid barriers). However, drilling in 2006 intersected gold-bearing, sulphide-rich tectonic and possible hydrothermal breccias in structures subparallel and near the Vortex fault. Similar mineralized material was also found in the Rochelieu deposit area, but the structural control remains uncertain pending additional drilling. These sulphide bodies indicate that both high- and low-angle structures may have been active during the gold mineralization. Finally, gold-bearing, north-northeast-striking, steeply southeast-dipping fracture zones cut across all faults and rock types in the deposit.

Figure 7-5: Lewis Area Section, Looking Northeasterly Showing Intrusive Rocks, Drill Holes (source Barrick)

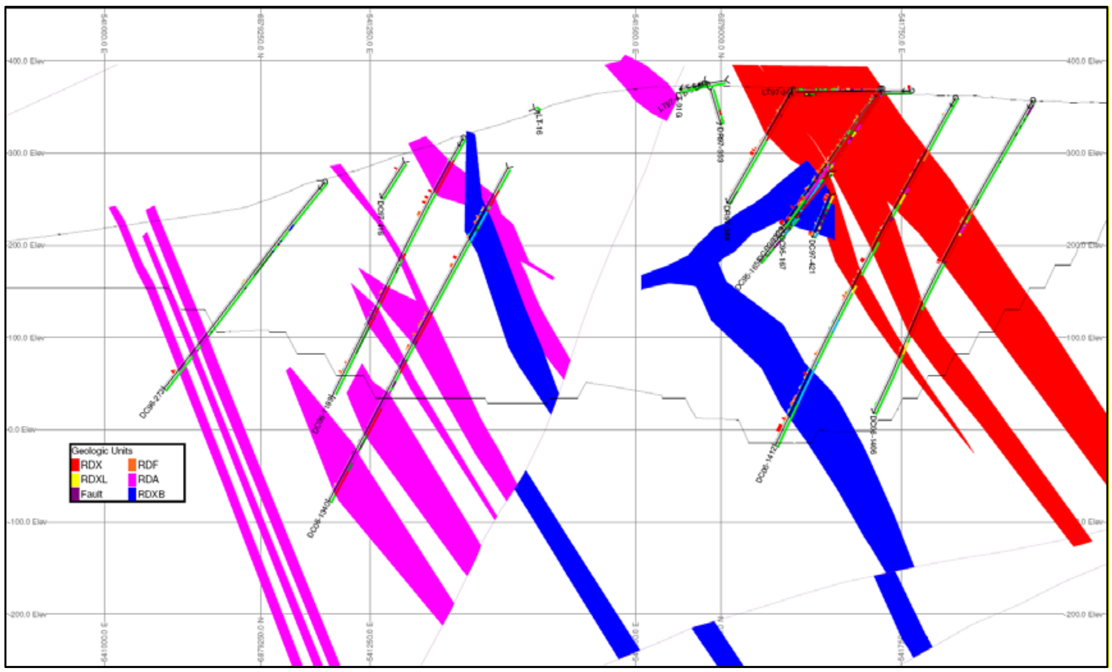
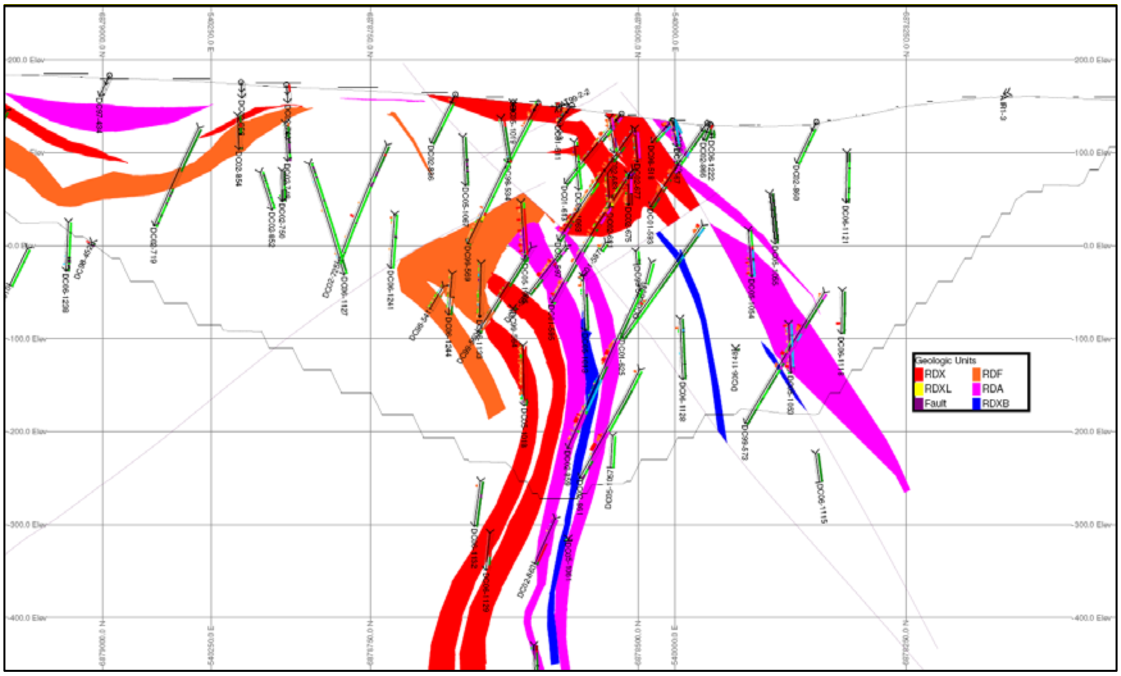


Figure 7-6: ACMA Area Section, Looking Southeasterly Showing Intrusive Rocks, Drill Holes (source Barrick)



8.0 DEPOSIT TYPES

Two distinct styles of gold-rich mineralization (Dome-Duqum style and ACMA-Lewis style) occur within the Donlin Creek trend. The Dome-Duqum mineralization, an early, high-temperature porphyry style, is characterized by fracture-controlled stockwork, laminated quartz-only veins containing varying proportions of copper, zinc, bismuth, silver, tellurium, selenium and local native gold mineralization. Silicification is locally associated with the veins. This style of mineralization occurs in the northern part of the property (Dome and Duqum prospects) (Figure 7.2). Contact metamorphism (hornfelsing) of the sedimentary rocks adjacent to host intrusives is common in areas containing this style of mineralization. Cross-cutting relationships were established in trench mapping during the 1999 field season that indicate the relative older age of the Dome mineralization.

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

9.0 MINERALIZATION

9.1.1 Mineralization and Alteration

Disseminated gold-bearing sulphides occur in the rhyodacite dikes and, to a lesser extent, in adjacent sediments. Structurally controlled mineralized veins formed within north-northeast-trending extensional fracture zones. These mineralized fracture zones are strongly developed where they intersect competent rock types such as felsic dykes and sills or massive greywacke. Quartz-carbonate-sulphide (pyrite, stibnite, and arsenopyrite) veins are the primary mineralized features, but gold also occurs in thin, discontinuous vein and fracture fillings. Veins seldom exceed 1 cm (0.4") wide; vein density can range up to 5 to 10 per metre; and vein zones vary from 2 to 35 m (6.5 to 115 ft) wide. Individual vein zones generally display limited lateral and vertical continuity. However, swarms of many anastomosing vein zones form larger mineralized corridors that display extensive lateral and depth continuity.

The ACMA–Lewis style of mineralization is consistent with a low-temperature, low-sulphidation, epithermal gold model involving a strongly reduced, CO_2-rich, weakly acidic, bisulphide-complexed, gold-bearing fluid. The deposit(s) is characterized by a gold-arsenic-antimony-mercury geochemical signature, sheeted quartz ± carbonate and sulphide veins, and disseminated sulphides. Common minerals observed in mineralized zones include pyrite, marcasite, arsenopyrite, stibnite, realgar, and native arsenic. Pyrite is the most common mineral and appears to be the earliest sulphide phase. It is ubiquitous in the rhyodacite and occurs as disseminated grains and micro-fracture fillings.

Disseminated pyrite in the sedimentary rocks occurs as fine to coarse grains (up to 5 mm across) preferentially concentrated near dyke/sill contacts or as syngenetic pyrite. Relative abundance of pyrite is not an indicator of gold grade.

Broad selvages of disseminated gold-bearing arsenopyrite and pyrite are found adjacent to veins and vein zones. Arsenopyrite commonly replaces pyrite and typically occurs as fine to very fine grains disseminated in intrusive rocks and as coarser aggregates in fractures and quartz-carbonate veins. In practice, fine-grained arsenopyrite can be difficult to distinguish from ubiquitous disseminated graphite.

Figure 9-1: ACMA Gold Distribution, Looking Southeasterly Showing Intrusive Rocks (colours) and Gold Grade Shells (>1 g/t stipple, >3 g/t pattern) (source Barrick)

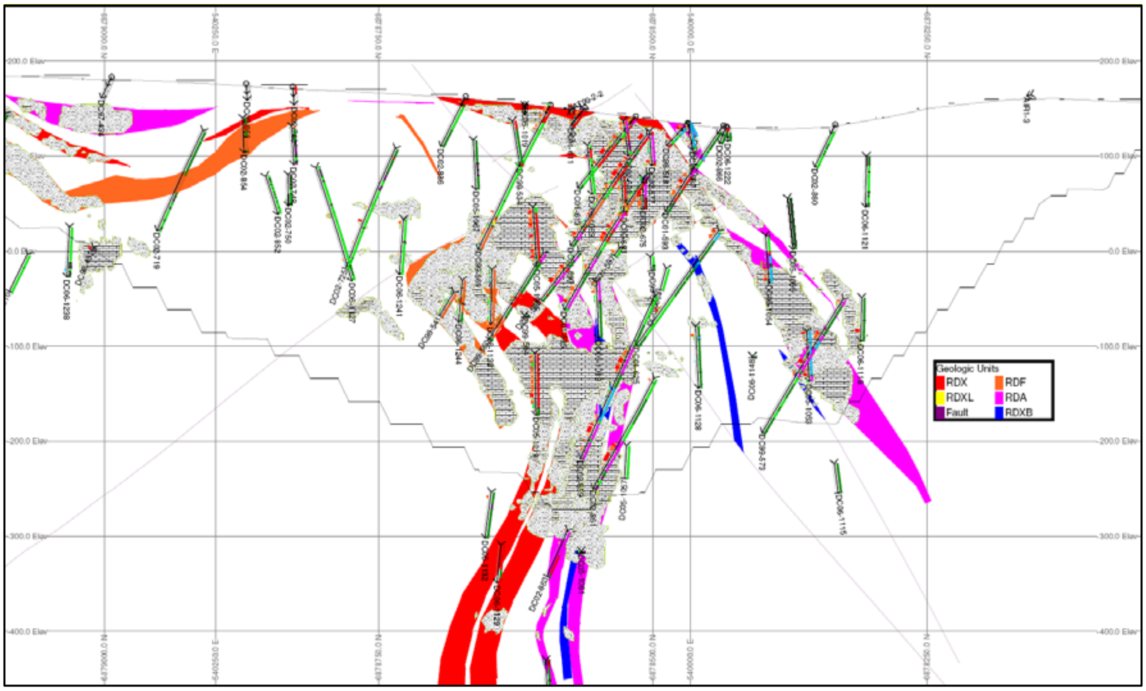
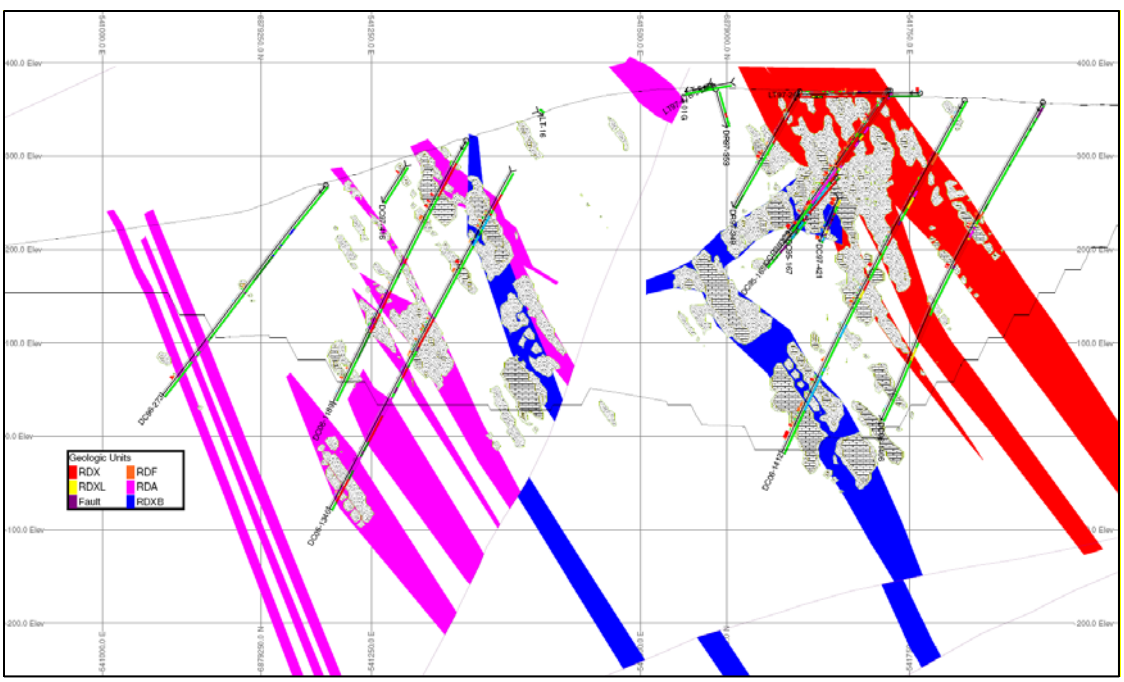


Figure 9-2: Lewis Gold Distribution Looking Northeasterly Showing Intrusive Rocks (colours) and Gold Grade Shells (>1 g/t stipple, >3 g/t pattern) (source Barrick)



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

Very rare native gold particles (1 to 20 µm) have been observed in process mineralogy studies of ACMA–Lewis area material, but most of the gold occurs in the crystal structure of arsenopyrite and, to a lesser extent, in pyrite. Fine-grained arsenopyrite (<20 µm diameter) contains 5 to 10 times more gold than coarse-grained arsenopyrite. Gold seen in polished sections occurs as 1 to 3 µm blebs with no clear paragenetic relationship to other minerals. Stibnite, realgar, and native arsenic are often associated with higher gold grades but contain very minor gold compared to arsenopyrite.

9.1.2 Vein Types

Multiple vein types apparently formed from a single hydrothermal fluid. Therefore, vein mineral assemblages show a continuum from pyrite through arsenopyrite, native arsenic, realgar, and orpiment, rather than discreet paragenetic stages. Stibnite is ubiquitous in all vein types but seems to increase in later vein stages. Gold grade and vein quartz generally increase from vein types V1 through V3 and then markedly decrease in V4, a carbonate-dominant vein type. Observed relationships in the vein assemblages are consistent with decreasing temperature, decreasing pH, and increasing fS_2 or fO_2 as a function of boiling or simple oxidation (fluid mixing).

The vein assemblages from earliest to latest are described below.

V1: The earliest veins are thin, discontinuous sulphide (>50%) veins with pyrite and trace arsenopyrite, little or no quartz (<30%) or carbonate (<50%), and a broad disseminated selvage of pyrite. They commonly contain minor amounts of ankerite in the veins, as well as ankerite with adjacent disseminated pyrite. The disseminated ankerite is pervasive in the rock matrix and also occurs as distinct overgrowths or rims around plagioclase phenocrysts. Such veins are typically low grade and show broad pervasive selvages of poorly crystalline illite alteration.

V2: Thin, discontinuous quartz (>30%) sulphide veins contain variable pyrite and arsenopyrite and may have broad, often pervasive selvages of fine-grained needle-like arsenopyrite. In many instances, a broad pyrite aureole surrounds the arsenopyrite selvage. Open-space vuggy textures are common, and trace amounts of stibnite are found in some veins. These veins show moderate grade, strong illite alteration, and variable but overall decreased ankerite content.

V3a: The highest grade veins are thicker, more continuous, open-space quartz veins with pyrite, arsenopyrite, native arsenic, and variable amounts of stibnite. These veins commonly show broad arsenopyrite-rich selvages with little to no ankerite. In some instances, minor amounts of calcite are present in the veins.

V3b: These are thick, continuous quartz veins with open-space textures and complex mineralogy, including pyrite, arsenopyrite, stibnite, native arsenic, realgar, orpiment, and trace sphalerite in intensely illite altered material. Realgar sometimes fills adjacent feldspar sites, creating a “pumpkin patch” texture in the wall rock. Gold grades are commonly very high.

V4: The latest vein phase consists of broadly oriented carbonate-quartz (>50% and <50%, respectively) vein sets. This set has the lowest gold grade, is common in the sedimentary rocks, and seems to form a halo around mineralized zones. The carbonate halo is zoned outward from ankerite to ferroan dolomite to calcite.

9.1.3 Alteration

Alteration mineral relationships appear to record decreasing pH due to boiling, falling temperature, liberated SiO_2, and the effective removal of Fe, Ca, and Na from the system. The following proximal to distal silicate alteration zones and carbonate and graphite alteration products are associated with the ACMA-Lewis hydrothermal system. Silica is largely restricted to veins and is not an important wall rock alteration product.

Illite Zone

Intrusive rocks within the mineralized corridors typically display pervasive ammonia-illite alteration of the feldspars. Intense and more crystalline illite often occurs with higher-grade material. Minor amounts of higher temperature dickite are found in some of the strong illite zones. Minor smectite with locally very high grade gold is also occasionally found in strong illite zones.

Illite–Kaolinite Zone

Broad haloes of admixed illite and kaolinite surround the illite-rich mineralized corridors and are generally very low-grade or barren.

Smectite Zone

Minor smectite as beidellite with minor ankerite and zeoloite in texturally destructive zones appears to occur distally within the system and probably records retrograde alteration during cooling and fluid mixing.

Carbonate

Carbonate veins appear to form an ankerite halo, often in the sedimentary package surrounding the mineralized intrusive phases. Ankerite is consistent with a reduced CO_2-rich vapour phase evolved during boiling and forming a distal oxidized ankerite/kaolinite halo. The ankerite would serve as a trap for calcium liberated from illite alteration of plagioclase and for iron liberated from illite alteration of biotite and the pyrite-to-arsenopyrite phase transition.

Graphite

Very fine-grained graphite is frequently found in igneous units in open spaces or high-porosity areas, often with coarse illite or as isolated, shred-like fragments and grains. Graphite is consistent with an early proximal alteration product prior to the onset of boiling.

Silica

Significant pervasive wall rock silicification does not exist in the ACMA–Lewis gold deposits. However, vein relationships show an increase in quartz content from early sulphide-dominant veins to late silica-dominant veins. This probably reflects a simple thermal gradient culminating in the distal, massive quartz/stibnite veins seen around the margins of the district and shown to have cooler fluid inclusion formation temperatures. Silica is made available through the majority of hydration reactions involved in alteration of the original potassium feldspar, plagioclase, and biotite. Kaolinite and smectite reactions consume available silica; therefore, no significant silica veins are present with these assemblages.

9.1.4 Minor Elements and Deleterious Materials

The most abundant minor elements associated with gold-bearing material are iron (Fe), arsenic (As), antimony (Sb), and sulphur (S). They are contained primarily in the mineral suite associated paragenetically or spatially with hydrothermal deposition of gold, including pyrite (FeS_2), arsenopyrite (FeAsS), orpiment (As_2S_3), realgar (AsS), and 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 ($Cu_{12}Sb_4S_{13}$), 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. Very minor Ni in the secondary sulphide mineral millerite (NiS), and minor Co in various secondary minerals have been observed in sedimentary rocks. The Ni and Co probably have a sedimentary origin.

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

Most of the Hg is contained in pyrite followed by marcasite and stibnite. Very low level Hg was also detected in arsenopyrite and associated with realgar. Primary Hg minerals such as cinnabar (HgS) are absent or exceedingly rare. Mercury can also be accommodated in the crystal structures of other minerals such as tetrahedrite and sphalerite, but it is not known whether these accessory minerals contain Hg at Donlin. Native gold in the general Donlin Creek area is also known to have relatively high concentrations of Hg compared to other gold occurrences in the region.

Process mineralogy studies show that muscovite ($KAl_2(AlSi_3O_{10})(OH)_2$) and apatite ($Ca_5(PO_4)_3(F,Cl,OH)$) are the principle sources of Cl and F and that the relatively more abundant muscovite accounts for most of the Cl and F. Muscovite is normally a rock forming mineral but it can also form during hydrothermal alteration along with structurally similar alteration products (illite) known to be associated with gold-bearing rocks in the deposit. Apatite is commonly found as an accessory mineral in intrusive and sedimentary rocks and as a hydrothermal alteration or vein mineral.

Graphitic carbon (C) is relatively abundant in the sediments and variably disseminated in the intrusive rocks as a possible alteration product. Carbonate minerals occur as both pervasive, fine-grained hydrothermal alteration products, often intergrown with fine disseminated sulphide, and also in carbonate and quartz-carbonate ± sulphide veins. They include ankerite ($CaFe(CO_3)_2$), dolomite ($CaMg(CO_3)_2$), calcite ($CaCO_3$), and very minor siderite ($Fe(CO_3)$).

9.1.5 Structural Controls on Mineralization

A detailed report on the geology and interpretation of the Donlin Creek gold deposit was completed by Piekenbrock and Petsel (2003) and forms the basis of the following discussion.

Mineralization was structurally controlled along north-northeast-trending fault/fracture zones and mineralized zones are best developed where they intersect favourable rock types such as felsic intrusive dykes and sills and greywacke.

The orientation of the mineralized zones is consistently sub-parallel to main $\delta 1$ axis (N24E) of the D2 compressive structural regime. Veins in the ACMA-Lewis resource transition through a continuum of changing mineralogy and increasing grade while maintaining a generally consistent north-northeast-strike and southeast-dip. The last phase of veining consists of a more dispersedly oriented carbonate-quartz vein set (V4) and has the lowest gold grade of all the vein types.

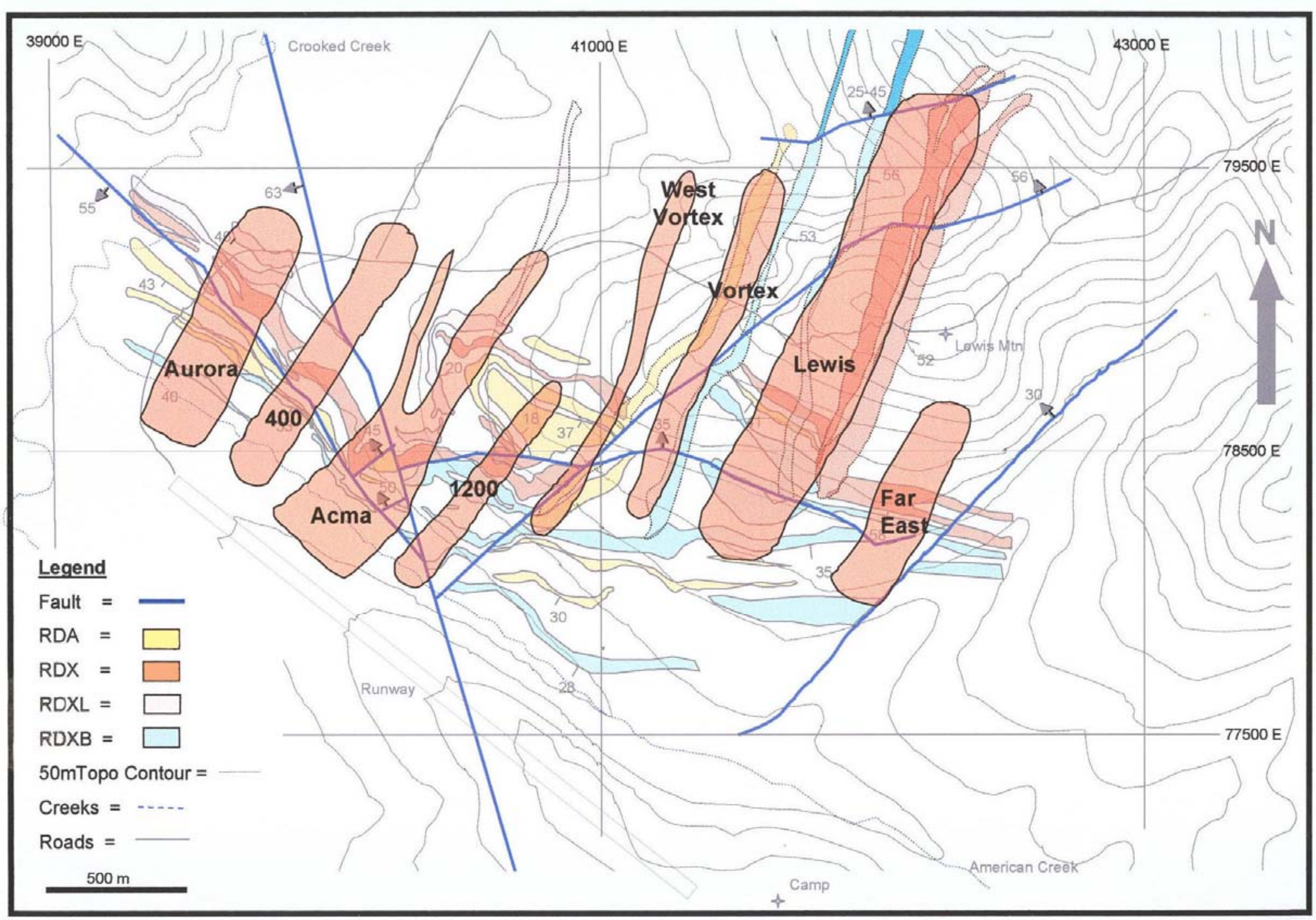
Structural orientations were collected from drill core using the clay impression method and recorded in right-hand rule conventions. At the time of the 2003 study, a total of 12,799 oriented data points were available for various structures in both trenches and drill holes. Table 9-1 summarizes the results of stereonet and scatter plots of veins by resource domain. The data were limited to drill hole data within the resource area.

Table 9-1: Results of Stereonet and Scatter Plots of Veins by Resource Domain

Domain	Number of Data Points	Bulls Eye on Stereonet
All Veins	4,496	N 024/68
ACMA	904	N 022/74
Vortex Lower	33	N 009/57
Lewis	1,244	N 017/57
Vortex Upper	157	N 013/70
Akivik	706	N 030/70
Tortured	755	N 024/65
Wedge	86	N 021/69
400	453	N 039/73
AC Block	158	N 032/65

Vein orientation measurements within each domain do not deviate significantly from the dominant overall vein orientation of N 024/68. As such, this formed the basis for selection of the search ellipse orientation for grade estimation during the 2006 resource estimation.

Figure 9-3: Major Mineralized Corridors – Donlin Creek Resource Area (Piekenbrock and Petsel, 2003)



10.0 EXPLORATION

Three major exploration/drill programs were conducted within the resource area at Donlin Creek between 2002 and 2007: one by NovaGold (2002), the other by Placer Dome (2005) and the third by Barrick (2006 and 2007). Each program is discussed below.

10.1 2002 NovaGold

NovaGold exploration work in 2002 consisted of diamond core drilling totalling 39,092 m (128,255 ft) and RC drilling totalling 11,589 m (38,022 ft) (see Figure 4-3). Drilling was concentrated in the western part of the resource area centered on ACMA. The purpose of the program was to extend known mineralized zones and potentially define new zones. An RC drill was used primarily to explore for zones of new mineralization with follow-up core drilling to better define the zones. The program was successful in expanding mineralization beyond the ACMA and 400 areas resulting in the discovery of the Aurora and Akivik zones. The initial step-out drilling was done on a rough 100 m by 100 m grid (328 ft by 328 ft) with a subsequent 50 m by 50 m drill spacing (164 ft by 164 ft) dependent on initial results. The majority of core holes were drilled to the north and inclined at 60° from horizontal to better intersect both sill contacts and vein zones.

NovaGold also re-logged select pre-2001 core holes in order to develop an updated lithology and structure model. These data aided in planning the exploration drilling.

10.2 2005 Placer Dome

The Placer Dome exploration work in 2005 consisted of diamond core drilling totalling 24,596 m (80,696 ft) and RC drilling totalling 3,644 m (11,955 ft). The majority of the core drilling was done in the Lewis and ACMA areas in order to reduce drill hole spacing in select areas containing inferred resource. A rough 80 m by 60 m grid (262 ft by 197 ft) for the new in-fill holes was used. This spacing proved optimal using the existing drill holes. Core holes were drilled westerly with an average inclination of 60° from horizontal to best intersect vein zones.

10.3 2006–2007 Barrick

Primary exploration activities were core drilling contracted to Boart Longyear of Salt Lake City, Utah. The drilling grid has been reduced to 25 m to 35 m centers (82 ft to 115 ft). The exploration results compared favourably to the pre-existing geologic and gold grade estimation model.

11.0 DRILLING

11.1 2002 NovaGold

NovaGold completed drilling in ACMA and adjacent mineralized areas of the Donlin Creek project in 2002 using two types of drills. Core drilling totalled 39,092 m (128,255 ft) in 194 drill holes and RC drilling totalled 11,589 m (38,022 ft) in 147 holes.

Core holes ranged in length from 17 m to 572 m (56 ft to 1,877 ft), averaging 201.5 m (661 ft). Drilling was done by wireline method using H-size equipment (HQ). Four core drill rigs were used. Drilling was well supervised, the sites were clean and safe, and work was efficiently conducted.

Holes were primarily drilled at a declination of between 60° and 70°. Down-hole surveys were taken about every 30 m (98 ft) using a reflex camera.

The RC holes ranged in length from 30.5 m to 140 m (100 ft to 459 ft), averaging 79 m (259 ft). One RC drill rig was used. The RC drilling was well supervised, the sites were clean and safe, and work was efficiently conducted.

Drill hole collars were located respective to a property grid. Proposed drill hole collars were located using a Garmin GPS. Final and completed collars were surveyed with an Ashtech GPS utilizing post-processing software for ±0.1 m (±0.3 ft) accuracy. Coordinates were given in the UTM coordinate system.

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 2 to 4 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 consist of type of structure, measurements relative to core axis and oriented core measurements, if applicable. The Geotechnical table records percent recovery and RQD for the entire hole, and fracture intensity where warranted. The protocols and coding are similar to those used by Placer Dome during its drilling campaigns.

In the fall of 2001, a preliminary study of alteration variability was undertaken on hand samples using a PIMA short-wave infrared spectrometer (SWIR). Based on those results, a PIMA was again utilized in early 2002 to ascertain alteration mineralogy in relation to detailed logging observations, as well as assay and geochemical results.

That study successfully demonstrated that SWIR spectrometry was efficient in defining alteration assemblages controlling the distribution of gold grade.

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

Drill core was well handled and maintained. Data collection was competently done and found to be consistent from hole to hole and between different loggers. Core recovery in the intrusive units, both where mineralized and unmineralized, was excellent, usually mid 90s to 100%. Recovery in the shale dominant sediments was more variable, ranging from 80s to high 90s. Overall, the 2002 drill program and data capture were conducted in a competent manner.

11.2 2005 Placer Dome

The 2005 Placer Dome drilling program at Donlin Creek utilized both core and RC drills. Core drilling totalled 24,596 m in 90 drill holes whereas RC drilling totalled 3,644 m in 30 holes.

Core drilling focussed on in-fill drilling primarily in the ACMA and Lewis areas. Core holes ranged in length from 79 m to 544 m, averaging 273 m. Drilling was done by wireline method using H-size equipment (HQ). Three core drill rigs were used. Most holes were drilled at a declination of between 50° and 60°.

The RC holes ranged in length from 102 m to 201 m, averaging 121.5 m. As in 2002, both drill programs were well supervised, the sites were clean and safe, and work was efficiently done.

Down-hole and collar survey methods, logging conventions and data entry procedures were the same as used in 2002. Alteration data were again collected using an ASD SWIR spectrometer. Core recovery was excellent. Overall, the 2005 drill program and data capture were conducted in a competent manner.

11.3 2006–2007 Barrick Drilling

2006

In 2006 the project team drilled 92,804 m (304,475 ft) of core with eight LF-70 drill rigs in 327 drill holes. Of that, 235 holes totalling 84,800 m (278,215 ft) of core were focused on converting inferred resource to measured and indicated resource. However, significant drilling was also devoted to a broad range of pre-feasibility and feasibility objectives, including pit slope stability, metallurgy, waste rock studies, facilities condemnation and engineering, and calcium carbonate resource bulk sampling, delineation, and exploration. Drilling is summarized in Table 11-1. Barrick continued the same logging procedures and down-hole and collar survey methods as used in the past. However, ACE core orientation tools were utilized for oriented holes and the data were entered into an acQuire database. Core recovery in both mineralized and unmineralized rocks was consistently excellent and generally exceeded 90% in intrusive rocks and 80 to >95% in sedimentary rocks. True widths of mineralization are difficult to determine but drill holes were oriented to reflect true width intercepts limited only by the equipment and geologic knowledge.

Table 11-1: Core Holes Drilled in 2006

Objective	Number	Feet (ft)	Metres (m)
Deep ACMA exploration	1	2,896	883
Infill resource conversion	223	259,871	79,229
Near-pit exploration (Akivik)	2	1,469	448
Waste dump condemnation	1	2,414	736
Waste (ABA)	8	10,411	3,174
Geotechnical	38	13,051	3,979
Port road geotechnical	8	768	234
Carbonate bulk sample (PQ)	14	1,036	316
Carbonate resource definition	20	6,891	2,101
Carbonate exploration	2	836	255
Metallurgical studies (PQ)	10	4,753	1,449
Total Core Holes	**327**	**304,397**	**92,804**

2007

Through October 3, 2007, 124 exploration holes comprising 50,562 m of core (165,886 ft) were completed. Core recovery in both mineralized and unmineralized rocks was consistently excellent and generally exceeded 90% in intrusive rocks and 80 to >95% in sedimentary rocks. True widths of mineralization are difficult to determine but drill holes were oriented to reflect true width intercepts limited only by the equipment and geologic knowledge.

11.4 Orientation of Mineralization

Figure 9-3 illustrates the orientation of the generally steeply dipping mineralized corridors. Gold mineralization is primarily found at the intersection of the structural corridors with intrusive rock types.

12.0 SAMPLING METHOD AND APPROACH

12.1.1 Logging

Standard logging conventions adopted by Barrick were used to capture geologic data from both core and RC chips. The chips were logged on paper forms and the data entered into an electronic database. Core logging data were captured in five tables: lithology, mineralization, alteration (visual), structural, and geotechnical (percent recovery and rock quality designation, or “RQD”). The logging manual is found in Appendix A.

In the fall of 2001, a preliminary study of alteration assemblages in intrusive rocks was conducted on hand specimens using a PIMA short-wave infrared spectrometer (SWIR). That study successfully demonstrated that alteration assemblages could be efficiently identified with SWIR spectrometry. A more serviceable, high-throughput analytic spectral device (ASD) SWIR was used in 2002 to collect alteration data for the entire Donlin Creek resource area. Virtually all core holes from the ACMA and Akivik areas, as well as a significant portion from south Lewis, were analyzed during the 2002 field season. All intrusive rock intervals drilled through 2007 were measured with the ASD SWIR.

12.1.2 Sampling

The sampling protocol for the 2006 and 2007 programs was similar to that followed by Placer Dome and NovaGold in previous campaigns.

Holes are sampled from the top of bedrock to the end of the hole. Overburden, excluding the organic layer, may also be sampled if abnormally thick and composed of abundant rock clasts. Geologists mark 2 m (6.6 ft) sample intervals through mineralized rock and variable width (up to 9.15 m or 30 ft) buffers of non-mineralized material adjacent to mineralized zones. Weakly mineralized to barren sedimentary rock can be marked on 3 m (9.8 ft) intervals at the discretion of the geologist. This sample interval approach is reliable and appropriate, and is used throughout the mining industry for this style of gold deposit. Sample intervals are broken at rock type contacts. 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 that uses water-cooled diamond saw blades. Core cutters orient the core in the saw to ensure a representative split. One-half of the core is returned to the core box for storage at site, and the other half is bagged for sample processing. In December and January (2007), a total of approximately 12,000 m (39,360 ft) 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.

13.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

13.1 Drill Hole Sample Preparation

13.1.1 Prior to 2006

Sample preparation, quality assurance and quality control for assays prior to 2002 were analyzed by AMEC and reported in their March 2002 Technical Report (Juras and Hodgson, 2002). Samples drilled in 2002 were analyzed by AMEC and reported in the first half of 2003. AMEC determined the assays and the database were suitable for resource estimation. No sample collection occurred within the mineral resource areas between 2003 and the beginning of 2005.

13.1.2 2006–2007

Most of the core samples were crushed at the Donlin camp facility. Samples of core split in Anchorage were shipped to an ALS Chemex preparation lab for crushing, pulverizing and assaying.

The Donlin camp preparation lab is housed in a heated steel building. A partition separates the core cutting area from the crushing area. The crushing side of the operation is equipped with a dust control ventilation system to minimize contamination. The camp sample preparation procedure consists of the following steps:

- The entire bagged sample is dried in an oven heated to between 85°C and 90°C for 12 hours.
- The sample is put into trays for processing through a jaw crusher. The sample tag stays with the sample.
- Blank samples (one of three QA/QC control samples) are inserted into the sample stream.
- The sample is crushed until the end product passes 70% minus 2 mm (10 mesh). Sieve analyses are performed periodically to check crush quality, and the crusher jaws are adjusted as necessary. Blank material is periodically used to clean the crushers, and operators are instructed to increase the cleaning frequency when unusually sulphide-rich material is processed.
- Sample is then passed through a riffle splitter four to six times to obtain a nominal 250 g (9 oz) 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—are inserted as specified on the cutting list prepared by the geologist. Control samples including SRM, duplicates and blank samples are included in every batch of 20. The blank is prepared by processing a sample from a bin of gravel-size crushed rock by passing it through the jaw crusher and riffle-splitting it to ~200 g (7 oz). When a duplicate is required, the crushed core sample is passed through the riffle splitter once, and each half is split repeatedly to obtain a ~200 g (7 oz) sample.

13.2 Sample Analysis

Final sample preparation and chemical analysis for gold, sulphur and trace element suites were completed at ALS Chemex in Vancouver, an ISO9001:2000 accredited laboratory. The preparation consists of the following:

- The splits of crushed core were reduced to rock flour or "pulp" in a ring-and-puck grinding mill (to better than 85% passing minus 75 µm or 200 mesh).
- A 1 oz (30 g) sub-sample of the pulp was fire assayed primarily using a fire assay-atomic absorption spectroscopy (AAS) method. Prior to 2007, the primary analytical method was Au-AA23 with detection range of 0.005 to 10 g/t gold. In 2007 analytical methods switched primarily to Au-AA25 with a detection range of 0.01 to 100 g/t gold to take advantage of the higher top detection range.
- Samples that assayed >10 g Au/t were re-assayed with a fire assay-gravimetric method (NovaGold, 2002) or from 2005 through 2006 by "ore grade" fire assay-AAS method (Au-AA25) with a detection range of 0.01 to 100 g/t gold. In 2007, one sample exceeding 100 g/t was assayed via gravimetric method Au-GRA21 with a detection range of 0.05 to 1000 g/t gold.
- Significant drill hole composites exceeding 4 g/t gold received through October 3, 2007 (the assay cut-off date for the DC7a block model) are tabulated in Appendix B.

13.3 Donlin Security and Sample Transport

13.3.1 Project Site

Core samples are transported from the field and are brought to the yard adjacent to the geology office and logging tents at the end of each drill shift. Core storage is secure because Donlin is a remote camp and access is strictly controlled. Unauthorized camp personnel have generally been excluded from the core cutting and sample preparation building, but strict access procedures were initiated following an audit by Barrick of the sample preparation lab in mid-2006 (Heberlein, 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 lab 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 lab returns a custody form that reports on the condition of security seals.

13.3.2 Anchorage Security and Sample Transportation

The Anchorage logging and splitting facility was housed in a secure, dedicated, warehouse/office facility. Visitor access to the facility was strictly controlled by AES, the facility manager. Outside visitation for tours or purposes other than daily delivery or pick-up required advance approval by the Donlin Project Manager. Whole core shipped from camp to the facility was transported on Lynden Air Cargo. Lynden waybills and Barrick custody forms were used to track samples from camp to Lynden's Anchorage airport facility and from there by Lynden trucks to the Anchorage logging facility. Split core samples shipped from camp to the ALS Chemex Fairbanks lab followed similar protocol. Bagged split core samples were tied into shipping bags and loaded into palletized supersacks. Supersacks were closed with numbered security seals and shipped on Lynden trucks to ALS Chemex in Fairbanks. Waybills aided tracking within the Lynden transport system, and ALS Chemex reported on the condition of security seals in the same manner as shipments from camp.

13.4 Assay Quality Assurance and Quality Control (QA/QC)

13.4.1 1995–2002 QA/QC Protocol

Beginning with Placer Dome in 1995 and continuing with NovaGold through 2002, a systematic and comprehensive program of QA/QC has been employed for rock sampling and definition drilling programs at Donlin Creek. The QA/QC protocols include the random and blind insertion of the following:

- standard reference materials (SRMs) to monitor the accuracy of lab results
- coarse reject duplicates to monitor analytical precision
- blank control samples to monitor contamination during sample preparation and analysis.

From 1996 to 2002, SRMs were inserted at an average rate of one per 24 samples. Over the same period, 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.

Based on the results obtained from the comprehensive QA/QC programs at Donlin Creek, AMEC concluded in 2003 that the quality of the Donlin Creek drill database was sufficient for resource estimation. A comprehensive discussion of Donlin Creek sample QA/QC procedures and results can be found in the report, “Status Update for the Donlin Creek Pre-feasibility Study” (AMEC, 2003).

A slightly modified QA/QC protocol was implemented in 2005. Three QA/QC samples—one blank, one coarse reject duplicate, and one SRM—were randomly inserted into every block of 20 sample numbers. Thus, in every block of 20 sample numbers there were 17 drill hole samples and three QA/QC control samples.

The 20 sample blocks met two criteria:

- The blocks must begin with a sample number in which the penultimate digit is an EVEN number, and end with a sample number in which the penultimate digit is an EVEN number.
- The blocks must begin with a sample number that ends in 1.

Example: XXXXevennumber1 - XXXXevennumber0, such as XXX021-XXX040, XXX041-XXX060, etc.

13.4.2 Standard Reference Material and Blank Material in 2005-2006

Leftover SRM material (Std-C and Std-D) from the 2002 campaign was used at the beginning of the 2005 season. When these SRMs were depleted, additional reference material was purchased from Analytical Solutions (OREAS 6Pb and OREAS 7Pb) and CDN Laboratories (CDN-GS-3). After the 2005 season, two additional standards (Std-G and Std-H) were created from Donlin Creek coarse reject material. These two new standards, in addition to CDN-GS-3, were used during the 2006 season. The new standards created from coarse reject material were certified by Barrick's chief geochemist after industry accepted round robin assay and statistical analyses.

Blank material consisting of washed river gravel was purchased from Anchorage Sand and Gravel for the 2005 season and the beginning of the 2006 season. In early to mid-2006, the blank material was changed to granite chips purchased from Lowe's in Anchorage.

13.4.3 QA/QC 2005 Results

Lab performance was checked by continuously monitoring control sample assays. Assay batches containing a control sample assay that exceeded two standard deviations from the accepted value were immediately re-assayed, and the original assays were replaced with assays that satisfied the QA/QC standard. An exception to this protocol was made for batches containing negligible gold values.

All control samples functioned well. SRM monitoring kept assay accuracy within acceptable limits. Duplicate sample data indicated no bias in sample preparation and analytical precision. Blank assay data demonstrated negligible contamination in the sample preparation and assay processes.

13.4.4 QA/QC 2006 and 2007 Results

The QA/QC protocol was identical to that used in 2005, except that the frequency of control samples was increased from 4% to 6%. No serious systematic preparation or analytical lab problems were detected.

Blanks

Two types of blank material were used, a product from Anchorage Sand and Gravel for the beginning of the 2006 season followed by granite chips purchased from Lowe's for later in 2006 and 2007. Each blank type was analyzed for gold by two different assay protocols, Au-AA23 and Au-AA25. A threshold of three times the gold detection limit was used by NovaGold for failure analysis. Dividing the samples into two time frames, a total of 5 samples (0.1%) failed before September 2006 and 163 samples (4.6%) of blanks failed after September 2006. NovaGold recommends follow up to determine the suitability of the granite chips as blank material or whether a change in sample preparation has been the cause of the change in failure rate. The failures were all much less than the proposed range of gold cut-off grades and, in NovaGold's opinion, the failures are not material.

Duplicates

A total of 3,885 duplicate pairs obtained from coarse reject material were taken at Donlin Creek during 2006 and 2007. Successful or "passing" duplicates were identified by calculating the Absolute Relative Difference (ARD):

*ARD = [A-B] / 0.5 * (A + B)*

A criterion of ARD must be less than 0.2 for 90% of the pairs used by NovaGold. The percent passing was an acceptable 93%.

Standards

A total of 3,573 results from 15 different standard reference materials (SRM) were provided by Barrick. Barrick set acceptable limits for all standards at the best value +/- 2 standard deviations. Failed standards were to be re-assayed until the results were acceptable or validated by duplication. NovaGold was not provided with the information necessary to recalculate best values so our results are indicative.

A total of 123 standards exceeded the range, and 58 (47%) were flagged with a comment regarding the failure. These comments commonly referred to a standard swap, or that the result failed by a calculated percentage but was "within acceptable limits". The definition of acceptable limits should become a defined term in Donlin Creek LLC procedure manuals. Donlin Creek LLC is aware of the remaining uncommented 65 failed standards and will be addressing the issue shortly. NovaGold's opinion is that the standards have been properly monitored and decisions regarding re-assay of failures have been reasonable.

The details of the QA/QC analysis are included in Appendix C.

13.5 Specific Gravity Determinations

Historically, only two specific gravity (SG) values were used in tonnage calculations: 2.65 g/cm^3 for the intrusive units and 2.71 g/cm^3 for the sediment units. Additional SG measurements were collected in 2006 to provide better coverage of deposit rock units and geographic subregions. Statistical evaluation of these new SG values shows that they are quite similar to the historical intrusive rock and sedimentary rock SG values. Therefore, the historical values were used for this mineral resource estimation.

The following methodology is used to determine SG:

- Samples of whole core approximately 5 to 10 cm (2 to 4") in length are first weighed dry and then weighed in water. This is accomplished by removing the dry weighing tray assembly from the scale, placing the sample in the wire basket, hooking the basket to the scale, and submerging the sample in a five-gallon bucket of water. A small sample bag containing rock chips placed in a small basket midway along the wire assembly acts as a tare weight to compensate for removing the weighing tray. This makes it easier to return the scale to zero when changing from dry to wet measurements.

- The formula for SG calculation is: Weight in Air/(Weight in Air – Weight in Water). The specific gravities are computed in acQuire once the weights are entered into the database.
- Measurements are collected for all rock types at a minimum frequency of one sample from all logged rock type intervals and one sample every 15 to 20 m (49.2 to 65.6 ft) in the larger rock units. Mineralized rock takes precedence over unmineralized rock in a given rock type interval, but sufficient measurements of unmineralized material are also collected to document potential variability.

14.0 DATA VERIFICATION

14.1 Prior to 2005 Campaign

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 checked several ways. AMEC concluded that the assay and survey database used for the Donlin Creek mineral resource estimation was sufficiently free of error to be adequate for resource estimation.

14.2 2005 Campaign

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

NovaGold verified the integrity of the 2005 data and it is sufficiently free of error to be adequate for resource estimation.

14.3 2006 and 2007 Drilling Campaigns

Drilling data were captured using acQuire software and stored in MS SQL Server. Geologic logs, collar and down-hole survey data were entered on site 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 in the middle of 2006, and about 50% of the 2006 assay data were imported directly into the master database in Anchorage.

The acQuire database was converted from the standard acQuire data-model to the more robust acQuire “Corp” data-model to aid in data verification. Further verification of legacy data took place by Barrick when data were migrated to the new data-model.

NovaGold verified the drill hole data in the following manner:

Collar Surveys

Collar survey information is transferred electronically from the electronic Ashtech survey instrument to the database, minimizing the chance of input error. 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. Strangely, the error rate against the geologic logs was much higher; most collar coordinates were off by several metres. NovaGold does not know the reason but it is possible that the geologic logs have the proposed coordinate and not the final coordinate written on them, or there are transcription errors from the electronic database to the paper log. NovaGold is satisfied that the collar surveys from the Ashtech data files are sufficiently error free to be used for resource estimation.

Down-hole Surveys

Down-hole surveys are transcribed by hand from the survey instrument to paper survey forms. The forms in turn are entered into the electronic database manually. Ten percent of the drill holes were checked and an unacceptable error rate of 4.4 percent was measured. The primary error was that the down-hole survey was omitted from the electronic database. Other errors were incorrect azimuth conversion, incorrect feet to metre conversion, incorrect depth and incorrect priority code. NovaGold recommends that the Donlin Creek LLC 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 will be minimal.

Assays

Electronic assay certificates made available by Barrick were merged into a single file and by matching on sample number were directly compared to the electronic database. For 2006 drilling, 70 percent 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. Although an acceptable error rate, NovaGold recommends that the source of the discrepancies be identified. NovaGold believes that the assay database is sufficiently error free to be used for resource estimation.

15.0 ADJACENT PROPERTIES

Adjacent properties are not relevant for the review of the Donlin Creek project.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Key testwork programs on Donlin Creek ores have been conducted at a number of laboratories over a period of approximately seven years as directed by the project manager (primarily Barrick). Major programs at the bench-scale level were initiated in 2006 to test grinding, flotation, pressure oxidation (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. Both levels of testwork were conducted to optimize process parameters and develop engineering information for use in pre-feasibility studies. NovaGold assumes that the work was completed by, or under the supervision of, a qualified person.

The testing has shown that the ores require pre-treatment ahead of cyanidation to recover the gold. The preferred method of pre-treatment is POX of the sulphide concentrate produced from flotation. Overall gold recovery is estimated to be 89.8%, based on the combined life-of-mine (LOM) average of 92.9% recovery from flotation and 96.6% from POX of the concentrate.

The metallurgical testwork, key results and recommendations are summarized below. Table 16-1 contains a summary of metallurgical reports.

16.1 Mineralogy

Sulphur occurs primarily as pyrite and arsenopyrite. The pyrite contains only 10 to 20% of the gold, while arsenopyrite is the main carrier of gold in solid solution (sub-microscopic) form. The finest arsenopyrite has the highest gold grade.

Little or no visible gold has been identified from all mineralogical investigations.

Mercury (2 ppm), arsenic (2,800 ppm), antimony (80 to 90 ppm) and chloride (20 to 25 ppm) are present in the ore. Carbonate content is 3.3 to 3.4%. These elements all affect the process.

16.2 Direct Leach / CIL

The ore is refractory to direct and carbon-in-leach (CIL) cyanidation and yields very low recoveries (<15%) from either leaching methodology. High gold recovery is only achieved by destruction of the sulphidic host matrix before cyanidation.

Table 16-1: Summary of Metallurgical / Process Reports on Donlin Creek

Author	Report Date	Report Reference	Title
A. Lanfranco & Assoc.	2007 (Jun)	June 2007 Trial Report	Summary of Autoclave gas Testing Results
Air Pollution Testing	2006 (Oct)	APT Project BAR6243	Emission Testing Results of Autoclave
AMEC	2007 (May)	152628-001	Trade-off Study No. 1 – BIOX® Study
AMEC	2007 (May)	152628-002	Trade-off Study No. 2 – Comminution Circuit Study
AMEC	2007 (Apr)	152628-003	Trade-off Study No. 3 – Feasibility of Calcining CSS
AMEC	2007 (Apr)	152628-004	Trade-off Study No. 4 – De-Chlorination Treatment
AMEC	2007 (Apr)	152628-005	Trade-off Study No. 5 – AARL vs. Zadra Elution Processes
AMEC	2007 (Apr)	152628-006	Trade-off Study No. 6 – Roasting as an Alternation for Oxidation of Donlin Creek Flotation Concentrate
AMEC	2007 (Apr)	152628-007	Trade-off Study No. 7 – Pressure Oxidation (POX) of a Flotation Concentration vs. Pressure Oxidation of the Whole Ore (WOP)
AMEC	2007 (Jun)	Project No. 155096	Donlin Creek Gold Project Metallurgical Model Report
AMEC	2007 (Sep)	Project No. 155096	Review of 2006 Flotation Testwork
AMEC	2007 (Sep)	Project No. 155096	AMEC, Coarse Ore Storage: Covered Stockpile vs. Concrete Silos
AMEC	2007 (Sep)	Project No. 155096	AMEC, MF2 vs. Conventional Grinding Circuit
AMEC	2007 (Sep)	Project No. 155096	AMEC, Limestone Report
AMEC	2007 (Sep)	Project No. 155096	AMEC, Flat Bottomed Thickeners
Amtel	2004 (Aug)	Report 04-24	Gold Deportment in ACMA Composite of Donlin Creek
Amtel	2005 (Feb)	Report 05-01	Flotation of Au-Rich Arsenopyrite from Donlin Creek Ore
Amtel	2006 (Feb)	Report 06-01	Testwork on Donlin Creek Ores Using Novel Flotation Scheme
Amtel	2006 (Nov)	Report 07-03	Process Mineralogy of Donlin Creek Ores – Gold Deportment in PP Testwork Products
Amtel	2007 (Jul)	Report 07-04	Process Mineralogy of Donlin Creek Ores – Deleterious Elements in PP Testwork Products
Amtel	2007 (Jul)	Report 07-17	Variability Mineralogy of Donlin Creek Ores
Amtel	2007 (May)	Report 07-26	QA/QC on Sulphate Assaying of Donlin Creek Flotation Concentrates
Amtel	2007 (Jul	Report 07-28	Chlorine and Fluorine Assay Data of Donlin Creek Ores
Amtel	1996 (Nov)	Report 96-11	Deportment of Gold in the Donlin Creek Ores
Barrick Corporate	2007 (Sep)	N/A	2005 ACMA Intrusive Composite
Barrick Corporate	2007 (Sep)	N/A	2005 Lewis Intrusive Composite
Barrick Corporate	2007 (Sep)	N/A	2005 Sediment Composite
Barrick Corporate	2007 (Sep)	N/A	2006 Dec SGS PP Composite
Barrick Corporate	2007 (Sep)	N/A	2007 Jan SGS PP Composite
Barrick Corporate	2007 (Sep)	N/A	2007 Flotation Variability Samples (x98)
Barrick Corporate	2007 (Sep)	N/A	2006 Flotation Variability Samples (x12)
Barrick Corporate	2007 (Sep)	N/A	2006 Grinding Variability Samples (x155)
Barrick Corporate	2007 (Sep)	N/A	2007 Grinding Variability Samples (x149)
Barrick Corporate	2007 (Sep)	N/A	2006 Grinding PQ Core Variability Samples (x9)

Author	Report Date	Report Reference	Title
Barrick Corporate	2008 (Sep)	N/A	2006 & 2007 Metallurgy Sample Plots
Barrick Technology Centre	2007 (Apr)	59001-01	Donlin Creek Pre-feasibility Study Pressure Oxidation Pilot-plant Results
Barrick Technology Centre	2007 (Apr)	59001-02	Donlin Creek Batch Pressure Oxidation Tests Summary : Pre-feasibility Study
Barrick Technology Centre	2007 (Apr)	59001-03	Donlin Creek Feasibility Pressure Oxidation Piloting – Phase 1
Barrick Technology Centre	2007 (May)	59001-04	Donlin Creek Pre-feasibility Neutralization Piloting
Barrick Technology Centre	2007 (Mar)	59001-05	Summary of Donlin Creek Feasibility Study Batch Neutralization Test Results
Barrick Technology Centre	2007 (Apr)	59001-06	Pilot-plant CIL Cyanidation of Donlin Creek Concentrate After Pressure Oxidation
Barrick Technology Centre	2007 (May)	59001-07	Donlin Creek Feasibility Pressure Oxidation Product CCD and Neutralization Piloting – Phase 1
Barrick Technology Centre	2007 (Aug)	59001-08	Donlin Creek Feasibility Pressure Oxidation Piloting – Phase 2
Barrick Technology Centre	2007 (Sep)	59001-09	Donlin Creek Batch Pressure Oxidation Summary – Feasibility Study
Barrick Technology Centre	2007 (Aug)	59001-10	Donlin Creek Feasibility Phase 2, Pre-Acidification and Neutralization Bench Test Results
Barrick Technology Centre	2007 (Aug)	59001-11	Donlin Creek Feasibility Phase 2, Neutralization Pilot-plant
Barrick Technology Centre	2007 (Aug)	59001-12	Donlin Creek Feasibility Miscellaneous Testwork (CIL, Detoxification, Mercury Precipitation)
Barrick Technology Centre	2007 (May)	Memorandum	Confirmatory Cyanide Detoxification Tests on Donlin Creek CIL Tails from Feasibility Study
Barrick Technology Centre	2007 (Jul)	NA	Mineralogy on Concentrate, Pre-acid Conc and AC discharge DC
Canadian Light Source	2007 (Sep)	CLSI Contract #5025	X-Ray absorption Near-Edge Structures (Xanes) Spectroscopy on Barrick Tailings Samples
Delkor	2006 (Nov)	Test Report 069301	Delkor Testwork on Thickening & Filtration Applications, Donlin Creek Autoclave Feed
Dorr-Oliver Eimco	2004 (Oct)	NA	Report on Testing of Donlin Creek Tailings applying Deep Cone Paste Thickening for Tailings Disposal
Dorr-Oliver Eimco	2006 (Oct)	NA	Report on Testing for Barrick Gold Donlin Creek Project Sedimentation and Rheology Tests on Autoclave Feed and Stage 1 CCD Feed
Dorr-Oliver Eimco	2006 (Sep)	NA	Summary of Thickening Results Donlin Creek Concentrates
Dorr-Oliver Eimco	2007 (Apr)	NA	Sedimentation and Rheology Tests on Flotation Concentrate,

Author	Report Date	Report Reference	Title
			Neutralized Tailings and Flotation Tailings
Dorr-Oliver Eimco	2007 (Jul)	NA	Sedimentation and Rheology Tests PP5 and PP10 Combined Tailings
Dorr-Oliver Eimco	2007 (Aug)	NA	Summary of Thickening Results of Autoclave Hot Cure Slurry
Dorr-Oliver Eimco	2007 (Sep)	NA	Thickener Testing on Concentrate, AC Discharge and Float Tails
Dynatec Corporation	2004 (Nov)	217-RS-101-000-R0	Donlin Creek Project Concentrate Pressure Oxidation, Cyanidation and Neutralization Studies
Environmental Geochemistry Int Pty Ltd	2004 (Aug)	2509/649	Acid Forming Characteristics of Waste Rock and Low Grade Ore Samples from the Donlin Creek Project, Alaska
G&T Metallurgical	2004 (Jul)	KM 1523 / P0407	Production of Concentrates from Donlin Creek Ores
G&T Metallurgical	2006 (Feb)	KM 1708	A Preliminary Metallurgical Study Model and Flotation Test Work
G&T Metallurgical	2006 (Mar)	KM 1787	A Preliminary Assessment of Flotation Response
G&T Metallurgical	2006 (Aug)	KM 1806	A Preliminary Assessment of Flotation Performance
G&T Metallurgical	2006 (Oct)	KM 1852	The Flotation Response of Donlin Creek Mineralization – Pilot-plant Testing
G&T Metallurgical	2006 (Oct)	KM 1904	An Assessment of Metallurgical Response of the Blended Ore
G&T Metallurgical	2007 (Mar)	KM 1931	Metallurgical Assessment of Aging on Donlin Creek Ore
GeoBiotics	2006 (Mar)	N/A	Conceptual Capital and Operating Cost Estimates GeoBiotics' GEOCOAT Heap Biooxidation Technology for Barrick Gold Corporation Donlin Creek Project
GeoBiotics	2006 (May)	N/A	Preliminary Assessment of GeoBiotics' GEOLEACH™ Technology for Barrick Gold Corporation Donlin Creek Project
Gold Fields Ltd	2006 (Dec)	DCK OOM Rev-A.06	Order of Magnitude Capital Cost Estimate for BIOX Plant
Gold Fields Ltd	2007 (Aug)	GFL 097/06/07	Progress Report for Phase 2
Hatch	2007 (Aug)	H322685	MetSim Model Design Criteria
Hatch	2007 (Jul)	H322685-00-GMP-F-0001	DC Feasibility Study Autoclave Design / Residence Time Control
Hatch	2007 (Aug)	H322685-00-GMP-F-0002	Pressure Oxidation and Carbon in Leach Gold Recovery
Hatch	2007 (Jul)	H-322685-00-GRP-F-0001	Mercury Reduction Technology Assessment
Hatch	2007 (Sep)	H322685-00-GRP-N-0002-Rev 0	Hatch Area 17 and 33 Report
Hatch	2007 (Jul)	H322685-17-GMP-F-0001	Revised Decant Return Water Recycle Assessment
Hatch	2007 (Jul)	H322685-17-GMP-F-0002	Autoclave Off-Gas Mercury Removal Process Description
Hatch	2007 (Aug)	H322685-DC-F-0001_2_V4	Autoclave Design Criteria
Hatch	2006 (Nov)	H-322685-MP-0001-CA01	Autoclave Feed Pre-Acidulation Trade-Off Study
Hatch	2007 (Mar)	H322685-MP-CA01-1000	Factored Autoclave Capital Cost Estimate
Hatch	2006 (Nov)	H-322685-RPT-0007-CA01-R0	Autoclave Trade-Off Study One vs. Two Trains
Hatch	2007 (May)	H-322685-RPT-0007-CA01-R1	Autoclave Trade-Off Study One vs. Two Trains
Hatch	2006 (Dec)	H-322685-RPT-0008-CA01	Autoclave Sizing Comparison

Author	Report Date	Report Reference	Title
Hatch	2007 (Apr)	H322685-RPT-CA01-10001	Minor Species Dissolution Study
Hatch	2007 (Jul)	Rev 3, 27 July 2007	MetSim Feasibility Model (Summer, average)
Hatch	2007 (Jul)	Rev 3, 27 July 2007	MetSim Feasibility Model (Winter, average)
Hatch	2007 (Jul)	Rev 3, 27 July 2007	MetSim Feasibility Model (Summer, maximum)
Hatch	2007 (Jul)	Rev 3, 27 July 2007	MetSim Feasibility Models (Winter, maximum)
Hazen Research	2007 (Aug)	Hazen Project 10367-05	N2TEC Flotation of Donlin Creek Ore
Hazen Research	2006 (Aug)	Hazen Project 10367-06	Microscopic Examination of Flotation Concentrates – Memo
Hazen Research	2007 (Jan)	Hazen Project 10367-06	Re-flotation of Donlin Creek Pilot-plant Tails
Intec Ltd	2005 (Dec)	N/A	Scoping Leach Trials Via The Intec Gold Process Utilising CIL, Donlin Creek Project
JKMRC	2007 (Jun)	JKTech Job No. 06148	Flotation Circuit Analysis of the Donlin Creek Pilot Campaign (First)
JKMRC	2007 (Jul)	JKTech Job No. 06430	Flotation Circuit Analysis of the Donlin Creek Pilot Campaign (Second)
Jones, Hackl	1997 (Dec)	N/A	Continuous Bacterial Leaching and Cyanidation of a Donlin Creek Concentrate
Larox	2006 (Nov)	Test No. 15567T1	Filter Tested Slurry – Donlin Creek Low Grade Sulphide Concentrate
Minnovex	2005 (Jun)	NA	Review of Donlin Creek Flotation Testing
Mutis Liber (Mike Adams)	2007 (May)	Report No. 155	Donlin Creek Kiln and Gold Room Off-Gas Estimation Study
Orway Mineral Consultants	2006 (Aug)	Report No. 46343	Donlin Creek Comminution Circuit Option Study
Orway Mineral Consultants	2006 (Nov)	Report No. 6456	Donlin Creek Comminution Circuit Option Study Phase II
Outokumpu Technology	2007 (Jan)	N/A	Thickening of Acidified and Non-Acidified Flotation Concentrates
Outokumpu Technology	2006 (Jul)	TH-0379	High Rate Thickening Flotation Concentrate and Tailings Samples
Outokumpu Technology	2006 (Sep)	TH-0383	Thickening of Concentrate and Autoclave Discharge
Outokumpu Technology	2006 (Oct)	TH-0387	Thickening of Neutralization Discharge
Outokumpu Technology	2007 (Jun)	TH-0403	High Rate Thickening of Flotation Tailings
Outokumpu Technology	2007 (Apr)	TH-0407	High Rate Thickening of Flotation Tailings, Concentrate, Acidified Concentrate and Autoclave Product
Placer Dome Research	1999 (Feb)	File No. E98012N	Static Net Acid Generation Tests
Placer Dome Research	1997 (Nov)	N/A	Donlin Creek Project Autoclave Pilot-plant Testwork Report No. 1
Placer Dome Research	1995 (Nov)	N/A	Donlin Creek Project Bench Scale Grinding, Cyanidation, CIL, Flotation and Pressure Oxidation Tests Report No. 1
Placer Dome Research	1996 (Nov)	N/A	Donlin Creek Project Compilation of Results of 1996 Metallurgical Test Programme Report No. 2
Placer Dome Research	1999 (Dec)	N/A	Donlin Creek Project Bench Scale Grinding, CIL, Flotation, and Pressure Oxidation Tests on ACMA, Rochelieu, North Lewis, and South Lewis Ore Zones Report No. 3
Placer Dome Research	2006 (May)	T1010C11-02	Determination of the Neutralization Capacity of Calcareous Sandstones

Author	Report Date	Report Reference	Title
			and Flotation Tailings
Polysius Research Centre	2005 (Nov)	Project 2337 2152	High-Pressure Grinding Tests on Gold Ore
R&C Environmental Services	2006 (Nov)	NA	Laboratory Evaluation of the SO_2 / Air and Prussian Blue Process
RDi Resource Devel. Inc.	2002 (Nov)	Project No. 02-018	Flotation and Leaching of Donlin Creek Sample
SGS (Minnovex)	2007 (Aug)	LR 11328-007 (Design) and 11490-001 (Geostatistics)	Grinding Circuit Study for the Donlin Creek Project
SGS Lakefield Research Ltd	2004 (Sep)	LR 10044-104	An Investigation into the Grindability Characteristics of Two Samples from the Donlin Creek Deposit – Progress Report
SGS Lakefield Research Ltd	2005 (Jan)	LR 10044-154 Report 1	The Grindability Characteristics of Samples (CSS) from the Donlin Creek Deposit
SGS Lakefield Research Ltd	2005 (Feb)	LR 10829-001 Report 1	The Separation of Arsenopyrite and Pyrite from a Donlin Creek Intrusive Composite
SGS Lakefield Research Ltd	2007 (Feb)	LR 11328-001	The Variability in Comminution Characteristics of Samples from the Donlin Creek Deposit
SGS Lakefield Research Ltd	2006 (Dec)	LR 11328-001 Report 141206	Geostatistical Analysis and Estimation of Grindability Data from the Donlin Creek Deposit
SGS Lakefield Research Ltd	2007 (Mar)	LR 11328-001 Report 2	An Investigation into the Cyclone Separation of a Flotation Scavenger Concentrate
SGS Lakefield Research Ltd	2007 (Sep)	LR 11328-001 Report 3	An Investigation into the Production of Mineralogy Samples Via Flotation of the Donlin Creek Composites
SGS Lakefield Research Ltd	2007 (Sep)	LR 11328-003	Environmental & Geotechnical Testing of Donlin Creek Tailings
SGS Lakefield Research Ltd	2007 (Sep)	LR 11328-004	Geochemical Characterization – BTC 2006 Pilot Autoclave Test Program
SGS Lakefield Research Ltd	2007 (Nov)	LR 11328-005	An Investigation into Proposed Grinding Circuit for the Donlin Creek Deposit Based on Small-Scale Tests
SGS Lakefield Research Ltd	2007 (Apr)	LR 11328-006	Environmental Testing of Donlin Creek Pilot-plant Feed Samples
SGS Lakefield Research Ltd	2007 (Apr)	LR 11328-006 LIMS No. MI5011-JAN07 Report 1	An Investigation by QemScan into the Mineralogical Characteristics of Pilot-plant Product Streams from the Donlin Creek Project
SGS Lakefield Research Ltd	2007 (Sep)	LR 11328-006 Report 1	A Preliminary Pilot-plant Investigation into the Recovery of Gold from a Donlin Creek Composite
SGS Lakefield Research Ltd	2007 (Jun)	LR 11328-006 Report 2	An Investigation into the Metallurgical Performance of the Donlin Creek Composites
SGS Lakefield Research Ltd	2007 (Apr)	LR 11328-006 Rev 1	An Investigation into the FLEET Modelling of the Donlin Creek Flotation Circuit

Author	Report Date	Report Reference	Title
SGS Lakefield Research Ltd	2007 (Apr)	LR 11328-007	The Variability in Comminution Characteristics of Samples from the Donlin Creek Deposit
SGS Lakefield Research Ltd	2007 (Aug)	LR 11328-008 Report 1	An Investigation into the Recovery of Gold from Variability Samples
SGS Lakefield Research Ltd	2007 (Sep)	LR 11328-008 Report 1	A Summary of Batch Neutralization Test Results for the Donlin Creek Project Variability Study
SGS Lakefield Research Ltd	2007 (Sep)	LR 11328-008 Report 4	Summary of Results – Mineralogical Variability in Donlin Creek Ore Samples
SGS Lakefield Research Ltd	2007 (Apr)	LR 11328-009	Environmental Testing of Donlin Creek Phase 1 Pilot-plant 7 Flotation Samples
SGS Lakefield Research Ltd	2007 (Jun)	LR 11328-010	Environmental Testing of Donlin Creek Phase 2 Pilot-plant Feed Samples
SGS Lakefield Research Ltd	2007 (Sep)	LR 11328-010 Report 1	A Pilot-plant Optimization Study on the Recovery of Gold from a Donlin Creek Composite
SGS Lakefield Research Ltd	2007 (Sep)	LR 11328-010 Report 2	An Investigation into the Development of a Flowsheet for the Donlin Creek Composites
SGS Lakefield Research Ltd	2007 (Jun)	LR 11328-011	Continuance/Decommissioning of MWMP Residue Humidity Cell Tests & Test Data
SGS Lakefield Research Ltd	2007 (May)	LR 11328-011-MI5008-MAR07	An Investigation into Mineralogical Characterization of Three Final Tailings Samples
SGS Lakefield Research Ltd	2007 (Aug)	LR 11328-011-MI5023-MAY07	An Investigation into the Mineralogical Characterization of a Final Tailings Sample,
SGS Lakefield Research Ltd	2007 (May)	LR 11328-013	Geochemical Characterization – SGS Pilot Flotation PP-03 Survey Samples – February 2007 Pilot – Conventional
SGS Lakefield Research Ltd	2007 (May)	LR 11328-014	Geochemical Characterization – SGS Pilot Flotation PP-12 Survey Samples – February 2007 Pilot – MF2
SGS Lakefield Research Ltd	2007 (Aug)	LR 11328-015	Geochemical Characterization – BTC February 2007 Pilot-plant Tails Samples
SGS South Africa	2006 (Nov)	BIOMET 06/18	Bacterial Oxidation Amenability Testing of the Donlin Creek Concentrate – Phase 1
Signet Technology Inc	1999 (Mar)	N/A	Process Design and Cost Study for Bacterial Oxidation of Concentrates at Donlin Creek
Surface Science Western	2006 (Nov)	SSW Ref. 59606brh.BAR	Evaluation of Particle Surfaces using ToF SIMS
Surface Science Western	2007 (May)	SSW Ref. 90906brh.BAR.1	Statistical Differentiation of the Surface Species from a Pilot-plant Study
Surface Science Western	2007 (Jul)	SSW Ref. 90906brh.BAR.2	Statistical Differentiation of the Surface Species of Primary Mill Discharge Samples.
University of Alberta	2004 (Jun)	N/A	Separation of Pyrite and Arsenopyrite – A Literature Review

Author	Report Date	Report Reference	Title
University of British Columbia	2006 (Oct)	N/A	Rheology Testing on Donlin Creek Samples (Hot Cure, CN Detox Slurry)
University of British Columbia	2007 (Mar)	N/A	Rheology Testing of Donlin Creek Samples (Hot Cure, CIL Feed)
University of British Columbia	2007 (Jun)	N/A	Rheology Testing of Donlin Creek Samples (Hot Cure, Pre-acidified Conc)
University of British Columbia	2007 (Jul)	N/A	Rheology Testing of Donlin Creek Samples (CN Detox, Neut Product)
University of British Columbia	2005 (Nov)	Project Number T1010A03	Quantitative Phase Analysis of Four Pulp Samples (CSS) Using the Rietveld Method and X-ray Powder Diffraction Data.
University of Cape Town	2007 (Jun)	MPTech C-06-20	Donlin Creek Gold Ore Use of Reactivity Number and Nitrogen based flotation

16.3 Crushing / Grinding

More than 300 samples from the deposit have been bench-tested for hardness.

Geostatistical analyses have been undertaken on the complete hardness dataset. The resulting hardness model was used for circuit design and capacity estimation.

The ores are considered moderately hard, with an average Ball Work Index of 13.61 kWh/st (15 kWh/t) (Table 16-2). The ores are amenable to SAG milling.

The hardness properties of core samples that have been exposed to the freeze/thaw cycle of an Alaska winter are considered unreliable because of alteration and cannot be used on their own (without proper calibration) for the purpose of grinding circuit design.

Table 16-2: Estimated Bond Ball Work Index (BWI)

Domain	BWI Mean (kWh/t)	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**

16.4 Flotation

Extensive bench flotation testwork has been undertaken. Flotation recoveries are highest from intrusive ores, at 94.7 to 97.5%, lower from the sedimentary ores at 89.7 to 91.3%, and problematic for partially geologically oxidized or altered ores, averaging 75.7% (see Table 16-3 and Figure 16-1).

Testwork confirmed the very close relationship between arsenic and gold recovery.

An MCF2 (mill chemical float, twice) style flowsheet with 100 minutes of residence time increases the gold recovery to a 7% sulphur flotation concentrate by an estimated 1.8%.

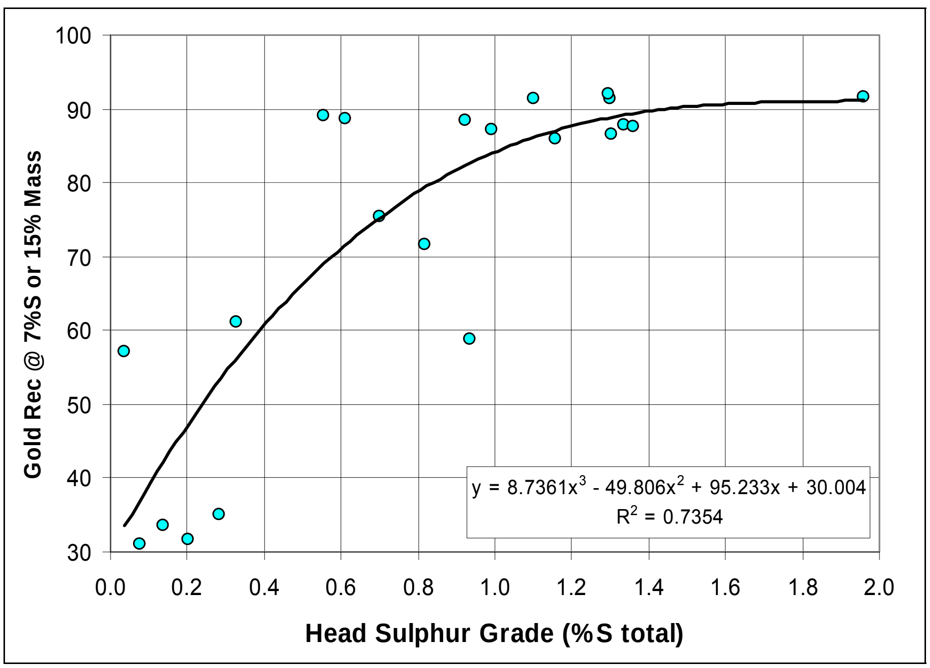
CIL leaching of the flotation tails does not yield economical gold recovery.

Table 16-3: Summary of Final Flotation Recovery by Geological Domain (without oxide)

	Tonnes (%)	Adjusted Recovery* to MCF2 Pilot Result (%)
AKIVIK	6.2	97.48
400	7.3	96.84
ACMA	16.0	96.32
AURORA	5.3	96.09
VORTEX	13.0	95.05
LEWIS	26.7	94.74
GWK	21.8	91.33
SHL	3.9	89.71
Overall	**100.0**	**94.49**

* Not corrected for oxide content

Figure 16-1: Variability of Flotation Gold Recovery Results for Oxidation Samples



16.5 Pressure Oxidation (POX)

Dynatec and Barrick Technology Center have completed numerous pressure oxidation bench-testing programs. Three separate autoclave pilot test programs, each incorporating a number of different test runs, were conducted in 2006 and 2007.

POX allows for 96.6% recovery of the gold in CIL following oxidation.

A counter-current decantation (CCD) washing circuit has been incorporated ahead of pressure oxidation to control soluble species from the concentrator in the feed to the autoclave.

Autoclave gas emissions will pass through a mercury recovery system before release to the atmosphere.

16.6 CIL / Gold Recovery

CIL processing of the washed autoclave product produces optimized gold recoveries of 96 to 97% and requires low amounts of cyanide.

Lime consumption by the CIL feed can be minimized by operating the CIL circuit at a pH of ~9.0 to avoid precipitation of magnesium hydroxide.

Ensuring high-efficiency washing of the autoclave product, preferably to 98% or more, minimizes lime and cyanide reagent addition in CIL.

A Cherokee Chemical Company UNR reagent can be used to control mercury leached into solution from the cyanidation of the autoclave product (mercury that is leached by cyanide and does not adsorb onto carbon) to low levels within the recirculating process water streams.

Mercury recovery systems for off-gas/vapour streams have been designed for the carbon regeneration kiln, electrowinning, retort system and smelting furnaces.

16.7 Environmental Considerations

The high temperature POX process is generally considered best practice for the generation of stable arsenic compounds suitable for long-term disposal in a tailings storage facility, depending on the amount of iron available for co-precipitation with soluble arsenic. The Donlin Creek mill feed has sufficient iron content to provide the recommended minimum stoichiometric ratio of 4:1 iron to arsenic. Iron is mainly associated with pyrite, arsenopyrite and ferroan dolomite; iron content as siderite is minor.

The tailings decant water from the process plant will likely contain levels of arsenic, mercury, magnesium, molybdenum, selenium and antimony above applicable standards. The tailings water will also be elevated in sulphates (greater than 10 g/L), particularly due to the presence of magnesium, which increases the solubility level of sulphate in solution.

Recycling of tailings decant waters will be considered.

Testing has been undertaken to confirm the use of the Air/SO2 process for cyanide destruction in the CIL tails before being blended with the neutralization circuit. The backup use of soluble iron sulphate for the destruction of weak acid dissociable cyanide (WAD CN) has also been shown to be effective.

All plant tailings will be neutralized before discharge to the tailings storage facility.

The ABA and humidity cell testing results to date indicate that the final plant tailings stream will be maintained as non-acid-generating.

16.8 Conceptual Process Plant Design

The design criteria for the conceptual process plant facilities are primarily based on testwork results stemming from pilot-plant runs and variability testwork programs. Modern modelling techniques were utilized in the design of the grinding, autoclave and flotation circuits. SGS Lakefield developed a CEET (Comminution Economic Evaluation Tool) model for the Donlin Creek project to determine the power requirements of the grinding circuit; Barrick and JK Tech developed a JKSimFloat model to size the flotation circuit. In addition, various aspects of the process design were benchmarked to compare the Donlin Creek project with other similar operations.

The design of the Donlin Creek process plant is based on the most current, modern-day technology for both the process circuits and equipment selection. Particular attention was paid to incorporate state-of-art technology—such as mercury abatement systems to control mercury—for safety and environmental protection.

The conceptual process plant is designed to recover a sulphide flotation product and oxidize the refractory gold concentrate in a pressure oxidation circuit prior to cyanidation. Highlights of the process plant are as follows:

- The gyratory crusher with a capacity for 45,000 t/d (49,600 st/d).
- The design is based on the MCF2 grinding and flotation circuit. The grinding circuit is SABC with a single SAG mill in closed circuit with parallel cone crushers followed by primary ball mill in closed circuit with cyclones. Primary ball mill product reports to primary rougher flotation. Rougher flotation tailings report to the secondary ball mill circuit, while in closed circuit with cyclones. Secondary ball mill product at P_{80} 50 µm reports to secondary rougher flotation. Secondary rougher flotation concentrate reports to cleaner flotation. A cleaner scavenger flotation circuit treats the cleaner flotation tailings.
- Combined flotation concentrates from primary rougher and cleaner flotation are dewatered in a thickener before acidulation and CCD washing to remove

solubilized species from the concentrator and acidulation portions of the process. It has been shown that high levels of soluble species in the feed to the autoclave have detrimental effects on the POX/CIL gold recovery.

- The autoclave circuit includes two autoclaves operating in parallel. The autoclaves are designed to operate at 437°F (225°C) with a retention time of 50 minutes.
- Thickened flotation tailings slurry is used as a cooling medium in the autoclave letdown circuit.
- Flotation tailings are combined with the flotation concentrate wash solution product to neutralize the acidic solution before discharge to the tailings storage facility (TSF). The carbonate in the flotation tailings slurry will provide primary neutralization. The final pH level will be adjusted by adding slaked lime.
- Flashed and cooled autoclave discharge slurry is cured for 6 hours at 212°F (100°C) before POX discharge CCD. The acidic solution recovered by CCD is recycled to acidulation and flotation feed conditioning.
- POX CCD product slurry should be neutralized with lime ahead of cyanidation.
- The CIL circuit will operate at pH 9.0. The CIL tanks should be fully enclosed and vented to a caustic scrubbing system to recover cyanide and recycle it back to the CIL circuit. The CIL circuit has a residence time of 24 hours. Carbon will be handled with in-tank revolving screens.
- The carbon handling area for the loaded carbon consists of an acid wash circuit and a modified pressure Zadra circuit for stripping carbon. Carbon will be reactivated in an electric kiln.
- Gold will be recovered in an electrowinning circuit. The electrowinning sludge will be treated in a retort before being melted in an induction furnace.
- Mercury that evolves in the process plant will be captured in a number of mercury abatement systems, which will treat the following streams: autoclave flash vent, regeneration kiln feed and discharge vents, electrowinning vents, retort furnace exhaust, induction furnace vent and general refinery area ventilation stream.

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 Introduction

The mineral resource estimates for the Donlin Creek project were calculated by Barrick in a model referred to as "DC7a". NovaGold has independently verified and validated the results. Kevin Francis, P.Geo of NovaGold is the qualified person regarding this work.

The Donlin Creek model (project area) was built using geological information and assay data collected from 1,000 core holes totalling 286,120 m (938,714 ft), 296 RC holes totalling 29,294 m (96,109 ft) and 21,897 m of trenching. A total of 160,041 assays were used in the model. The estimates were made from a 3D block model utilizing Vulcan™ mine modelling software. The cell size for the model is 6 m east by 6 m north by 6 m high.

There are presently no mineral reserves at Donlin Creek.

17.2 Estimation Approach

At Donlin Creek, the felsic dykes and sills are the main host for gold mineralization. In Lewis, and to a lesser extent at ACMA and Aurora-Akivik, the hosting sediments can also contain gold mineralization. As gold mineralization is not pervasive in these host units (the mineralization only occurs in areas which have been structurally prepared), Barrick chose to define mineralized envelopes within intrusive and sedimentary rocks using probability-assisted domaining. Sulphur was constrained in a similar manner. Once the mineralized envelopes are defined, grade is estimated outside and within the envelopes using ID3.

17.3 Geologic Model

Three-dimensional (3D) solids of the geological model were generated from polygon interpretations constructed from cross-section and level plans. Tools available in Vulcan were used to create the polygons, which 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 used to assign the corresponding geological code to the 3D block model. Priorities were assigned to each solid depending on the timing of the dyke or sill and to account for any overlapping. 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 specified in Table 17-1. Rocks assigned a greywacke code had the lowest priority value.

Table 17-1: Three-Dimensional Solids Used in the Modelling Process

Triangulation Name	Block Variable	Block Value	Block Variable	Block Value	Composite Field	Composite Value	Domain	Rock Type	Priority
other_1a.00t	rock	1	rock_mine	1	ore	1	ACMA	Intrusive	5
other_3b.00t	rock	2	rock_mine	1	ore	2	Lewis	Intrusive	5
other_5a.00t	rock	3	rock_mine	1	ore	3	Akivik	Intrusive	5
other_8.00t	rock	4	rock_mine	1	ore	4	400	Intrusive	5
other_9.00t	rock	5	rock_mine	1	ore	5	Aurora L.	Intrusive	5
rda_dyke_2.00t	rock	6	rock_mine	2	ore	6	Vortex	Intrusive	45
rda_dyke_3c.00t	rock	7	rock_mine	2	ore	7	Lewis	Intrusive	45
rda_dyke_4a.00t	rock	8	rock_mine	2	ore	8	Vortex	Intrusive	45
rda_dyke_4b.00t	rock	9	rock_mine	2	ore	9	Vortex	Intrusive	45
rda_dyke_4c.00t	rock	10	rock_mine	2	ore	10	Vortex	Intrusive	45
rda_rdxb_sill_9.00t	rock	11	rock_mine	2	ore	11	Aurora	Intrusive	50
rda_sill_1a.00t	rock	12	rock_mine	2	ore	12	ACMA	Intrusive	40
rda_sill_3c.00t	rock	13	rock_mine	2	ore	13	Lewis	Intrusive	40
rda_sill_3d.00t	rock	14	rock_mine	2	ore	14	Lewis	Intrusive	40
rda_sill_4a.00t	rock	15	rock_mine	2	ore	15	Vortex	Intrusive	40
rda_sill_5b.00t	rock	16	rock_mine	2	ore	16	Akivik	Intrusive	40
rda_sill_6.00t	rock	17	rock_mine	2	ore	17	Wedge	Intrusive	40
rda_sill_8.00t	rock	18	rock_mine	2	ore	18	400	Intrusive	40
rda_sill_9.00t	rock	19	rock_mine	2	ore	19	Aurora	Intrusive	40
rdf_dyke_3a.00t	rock	20	rock_mine	3	ore	20	Lewis	Intrusive	15
rdf_dyke_3b.00t	rock	21	rock_mine	3	ore	21	Lewis	Intrusive	15
rdf_dyke_3c.00t	rock	22	rock_mine	3	ore	22	Lewis	Intrusive	15
rdf_dyke_3d.00t	rock	23	rock_mine	3	ore	23	Lewis	Intrusive	15
rdf_dyke_5a.00t	rock	24	rock_mine	3	ore	24	Akivik	Intrusive	15
rdx_dyke_3a.00t	rock	25	rock_mine	4	ore	25	Lewis	Intrusive	25
rdx_dyke_3b.00t	rock	26	rock_mine	4	ore	26	Lewis	Intrusive	25
rdx_dyke_3c.00t	rock	27	rock_mine	4	ore	27	Lewis	Intrusive	25
rdx_sill_1a.00t	rock	28	rock_mine	4	ore	28	ACMA	Intrusive	20
rdx_sill_3b.00t	rock	29	rock_mine	4	ore	29	Lewis	Intrusive	20
rdx_sill_3c.00t	rock	30	rock_mine	4	ore	30	Lewis	Intrusive	20
rdx_sill_3d.00t	rock	31	rock_mine	4	ore	31	Lewis	Intrusive	20
rdx_sill_3e.00t	rock	32	rock_mine	4	ore	32	Lewis	Intrusive	20
rdx_sill_5b.00t	rock	33	rock_mine	4	ore	33	Akivik	Intrusive	20
rdx_sill_7.00t	rock	34	rock_mine	4	ore	34	Tortured	Intrusive	20
rdx_sill_8.00t	rock	35	rock_mine	4	ore	35	400	Intrusive	20
rdx_sill_9.00t	rock	36	rock_mine	4	ore	36	Aurora	Intrusive	20
rdxb_dyke_3a.00t	rock	37	rock_mine	5	ore	37	Lewis	Intrusive	55

Triangulation Name	Block Variable	Block Value	Block Variable	Block Value	Composite Field	Composite Value	Domain	Rock Type	Priority
rdxb_dyke_3b.00t	rock	38	rock_mine	5	ore	38	Lewis	Intrusive	55
rdxb_dyke_3c.00t	rock	39	rock_mine	5	ore	39	Lewis	Intrusive	55
rdxb_dyke_3d.00t	rock	40	rock_mine	5	ore	40	Lewis	Intrusive	55
rdxb_sill_1a.00t	rock	41	rock_mine	5	ore	41	ACMA	Intrusive	50
rdxb_sill_3c.00t	rock	42	rock_mine	5	ore	42	Lewis	Intrusive	50
rdxb_sill_3d.00t	rock	43	rock_mine	5	ore	43	Lewis	Intrusive	50
rdxb_sill_6.00t	rock	44	rock_mine	5	ore	44	Wedge	Intrusive	50
rdxb_sill_9.00t	rock	45	rock_mine	5	ore	45	Aurora	Intrusive	50
rdxl_dyke_1a.00t	rock	46	rock_mine	6	ore	46	ACMA	Intrusive	35
rdxl_dyke_5a.00t	rock	47	rock_mine	6	ore	47	Akivik	Intrusive	35
rdxl_dyke_5b.00t	rock	48	rock_mine	6	ore	48	Akivik	Intrusive	35
rdxl_sill_1a.00t	rock	49	rock_mine	6	ore	49	ACMA	Intrusive	30
rdxl_sill_5b.00t	rock	50	rock_mine	6	ore	50	Akivik	Intrusive	30
rdxl_sill_8.00t	rock	51	rock_mine	6	ore	51	400	Intrusive	30
basal_shales.00t	rock	92	rock_mine	8	ore	92	All	Shale	1
main_shl.00t	rock	92	rock_mine	8	ore	92	All	Shale	1
mid_shl.00t	rock	92	rock_mine	8	ore	92	All	Shale	1
upper_shl.00t	rock	92	rock_mine	8	ore	92	All	Shale	1
dc5_whit_air.00t	topo	99				99	All	Air	10
ovb_topo_edited_20070330.00t	topo	2				2	All	Overburden	1
topo_surface_061107.00t	topo	3				3	All	Topo	5

In addition to the intrusive wireframes, solids were generated to represent overburden and existing wireframes were used to constrain areas dominated by shale. Topography was also incorporated along with a variable to account for the percentage of material above and below the topographic surface.

NovaGold validated and checked for crossing errors, consistency and closure. NovaGold found that rdx_sill_3b_dc7.00t, one of the smallest intrusive triangulations, was improperly applied to the geologic model resulting in intrusive blocks being coded as sediment. The volume represented by this triangulation is minor and, in the opinion of NovaGold, immaterial.

17.4 Capping

Raw core and RC assays were examined for the presence of local high-grade outliers. Once these outliers were identified, the overall grade distributions were utilized 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.

Rock codes present in the database and major rock types used in the statistical analysis are listed in Table 17-2. A review of the codes identified several lithological codes that are relict from early drill logs and represent a very small proportion of the logged intervals. Barrick determined that several of these codes could be grouped into the major rock types for the purpose of statistical analysis, reducing the number of rock types from 34 to 12 The numerical “grouped code” value identifies all rock types assigned to a given major rock type.

Assay top cuts were selected from cumulative frequency plots. The distribution for all gold assays (Figure 17-1) 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. This is therefore the position at which the raw assay would be capped and represents a total metal loss of 2.1%.

Individual frequency distribution plots were generated to determine the appropriate grade cap for each rock type. The selected grade cap for each rock type was applied to all raw assays prior to compositing. Cumulative frequency distribution plots for each of the major rock types are provided in Figure 17-2 to Figure 17-9.

The cumulative frequency distribution for gold assays logged as greywacke (Figure 17-2) departs from the approximate lognormal trend at grades above 25 g/t Au, corresponding to the 98.6th percentile. This grade level cap was therefore applied to a total of 94 gold assays before compositing, representing a metal loss of 4.20%.

The cumulative frequency distributions for gold assays from lithological units logged as shale or argillite (Figure 17-3), rhyodacite aphanitic porphyry (Figure 17-5), rhyodacite porphyry (Figure 17-7) and rhyodacite coarse-grained blue porphyry (Figure 17-8) depart from the approximate lognormal trend at about 30 g/t Au. This grade level cap was therefore applied to a total of 14 shale and argillite, 14 rhyodacite aphanitic porphyry, 26 rhyodacite porphyry and 13 rhyodacite coarse-grained blue porphyry gold assays before compositing, representing metal losses of 6.89%, 0.43%, 1.36% and 0.84%, respectively.

Table 17-2: Rock Codes in the Drill Database

Lithology Code	Rock Type Description	Grouped Code
CHT	Chert	14
CGL	Conglomerate	14
GWK	**Greywacke**	**14**
BHF	Biotite Hornfels	15
CHF	Calcsilicate Hornfels	15
HFL	**Hornfels**	**15**
MD	**Mafic Dyke**	**17**
GRN	Ground Core/Missing Core	19
NS	No Sample	19
OVB	**Overburden**	**20**
OVR	Channel Deposits	20
ARH	Aphanitic Rhyolite	26
RDA	**Rhyodacite Aphanitic Porphyry**	**26**
RDF	**Rhyodacite Fine-grained Porphyry**	**27**
GDR	Granodiorite	28
GP	Grey Porphyry	28
GPX	Grey Crystalline Porphyry	28
GRH	Grey Rhyolite	28
IBX	Intrusive Breccia	28
MZD	Monzodiorite	28
QM	Quartz Monzonite	28
QLT	Quartz-latite Porphyry	28
QMP	Quartz-monzonite Porphyry	28
RD	Rhyodacite	28
RDX	**Rhyodacite Porphyry**	**28**
RHY	Rhyolite	28
BP	Blue Porphyry	29
BR	Blue Rhyolite	29
RDXB	**Rhyodacite Coarse-grain Blue Porphyry**	**29**
RDXL	**Rhyodacite Lath-rich Porphyry**	**30**
ARG	Argillite	33
SED	Sediments	33
SHL	**Shale**	**33**
SLT	**Siltstone**	**34**

Note: Major rock types used in the block model are highlighted in **bold type**. Assignment of minor rock types is shown in the grouped code

Figure 17-1: Raw Au Assays – Cumulative Frequency Distribution of All Rock Types

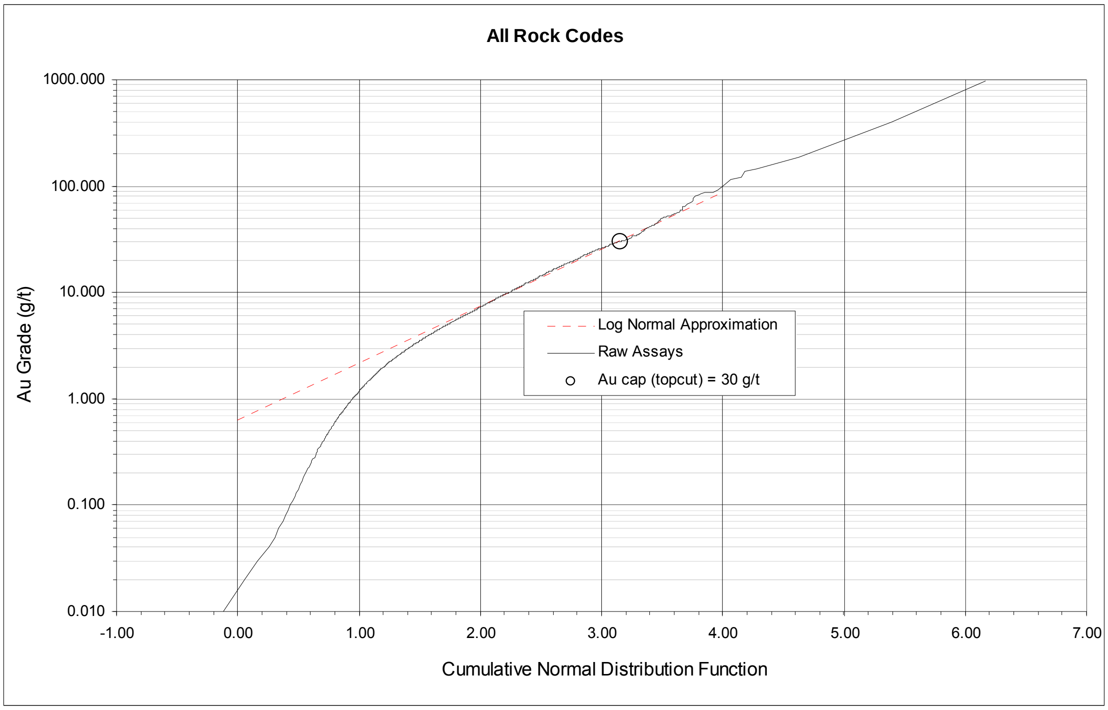


Figure 17-2: Raw Au Assays – Greywacke Cumulative Frequency Distribution Plot

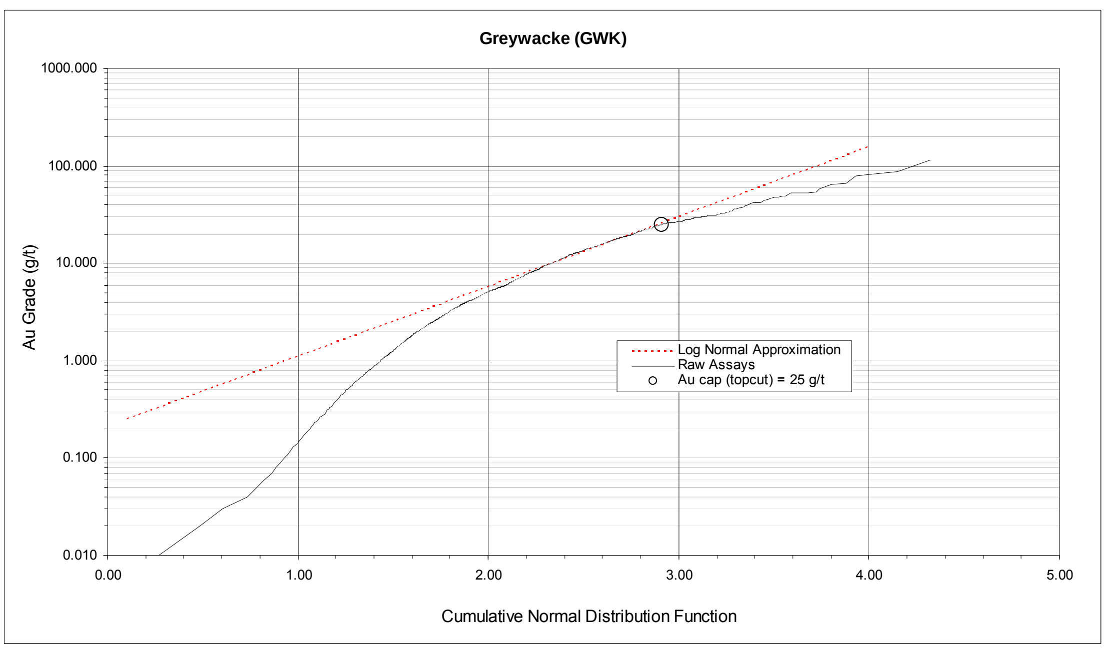


Figure 17-3: Raw Au Assays – Shale and Argillite Cumulative Frequency Distribution Plot

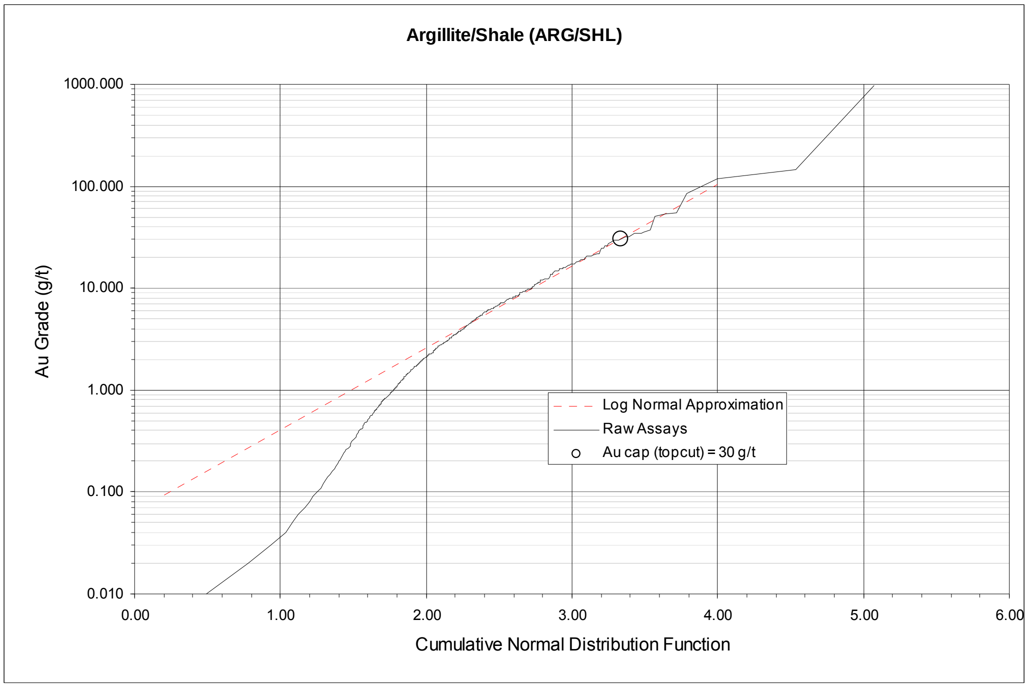


Figure 17-4: Raw Au Assays – Mafic Dyke Cumulative Frequency Distribution Plot

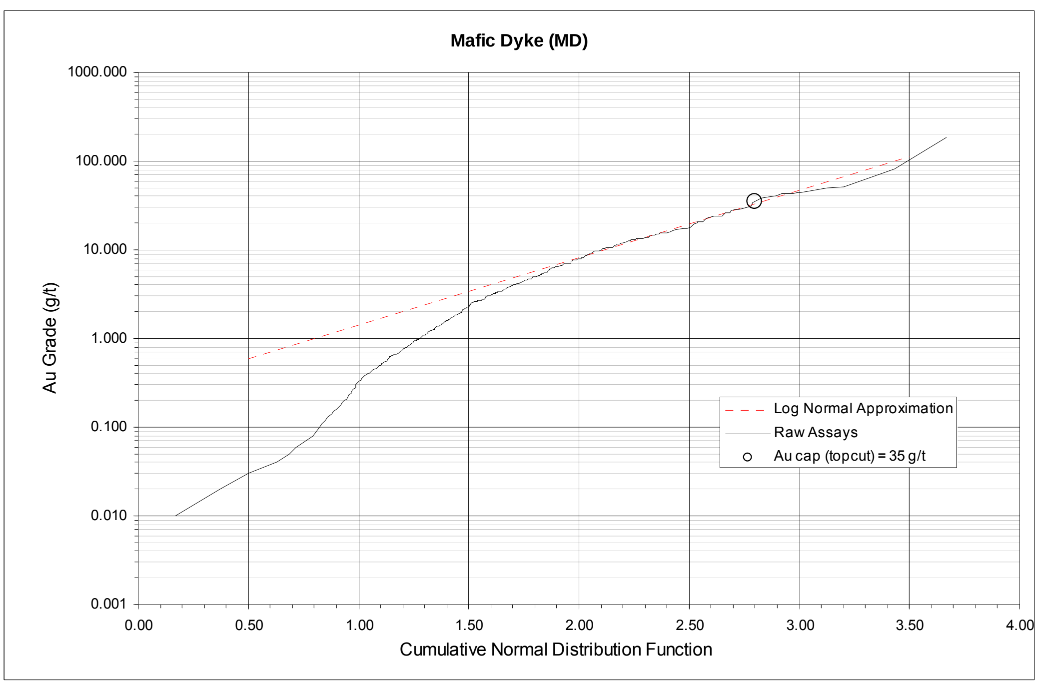


Figure 17-5: Raw Au Assays – Rhyodacite Aphanitic Porphyry Cumulative Frequency Distribution Plot

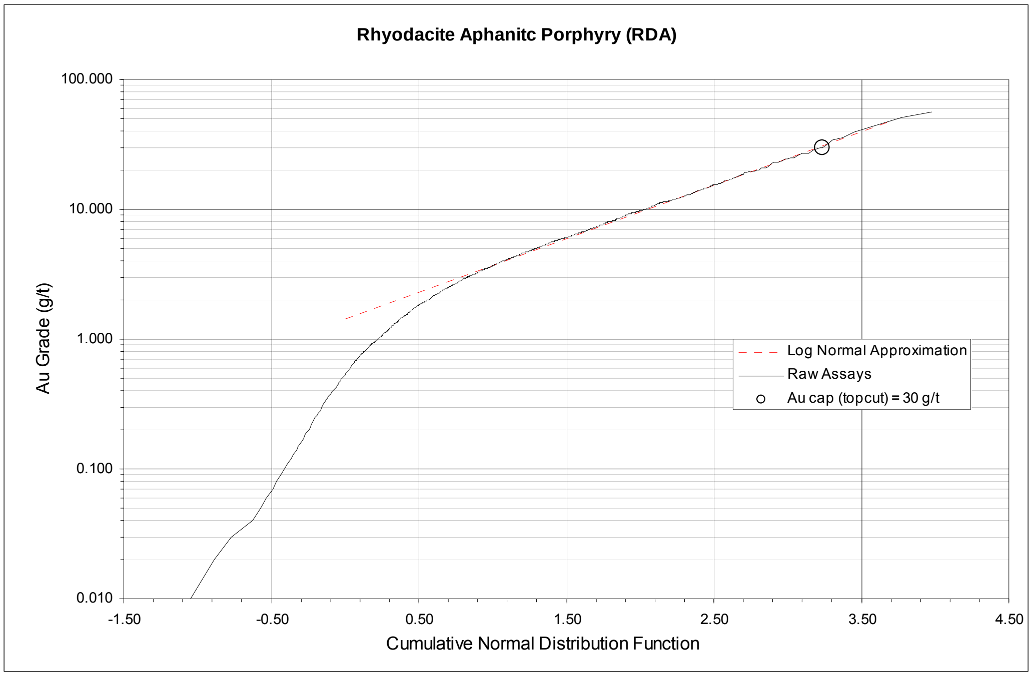


Figure 17-6: Raw Au Assays – Rhyodacite Fine-Grained Porphyry Cumulative Frequency Distribution Plot

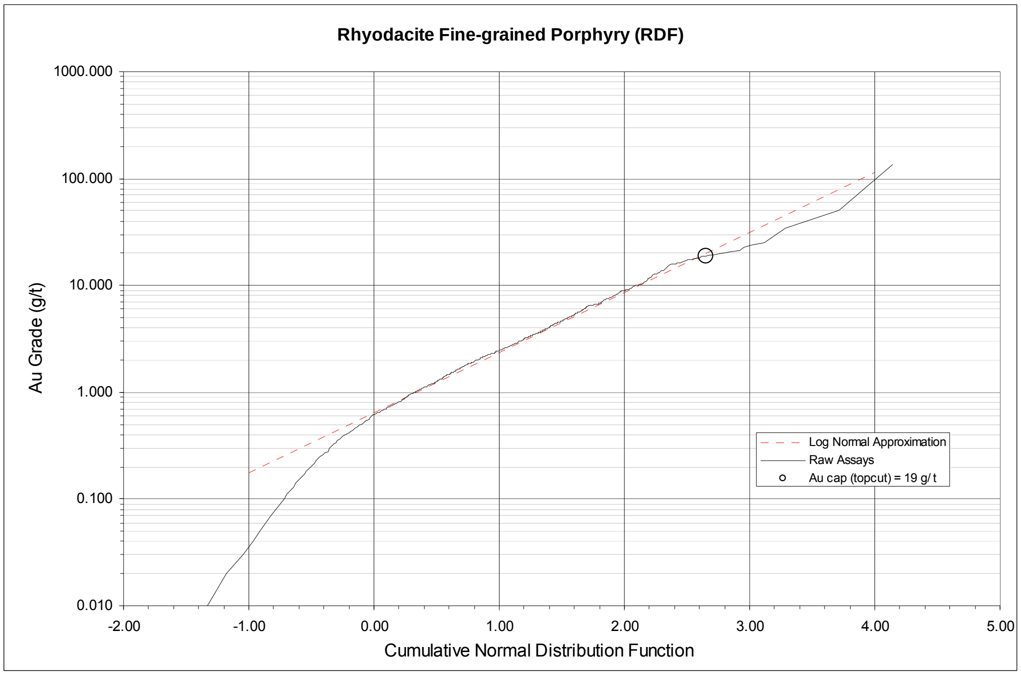


Figure 17-7: Raw Au Assays – Rhyodacite Porphyry Cumulative Frequency Distribution Plot

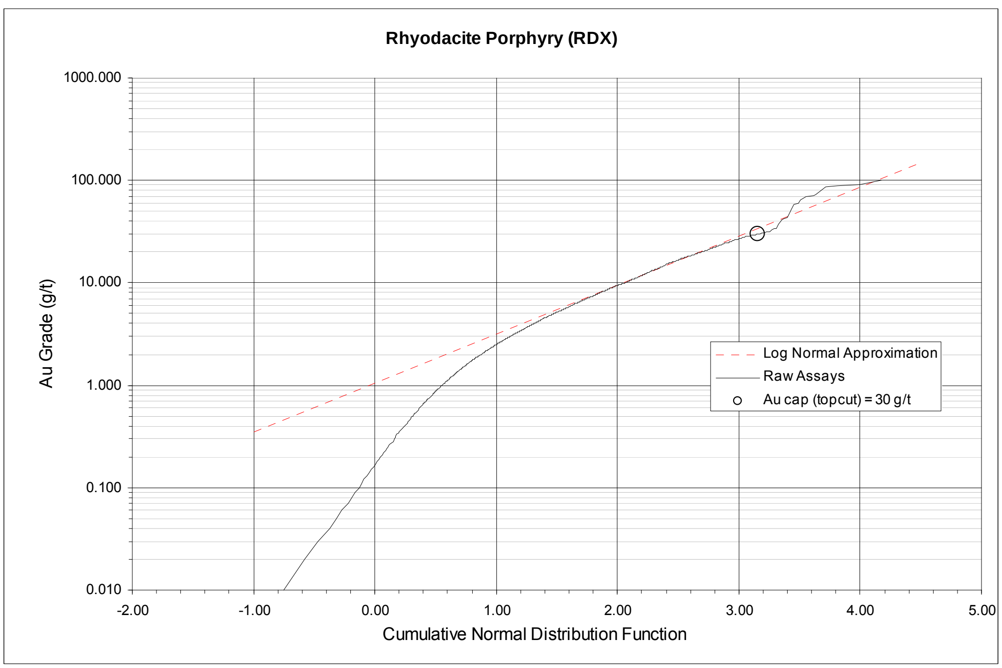


Figure 17-8: Raw Au Assays – Rhyodacite Coarse-Grained Blue Porphyry Cumulative Frequency Distribution Plot

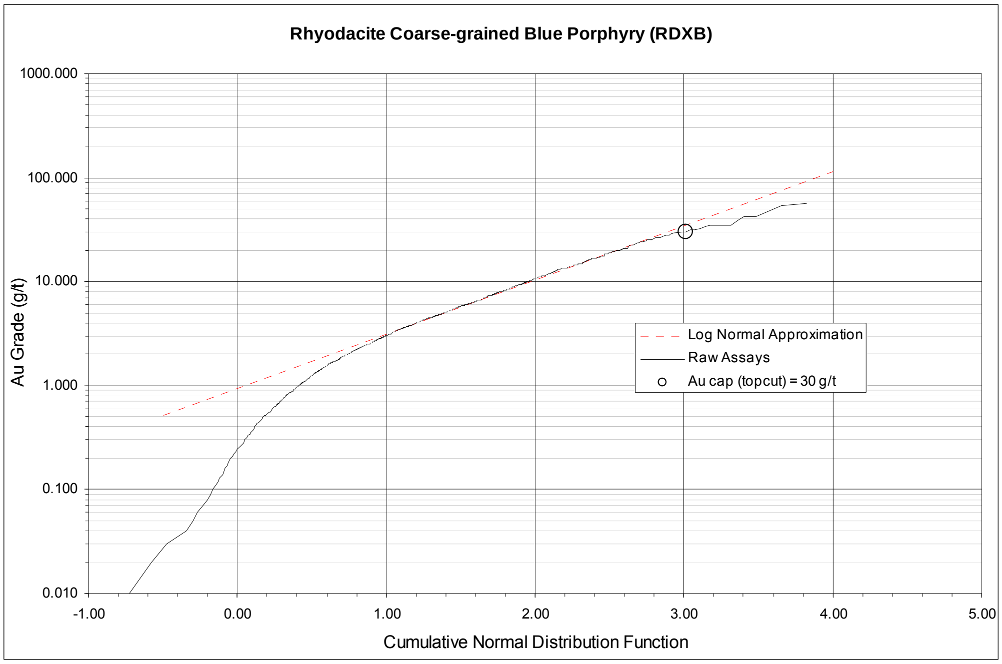
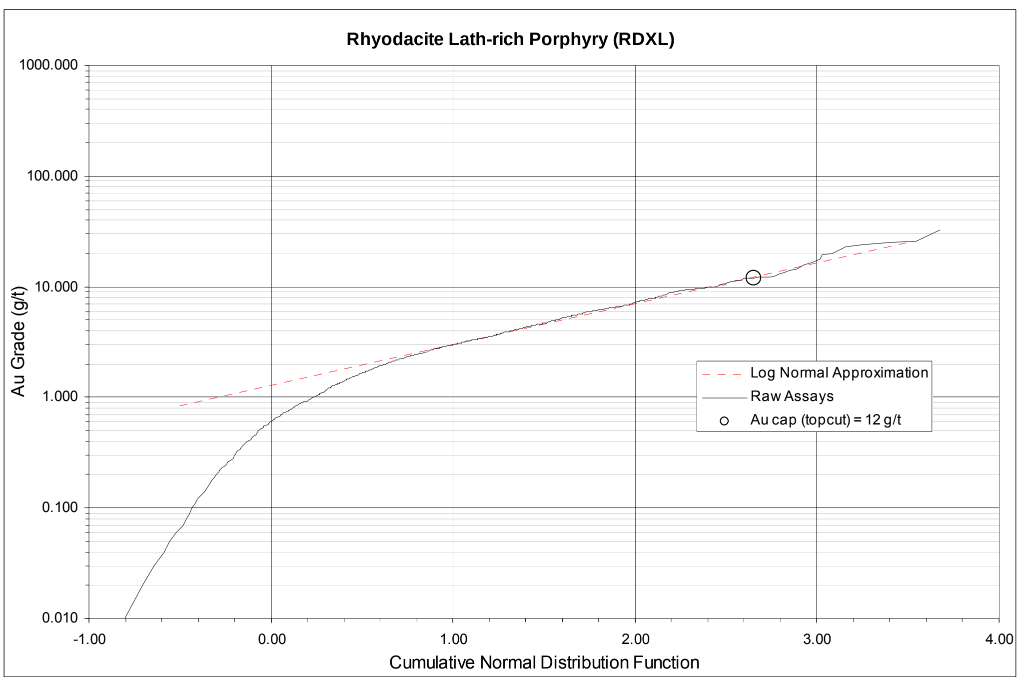


Figure 17-9: Raw Au Assays – Rhyodacite Lath-Rich Porphyry Cumulative Frequency Distribution Plot



The cumulative frequency distribution for gold assays for rock types logged as mafic dyke (Figure 17-4) departs from the approximate lognormal trend and becomes erratic at grades above 35 g/t Au, corresponding to the 99.74th percentile. This grade level cap was therefore applied to a total of 13 gold assays before compositing, representing a metal loss of 10.58%.

The cumulative frequency distribution for gold assays logged as rhyodacite fine-grained porphyry (Figure 17-6) departs from the approximate lognormal trend and becomes more erratic above 19 g/t Au, corresponding to the 99.27th percentile. This grade level cap was therefore applied to a total of 13 gold assays before compositing, representing a metal loss of 2.71%.

The cumulative frequency distribution for gold assays logged as rhyodacite lath-rich porphyry (Figure 17-9) departs from the approximate lognormal trend and becomes more erratic above 12 g/t Au, corresponding to the 98.4th percentile. This grade level cap was therefore applied to a total of 19 gold assays before compositing, representing a metal loss of 1.52%.

The effect of capping on the co-efficient of variance (COV) is summarized in Table 17-4. Values represent all assays greater than 0.1 g/t Au.

Table 17-3: Donlin Capping

Major Group	Proposed Threshold and Cap (g/t)	Actual Threshold and Cap (g/t)
GWK	25	25
HFL	10	10
MD	35	35/30
No Sample / Ground Core	30	30
OVB	5	5
RDA	30	30
RDF	19	19/16
RDX	30	30/28
RDXB	30	30/28
RDXL	12	12/10
SHL	30	30
SLT	30	30/20

Table 17-4: Coefficient of Variance for Au Greater than 0.1 g/t, by Rock Type

Rock Type Code	COV – Uncapped	COV – Capped
GWK	2.04	1.71
SHL/ARG	3.93	1.86
SLT	4.09	1.76
MD	2.73	0.92
OVB	2.61	1.28
RDA	1.25	1.20
RDF	1.78	1.42
RDX	1.64	1.46
RDXB	1.45	1.38
RDXL	1.09	0.99
All Rock Types	**1.87**	**1.51**

NovaGold has reviewed the cumulative distributions, the capping script and compared the capped and uncapped assays. NovaGold has found that the capping script has errors where the threshold and cap value are not the same. The result is that there are 57 assays that should have been capped, based on the plan, that were not capped. In NovaGold's opinion, it is unlikely that the error would result in a materially different global resource.

17.5 Specific Gravity Data and Statistics

Specific gravity (SG) data were gathered from the 1996–1997 and the 2006 drilling program. As of March 5, 2007, the total number of SG measurements collected amounted to 7,370 points. The weighted average of these SG data points is 2.69 (g/cm^3).

Table 17-5 summarizes the average SG values by rock type.

The SG data were further evaluated based upon the grouped rock codes used when estimating gold. The data were grouped and then divided into three main rock types: intrusive rocks, greywacke and shale. For each grouped rock type, data points clearly identified as outliers were removed before the average was determined. As is observed in Table 17-5, the SG data are quite similar within each grouped unit. Based on these statistical results, it was decided that the grouped average SG values listed in Table 17-6 would be sufficient for tonnage estimation and that a block model of SG estimates was not warranted.

Table 17-5: Specific Gravity Values by Rock Type

Rock Code in DH Database	Rock Types and Domain	No. of Samples	Specific Gravity (g/cm^3)
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	**All Rock Types**	**7,370**	**2.69**

Table 17-6: Specific Gravity Values by Grouped Rock Type

Grouped Rock Type	Individual Rock Types	No. of Samples	Specific Gravity (g/cm^3)
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	**All Rock Types**	**6,763**	**2.68**

17.6 Assay Statistics

Most of the potentially economic grades are hosted by the felsic dykes, with significant gold grades and grade thicknesses also hosted in shale and other sedimentary rocks, greywacke, and in rocks logged as mafic dykes.

The logged rock code was used to cap raw assays but was not used to constrain the estimates during grade estimation.

17.7 Compositing

Compositing is done prior to grade estimation to place the assay data on a near-constant support. Metal grades, geology and alteration data were composited "down-the-hole" into equal length, 6 m (20 ft) composites. Integer codes for domain and lithology were added to each composite by back tagging from the block model. NovaGold carried out checks on the calculation of the length-weighted grade and assignment of the integer codes. NovaGold confirmed these were correctly performed.

Three composite databases were generated: one for gold values where non-assayed (missing) intervals are set to zero, one for sulphur, and one for arsenic, antimony and mercury (multi-elements). In the latter two databases, assay values are not capped and missing intervals are ignored. Principal composite fields are described in Table 17-7.

Table 17-7: Composite Database – Fields Used to Generate Estimates

Database Field	Description	Use During Estimation Process
AU_USE	Capped Gold Grade Composite	Raw assays are capped prior to compositing. This variable is used to estimate gold. Composite is reported in g/t
LENG02	Gold Composite Length	Records the composite assay length for gold. Block estimates for gold are weighted by length.
SU_USE	Sulphur Composite	Used to estimate total sulphur. Composite is reported in percent. Sulphur assays were not capped prior to compositing.
LENG03	Sulphur Composite Length	Records the composite assay length for total sulphur. Block estimates for sulphur are weighted by length.
AS_USE	Arsenic Composite	Used to estimate arsenic. Composite is reported in ppm. Arsenic assays were not capped prior to compositing
LENG04	Arsenic Composite Length	Records the composite assay length for arsenic. Block estimates for arsenic are weighted by length.
HG_USE	Mercury Composite	Used to estimate mercury. Composite is reported in ppm. Mercury assays were not capped prior to compositing.
LENG05	Mercury Composite Length	Records the composite assay length for mercury. Block estimates for mercury are weighted by length.
SB_USE	Antimony Composite	Used to estimate antimony. Composite is reported in percent. Antimony assays were not capped prior to compositing.
LENG09	Antimony Composite Length	Records the composite assay length for antimony. Block estimates for antimony are weighted by length.
INTIND	Intrusive Indicator Field.	Contains integer value 0, 1 or -99. For assays with: INTIND =1, if Rock >=1 and <= 55 & Au >= 0.25 g/t INTIND =0, if Rock >=1 and <= 55 & Au < 0.25 g/t INTIND =-9, if Rock >=1 and <= 55 or ≠ 1-6 & Au < 0 g/t
SEDINT	Sediment Indicator Field	Contains integer value 0, 1 or -99. For assays with: SEDINT =1, if Rock >=92 and <= 93 & Au >= 0.25 g/t SEDINT =0, if Rock >=92 and <= 93 & Au < 0.25 g/t SEDINT =-9, if Rock >=92 and <= 93 or ≠ 1-8 & Au < 0 g/t
ORE	Rock Code Field	Rock codes were assigned to the composite using the rock code in the block model (see Table 3.6). This field is used to restrict composites used during the estimation of gold and other elements.

17.8 Block Model Construction

The block model for the Donlin Creek resource model was created using Vulcan™ mine modelling software. There was no sub-blocking used against the geological surfaces or topography and all blocks are 6 m × 6 m × 6 m in size. The coordinate system is UTM Zone 4 metres – NAD83. The model framework is provided Table 17-8.

Table 17-8: Block Model Origin and Extent

Model Start Offset (UTM metres)			Block Size (m)			Model End Offset (UTM metres)		
X	Y	Z	X	Y	Z	X	Y	Z
539,000	6,877,000	-500	6	6	6	543,002	6,881,002	454

17.9 Definition of Mineralized Envelopes

Probability shells were constructed for use in constraining gold and sulphur grade estimates in intrusives and sediments.

17.9.1 Gold

The first stage in defining the mineralized envelopes is to divide the composites into intrusive and sediment, mineralized and non-mineralized groups using an indicator grade of 0.25 g/t gold. An indicator value[1] was set to one if the composite value was equal to or greater than the chosen threshold and to zero if the composite value was less than the threshold. More formally:

$i(x, z_c) = 1$ If the grade z_x is ∃ threshold

$i(x, z_c) = 0$ If the grade z_x is < threshold

Where z_x is the composite defined in X, Y and Z space.

The probability that gold in any block in the model exceeded the threshold was estimated using the indicator data and inverse distance to a power squared (ID2). After visual inspection, a threshold of 0.5 was selected to divide mineralized from non-mineralized domains.

17.9.2 Sulphur

Similarly, indicator values were assigned to the sulphur 6 m composite database such that all intrusive data that were below 0.50% S were assigned an intrusive indicator value of 0 and above 0.5% S a value of 1. All sediment data with a composite sample grade below 0.50% S were assigned a sediment indicator value of 0 and above 0.5% S a value of 1.

[1] The indicator value is a binary code with the value 1 or 0.

17.9.3 Shell Construction

A block discriminator model was then developed by estimation using the assigned indicator values. Indicator values were estimated in two passes for each major rock type. The inverse distance to a power squared (ID2) was utilized for block indicator assignment. The estimation and search parameters are summarized in Table 17-9 and Table 17-10.

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, some areas in the indicator model for intrusive rocks, greywacke and shale did not receive an indicator value in the first pass. A second indicator pass was performed with search and selection criteria the same as pass one except that a minimum number of four composites was required. This change required two drill holes per estimate instead of three. A flag variable was stored in the model in order to track blocks that were assigned an indicator during this pass.

The resulting block estimates were values between 0 and 1. For the gold indicator values, a threshold was used in intrusive rocks, shale and greywacke to separate non-mineralized material from mineralized material. These indicator values are best considered as probabilities of having a block grade above a specific threshold. 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.

Table 17-9: Discriminator Model Search Parameters

Indicator Pass		Variables		Search Distance (m)			Search Rotation		
Rock	Indicator Pass	Composite Input Variable	Block Model Output Variable	Major Axis Z	Semi-major Axis Y	Minor Axis X	X	Y	Z
Intrusive	1	INTIND	INTIND	175	175	100	024	0	-68
Intrusive	2	INTIND	INTIND	175	175	100	024	0	-68
Shale	3	SEDINT	SEDINT	175	175	100	024	0	-68
Shale	3	SEDINT	SEDINT	175	175	100	024	0	-68
Greywacke	4	SEDINT	SEDINT	175	175	100	024	0	-68

Table 17-10: Discriminator Model Sample Constraints

Indicator Pass		Variables		Sample Selection Criteria		
Rock	Indicator Pass	Composite Input Variable	Block Model Output Variable	Minimum Samples per Estimate	Maximum Samples per Estimate	Maximum Samples per Drill Hole
Intrusive	1	INTIND	INTIND	6	13	2
Intrusive	2	INTIND	INTIND	4	13	2
Shale	3	SEDINT	SEDINT	6	13	2
Shale	3	SEDINT	SEDINT	4	13	2
Greywacke	4	SEDINT	SEDINT	6	13	2
Greywacke	4	SEDINT	SEDINT	4	13	2

The sulphur indicator model was created using the same search parameters and sample constraints as shown in Table 17-9 and Table 17-10; however, estimated indicator values were output to the INTIND_S and SEDINT_S variables in the block model.

Composites inside a model cell that met or exceeded the probability thresholds were classified as within a mineralized envelope. NovaGold confirmed that the models were properly constructed and that the composites were properly back-coded with their respective domain.

17.10 Estimation Parameters

17.10.1 Gold Grade Estimation

Gold grades were then estimated into the block model using an ID3 methodology for the two populations: (1) internal to the mineralized envelope, defined as blocks with indicator values greater than or equal to 50%; and (2) external to the mineralized envelope, defined as blocks with indicator values less than 50%. Composites 6 m long were flagged as being either inside the 0.25 g/t Au indicator threshold (i.e., passing through blocks with an estimated probability of at least 50%) or outside the 0.25 g/t Au indicator threshold.

Estimation of grade into the blocks was broken into five passes based upon increasing search distances. 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. Each successive estimation pass used increasingly longer ranges out to a maximum of 125 m (410 ft). Search ellipse and sample weights were adjusted based on an anisotropic model. Once estimated, blocks could not be overwritten by subsequent estimation passes. Table 17-11 and Table 17-12 summarize the grade interpolation parameters and constraints for the intrusive rocks. Table 17-13 and Table 17-14 document the estimation parameters and search constraints for shale and greywacke. Example bench sections through the gold grade model are shown in Figure 17-10 and Figure 17-11.

Table 17-11: Au Estimation Parameters Internal to Intrusive Indicator

Pass	Input Composite Variable	Output Block Variable	Est. Method	Major Axis	Semi-Major Axis	Minor Axis	Bearing (Z)	Plunge (Y)	Dip (X)	Min Samples/ Estimate	Max Samples/ Estimate	Max/ Drill Hole	Conditions	Resultant Estimate Flag
1	AU_USE	AUE	ID2	3	3	3	024	0	-68	1	99	99	Composites (BIND) and blocks with intrusive indicator INTIND >= 0.5 & assigned rock values 1-55.	1
2	AU_USE	AUE	ID3	75	75	15	024	0	-68	2	3	1	Composites (BIND) and blocks with intrusive indicator INTIND >= 0.5 & assigned rock values 1-55.	2
3	AU_USE	AUE	ID3	75	75	30	024	0	-68	2	3	1	Composites (BIND) and blocks with intrusive indicator INTIND >= 0.5 & assigned rock values 1-55.	3
4	AU_USE	AUE	ID3	125	125	55	024	0	-68	2	3	1	Composites (BIND) and blocks with intrusive indicator INTIND >= 0.5 & assigned rock values 1-55.	4
5	AU_USE	AUE	ID3	30	30	10	024	0	-68	1	3	1	Composites (BIND) and blocks with intrusive indicator INTIND >= 0.5 & assigned rock values 1-55.	5

Table 17-12: Au Estimation Parameters External to Intrusive Indicator

Pass	Input Composite Variable	Output Block Variable	Est. Method	Major Axis	Semi-Major Axis	Minor Axis	Bearing (Z)	Plunge (Y)	Dip (X)	Min Samples/ Estimate	Max Samples/ Estimate	Max/ Drill Hole	Conditions	Resultant Estimate Flag
6	AU_USE	AUE	ID2	3	3	3	024	0	-68	1	99	99	Composites and blocks with intrusive indicator INTIND <0.5 and assigned rock values 1-55	6
7	AU_USE	AUE	ID3	75	75	15	024	0	-68	2	3	1	Composites and blocks with intrusive indicator INTIND <0.5 & assigned rock values 1-55	7
8	AU_USE	AUE	ID3	75	75	35	024	0	-68	2	3	1	Composites and blocks with intrusive indicator INTIND <0.5 & assigned rock values 1-55	8
9	AU_USE	AUE	ID3	125	125	55	024	0	-68	2	3	1	Composites and blocks with intrusive indicator INTIND <0.5 & assigned rock values 1-55	9
10	AU_USE	AUE	ID3	30	30	10	024	0	-68	1	3	1	Composites and blocks with intrusive indicator INTIND <0.5 & assigned rock values 1-55	10

Table 17-13: Au Estimation Parameters Internal to the Shale Indicator

Pass	Input Composite Variable	Output Block Variable	Est. Method	Major Axis	Semi-Major Axis	Minor Axis	Bearing (Z)	Plunge (Y)	Dip (X)	Min Samples/ Estimate	Max Samples/ Estimate	Max/ Drill Hole	Conditions	Resultant Estimate Flag
1	AU_USE	AUE	ID2	3	3	3	024	0	-68	1	99	99	Composites and blocks with SEDINT indicator >= 0.5 & Rock 92	21
2	AU_USE	AUE	ID3	75	75	15	024	0	-68	2	3	1	Composites and blocks with SEDINT indicator >= 0.5 & Rock 92	22
3	AU_USE	AUE	ID3	75	75	30	024	0	-68	2	3	1	Composites and blocks with SEDINT indicator >= 0.5 & Rock 92	23
4	AU_USE	AUE	ID3	125	125	55	024	0	-68	2	3	1	Composites and blocks with SEDINT indicator >= 0.5 & Rock 92	24
5	AU_USE	AUE	ID3	30	30	10	024	0	-68	1	3	1	Composites and blocks with SEDINT indicator >= 0.5 & Rock 92	25

Table 17-14: Au Estimation Parameters External to the Shale Indicator

Pass	Input Composite Variable	Output Block Variable	Est. Method	Major Axis	Semi-Major Axis	Minor Axis	Bearing (Z)	Plunge (Y)	Dip (X)	Min Samples/ Estimate	Max Samples/ Estimate	Max/ Drill Hole	Conditions	Resultant Estimate Flag
6	AU_USE	AUE	ID2	3	3	3	024	0	-68	1	99	99	Composites and blocks with SEDINT indicator <0.5 & Rock 92	26
7	AU_USE	AUE	ID3	75	75	15	024	0	-68	2	3	1	Composites and blocks with SEDINT indicator <0.5 & Rock 92	27
8	AU_USE	AUE	ID3	75	75	30	024	0	-68	2	3	1	Composites and blocks with SEDINT indicator <0.5 & Rock 92	28
9	AU_USE	AUE	ID3	125	125	55	024	0	-68	2	3	1	Composites and blocks with SEDINT indicator <0.5 & Rock 92	29
10	AU_USE	AUE	ID3	30	30	10	024	0	-68	1	3	1	Composites and blocks with SEDINT indicator <0.5 & Rock 92	210

Table 17-15: Au Estimation Parameters Internal to the Greywacke Indicator

Pass	Input Composite Variable	Output Block Variable	Est. Method	Major Axis	Semi-Major Axis	Minor Axis	Bearing (Z)	Plunge (Y)	Dip (X)	Min Samples/ Estimate	Max Samples/ Estimate	Max/ Drill Hole	Conditions	Resultant Estimate Flag
1	AU_USE	AUE	ID2	3	3	3	024	0	-68	1	99	99	Composites and blocks with SEDINT indicator >= 0.5 & Rock 93	31
2	AU_USE	AUE	ID3	75	75	15	024	0	-68	2	3	1	Composites and blocks with SEDINT indicator >= 0.5 & Rock 93	32
3	AU_USE	AUE	ID3	75	75	30	024	0	-68	2	3	1	Composites and blocks with SEDINT indicator >= 0.5 & Rock 93	33
4	AU_USE	AUE	ID3	125	125	55	024	0	-68	2	3	1	Composites and blocks with SEDINT indicator >= 0.5 & Rock 93	34
5	AU_USE	AUE	ID3	30	30	10	024	0	-68	1	3	1	Composites and blocks with SEDINT indicator >= 0.5 & Rock 93	35

Table 17-16: Au Estimation Parameters External to the Greywacke Indicator

Pass	Input Composite Variable	Output Block Variable	Est. Method	Major Axis	Semi-Major Axis	Minor Axis	Bearing (Z)	Plunge (Y)	Dip (X)	Min Sample/ Estimate	Max Samples/ Estimate	Max/ Drill Hole	Conditions	Resultant Estimate Flag
6	AU_USE	AUE	ID2	3	3	3	024	0	-68	1	99	99	Composites and blocks with SEDINT indicator <0.5 & Rock 93	36
7	AU_USE	AUE	ID3	75	75	15	024	0	-68	2	3	1	Composites and blocks with SEDINT indicator <0.5 & Rock 93	37
8	AU_USE	AUE	ID3	75	75	30	024	0	-68	2	3	1	Composites and blocks with SEDINT indicator <0.5 & Rock 93	38
9	AU_USE	AUE	ID3	125	125	55	024	0	-68	2	3	1	Composites and blocks with SEDINT indicator <0.5 & Rock 93	39
10	AU_USE	AUE	ID3	30	30	10	024	0	-68	1	3	1	Composites and blocks with SEDINT indicator <0.5 & Rock 93	310

Figure 17-10: Bench Section Depicting Estimated Measured and Indicated Au Blocks Plotted Against Drill Holes – 0 masl – 50 m Section Width (source Barrick)

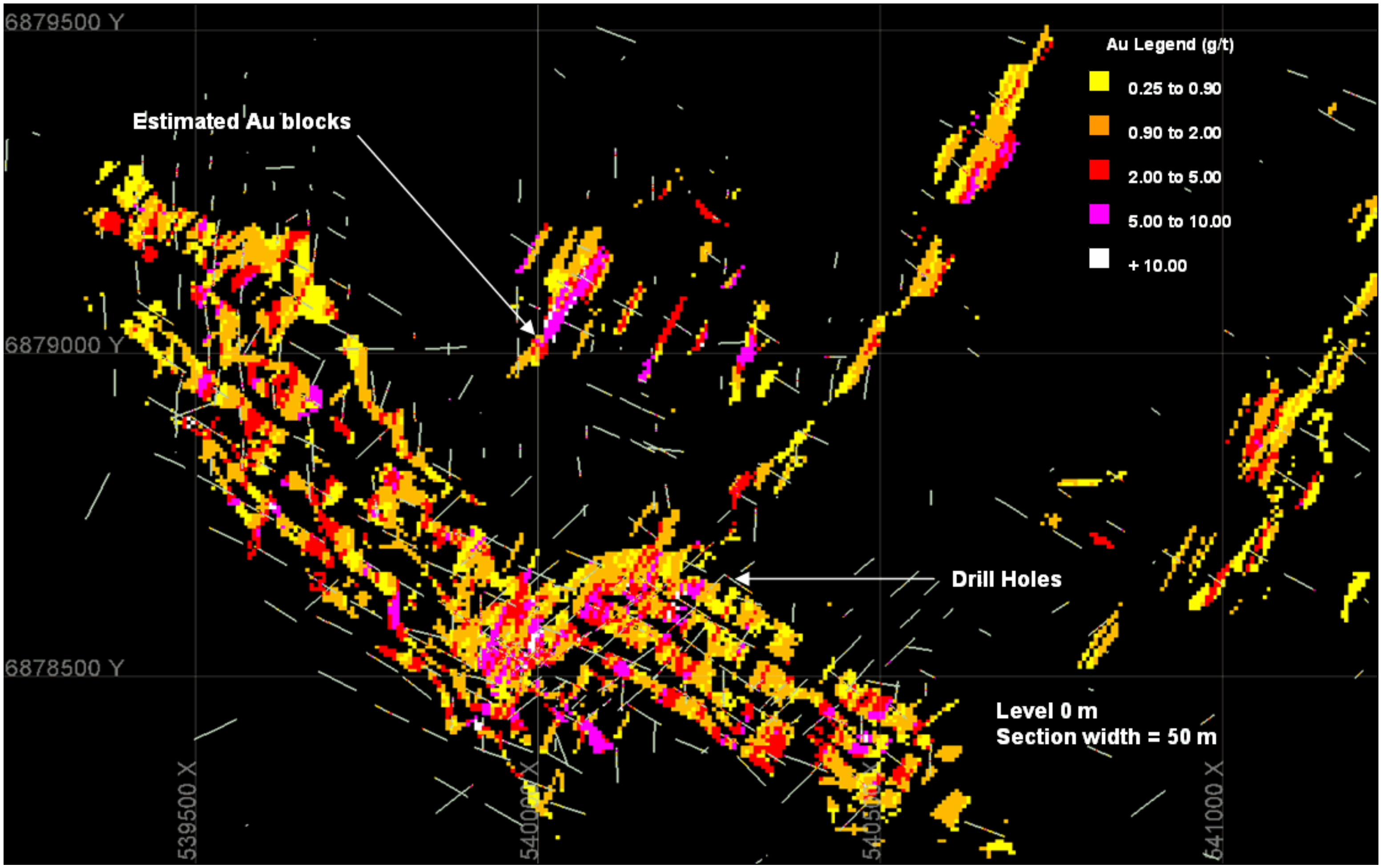
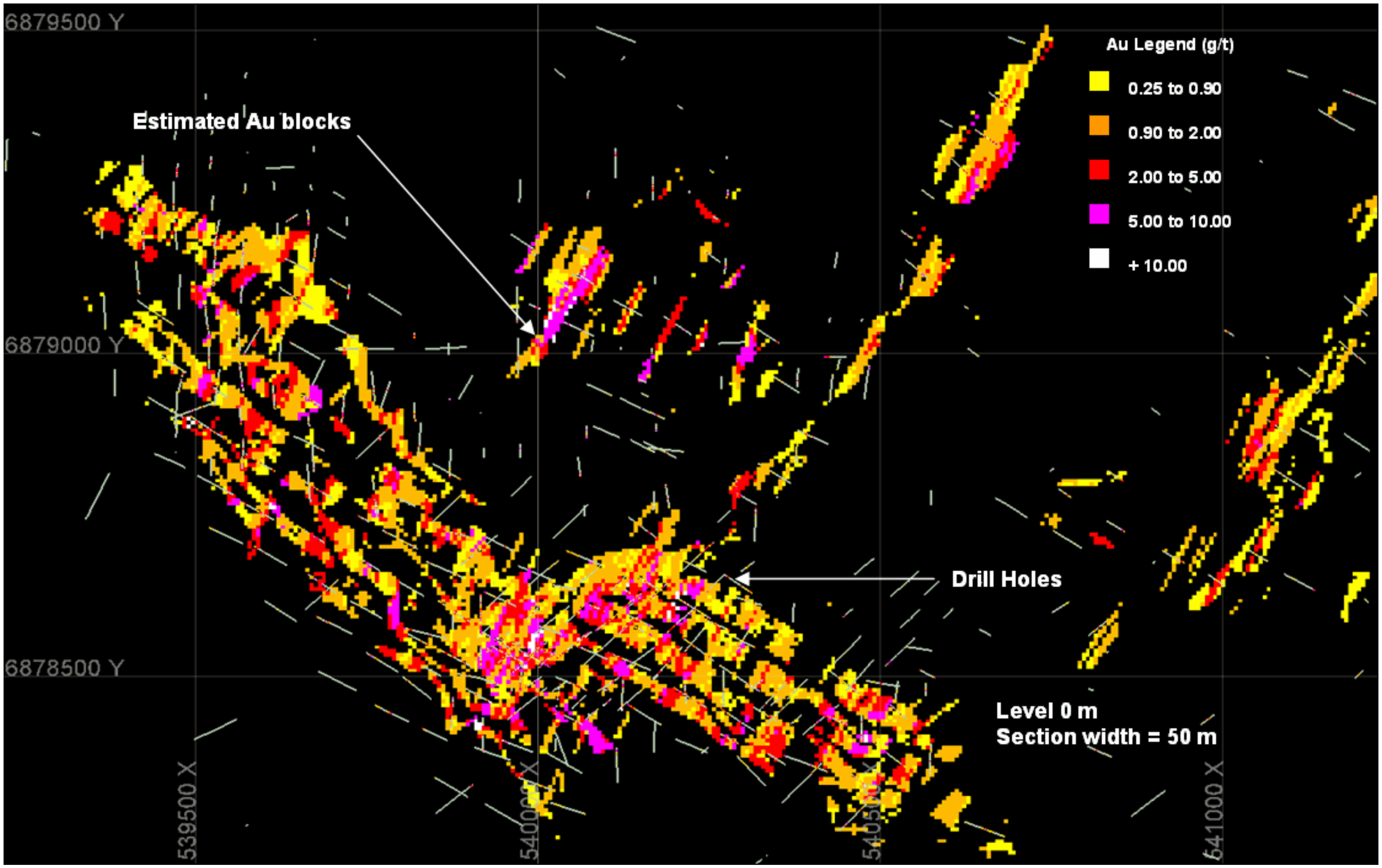


Figure 17-11: Bench Section Depicting Estimated Measured and Indicated Au Blocks Plotted Against Drill Holes – 100 masl – 50 m Section Width (source Barrick)



17.10.2 Sulphur Grade Estimation

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. Composites 6 m long were flagged as being either inside the 0.50% sulphur indicator threshold (i.e., blocks with an estimated probability of at least 50% for intrusive and 50% for shale and greywacke) or outside the 0.50% sulphur indicator threshold.

Sulphur data are less extensive than gold data; therefore, a number of blocks are not estimated during the inverse distance estimation runs. 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 un-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. The formulae used to assign un-estimated sulphur blocks are summarized in Table 17-17.

Table 17-17: Sulphur Regression Formulae

Rock Type	Block Model Rock Code		Regression Formula
	Minimum	Maximum	
'Other' Intrusives	1	5	sulphur = 0.8190*Au^0.3164
RDA dykes	6	11	sulphur = 0.8848*Au^0.2925
RDA sills	12	19	sulphur = 0.7203*Au^0.2623
RDF dykes	20	24	sulphur = 1.3896*Au^0.2473
RDX dykes	25	27	sulphur = 0.8619*Au^0.3584
RDX sills	28	36	sulphur = 0.7629*Au^0.3586
RDXB dykes	37	40	sulphur = 0.6576*Au^0.3748
RDXB sills	41	45	sulphur = 0.7345*Au^0.3115
RDXL dykes	46	48	sulphur = 0.6559*Au^0.2605
RDXL sills	49	51	sulphur = 0.7668*Au^0.2884
Shale	92	92	sulphur = 1.0409*Au^0.3406
Greywacke	93	93	sulphur = 0.9227*Au^0.3973

17.10.3 Grade Estimation for Other Elements

Arsenic, mercury and antimony 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 0.25 g/t gold grade indicator populations. Separate estimation runs were generated for intrusive rocks, shale and greywacke. Composites 6 m long were flagged as being either inside the 0.25 g/t Au indicator threshold (i.e., blocks with an estimated probability of at least 50% for intrusive and 50% for shale and greywacke) or outside the 0.25 g/t Au indicator threshold.

Data for arsenic, mercury and antimony are limited. 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 arsenic, mercury and antimony values to un-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. Formulae used to assign un-estimated arsenic, mercury and antimony blocks are summarized in Table 17-18 to Table 17-20.

Table 17-18: Arsenic Regression Formulae

Rock Type	Block Model Rock Code		Regression Formula
	Minimum	Maximum	
‘Other’ Intrusives	1	5	arsenic = 1625.1*Au^0.4618
RDA dykes	6	11	arsenic = 1842.5*Au^0.4759
RDA sills	12	19	arsenic = 1828.5*Au^0.4541
RDF dykes	20	24	arsenic = 1453.7*Au^0.4054
RDX dykes	25	27	arsenic = 1509.7*Au^0.516
RDX sills	28	36	arsenic = 1958*Au^0.5425
RDXB dykes	37	40	arsenic = 1511.4*Au^0.5284
RDXB sills	41	45	arsenic = 1788.2*Au^0.5699
RDXL dykes	46	48	arsenic = 1650.9*Au^0.4454
RDXL sills	49	51	arsenic = 2475.8*Au^0.5344
Shale	92	92	arsenic = 1693.5*Au^0.7126
Greywacke	93	93	arsenic = 1479.2*Au^0.6026

Table 17-19: Mercury Regression Formulae

Rock Type	Block Model Rock Code		Regression Formula
	Minimum	Maximum	
‘Other’ Intrusives	1	5	mercury = 0.8172*Au^0.0832
RDA dykes	6	11	mercury = 1.4318*Au^0.2591
RDA sills	12	19	mercury = 1.1658*Au^0.1852
RDF dykes	20	24	mercury = 1.8751*Au^0.2441
RDX dykes	25	27	mercury = 1.7817*Au^0.2775
RDX sills	28	36	mercury = 1.1122*Au^0.1713
RDXB dykes	37	40	mercury = 1.481*Au^0.1772
RDXB sills	41	45	mercury = 1.1273*Au^0.2093
RDXL dykes	46	48	mercury = 0.9299*Au^0.106
RDXL sills	49	51	mercury = 0.8941*Au^0.0869
Shale	92	92	mercury = 1.1091*Au^0.1709
Greywacke	93	93	mercury = 1.2638*Au^0.1912

Table 17-20: Antimony Regression Formulae

Rock Type	Block Model Rock Code Minimum	Block Model Rock Code Maximum	Regression Formula
'Other' Intrusives	1	5	antimony = 15.711*Au^0.2532
RDA dykes	6	11	antimony = 21.859*Au^0.2295
RDA sills	12	19	antimony = 22.391*Au^0.338
RDF dykes	20	24	antimony = 46.548*Au^0.333
RDX dykes	25	27	antimony = 24.714*Au^0.254
RDX sills	28	36	antimony = 24.961*Au^0.4095
RDXB dykes	37	40	antimony = 24.653*Au^0.275
RDXB sills	41	45	antimony = 20.141*Au^0.2973
RDXL dykes	46	48	antimony = 25.098*Au^0.3221
RDXL sills	49	51	antimony = 21.306*Au^0.3204
Shale	92	92	antimony = 22.581*Au^0.4417
Greywacke	93	93	antimony = 28.796*Au^0.364

17.10.4 CO_2, Ca, and Mg Assays

Values for carbon dioxide, calcium and magnesium were estimated separately and incorporated into the current block model. Estimates of CO_2 and Ca are used for waste rock management and environmental assessment. Magnesium content is used for metallurgical models.

17.11 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 Creek, as described below:

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) from carbonate minerals was estimated from:

NP_{CO_3} = 0.94.NP + 0.98

To avoid a slight bias at low NP below 16.3 kg $CaCO_3$/t resulting from the regression equation, the calculated NP_{CO_3} should not exceed analytical NP when NP is below 16.3 kg $CaCO_3$/t. Therefore, the following rules were applied to the calculation:

If NP≤16.3 kg $CaCO_3$/t: NP_{CO_3} = NP
If NP>16.3 kg $CaCO_3$/t: NP_{CO_3} = 0.94.NP + 0.98

The variables NPCO3 and AP were estimated for each block separately and were then used to calculate acid rock drainage (ARD) potential. ARD was modelled using the ratio $NPCO_3/AP$. Recommended ARD categories were assigned to each block according to calculated ARD potential, as shown in Table 17-21.

Table 17-21: ARD Categories

Category	Block Model Value	Category Description	NP_{CO3}/AP Range
A	1	Very unlikely to generate ARD	$NP_{CO3}/AP > 2$
B	2	Unlikely to generate ARD	$1.4 < NP_{CO3}/AP \leq 2$
C	3	Potentially acid generating but with very long delays (several decades) to onset of ARD	$1.0 < NP_{CO3}/AP \leq 1.4$
D	4	Potentially acid generating in the life of the mine (possibly less than a decade)	$0.2 < NP_{CO3}/AP \leq 1.0$
E	5	Potentially acid generating but with shorter delays to onset (less than a few years)	$NP_{CO3}/AP \leq 0.2$

The block model estimates for arsenic and sulphur values were used to calculate the ratio of arsenic to sulphur (As/S) for each block. The ARD potential and As/S ratio were then used to classify blocks into seven waste rock management categories (WRMC) subdivided into potentially acid generating (PAG) and non-PAG groups. These WRMC codes are summarized in Table 17-22. The two non-PAG categories (A and B) are each split into two sub-categories to allow for arsenic leaching. The (ii) sub-categories indicate rock that has low potential to generate ARD but could result in arsenic leaching above 1 mg/L.

Table 17-22: Recommended Waste Rock Management Categories

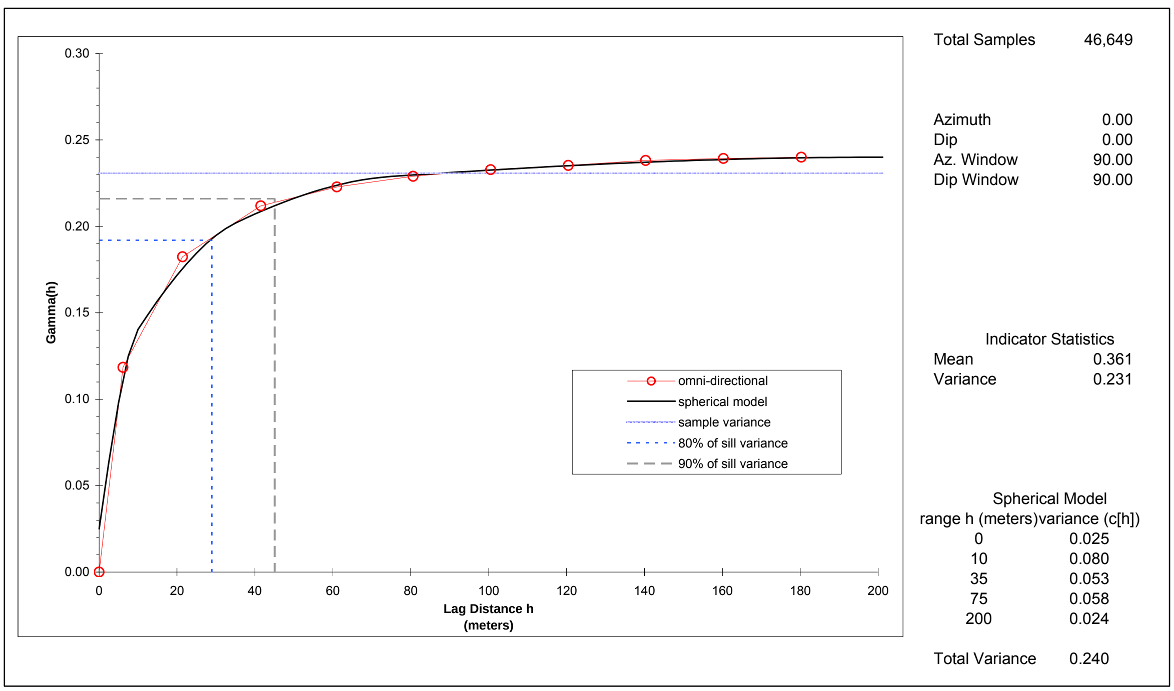
Category	Block Model Value	Category Description	NP_{CO3}/AP Range	As/S (As in mg/kg and S in %)
A(i)	1	Very unlikely to generate ARD, and "low" arsenic leaching.	$NP_{CO3}/AP > 2$	As/S<196 and As<250
A(ii)	2	Very unlikely to generate ARD, and arsenic leaching potentially significant.	$NP_{CO3}/AP > 2$	As/S>196 or As>250
B(i)	3	Unlikely to generate ARD and "low" arsenic leaching".	$1.4 < NP_{CO3}/AP \leq 2$	As/S<196 and As<250
B(ii)	4	Unlikely to generate ARD and arsenic leaching potentially significant.	$1.4 < NP_{CO3}/AP \leq 2$	As/S>196 or As>250
C	5	PAG but with very long delays (several decades) to onset of ARD.	$1.0 < NP_{CO3}/AP \leq 1.4$	All
D	6	PAG in the life of the mine (possibly less than a decade).	$0.2 < NP_{CO3}/AP \leq 1.0$	All
E	7	PAG but with shorter delays to onset (less than a few years).	$NP_{CO3}/AP \leq 0.2$	All

17.12 Variography

The 6 m composites were used to develop indicator and relative pair-wise variograms. Relative pair-wise variograms were generated for all sample data and by domain using orientations along the average strike and dip of the mineralized zones. This orientation was identified both geologically and through 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 variograms were generated at 0.25 g/t Au for the 6 m composites. The correlograms at 0.25 g/t Au were fitted with a spherical model as shown in Figure 17-12. Ranges of 30 m (98 ft) and 45 m (148 ft) can be observed at 80% and 90% of the total sill variance. Barrick used the variogram solely for classification of resources, described in Section 17.12.

Figure 17-12: 0.25 g/t Au Indicator Variogram for 6 m Composites



17.13 Model Validation

To validate the gold and sulphur grade estimates a number of checks were carried out. These include:

- Visual inspection of gold estimation results on plans and sections

- Comparison of estimated and nearest neighbour statistics
- A change of support check
- A check of "script files" used within the Vulcan™ software system
- A check of summation of the mineral resources

Validation by Visual Inspection on Sections and Plans

Estimated grades, resource classification, drill hole assays and mineralized shells were inspected on-screen by NovaGold. Overall, NovaGold found a reasonable agreement between the assays and the estimated gold and sulphur grades. The inferred estimate appears to be well constrained and no significant areas of overestimation were observed.

Comparison to a Nearest Neighbour Estimate

For the purposes of validation, nearest neighbour (NN) models with the 6 m gold and sulphur composites were generated.

Table 17-23 and Table 17-24 summarize the comparisons between nearest neighbour and estimated gold and sulphur grades. The results are tabulated at a zero cutoff. The differences shown are well within 5%, which NovaGold considers the accepted limit with the exception of the gold estimate of rock group "rck01_other". This exception represents less than 0.1% of the global tonnage.

The comparison with the nearest neighbour statistics is considered acceptable for the grade estimation model.

Swath Plots

The term swath plot refers to an assessment of local trends. Groups of estimates and data are compared for different sections or plans, i.e., swaths of data, and these are compared to ensure that the estimation methodology honours the trends in the data.

For this review, the program *tm_1davg.exe* was used to determine the trends in the ID3 and NN models and the principle directions of easting, northing and elevation were examined by plotting graphs of grade versus distance.

Figure 17-13 shows an example of the trend check results for rck02_rda intrusive. Note that both the red and green lines display similar trends. A complete set of the trend plots is provided in Appendix D.

Based on this assessment, the DC7a model appears to honour the local trends in the data.

Change of Support Check

Change of support issue becomes important when a cut-off is applied to a distribution. The validation of a model is to determine if it is conditionally unbiased and properly reflect tonnes and grade above a cut-off, or range of cut-offs. The range of economic cut-offs considered range between 0.7 and 1.5 g/t gold.

One measure of the change of support in the model is the coefficient of variation (CV). NovaGold calculated the theoretical CV for 6 m x 6 m x 6 m blocks and compared to the block model distribution using a Discrete Gaussian Method (DGM), which adjusts the variance of the point information (NN estimate) to represent the variance of the proposed 6 m by 6 m by 6 m SMU. The calculation of the expected CV can be made taking into account "perfect selection". In perfect selection, the grade of the block is known "perfectly" and the discriminating decision of ore or waste is made without error.

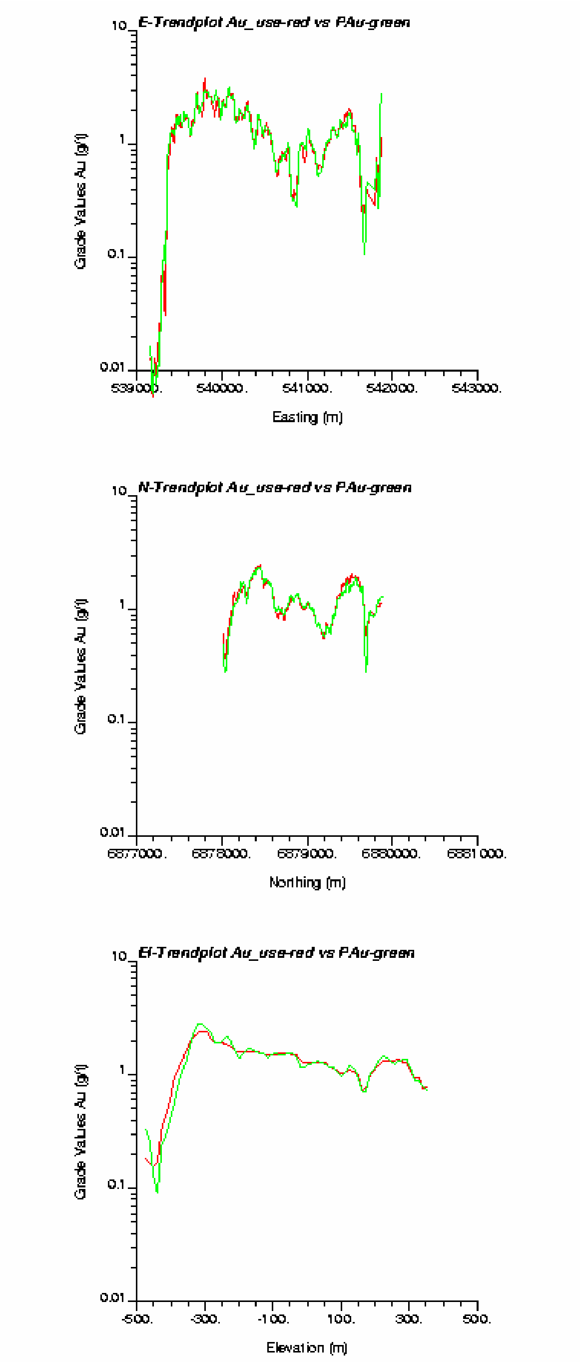
Table 17-23: Comparison of Estimated Gold Grade (au_use) to Nearest Neighbour (pau) Model

	Au_use (g/t)					Pau (g/t)					Au_use/Pau	Au_use/Pau
Rock_mine	**no blks**	**mean**	**std dev**	**cv**	**max**	**no blks**	**mean**	**std dev**	**cv**	**max**	**mean**	**CV**
rck01_other	4,409	1.250	1.229	0.983	9.010	4,409	1.159	1.419	1.224	11.420	1.079	0.803
rck02_rda	433,947	1.252	1.684	1.345	20.520	433,947	1.257	1.985	1.580	26.620	0.996	0.852
rck03_rdf	11,223	1.330	1.331	1.001	8.590	11,223	1.318	1.644	1.248	10.230	1.009	0.802
rck04_rdx	594,596	1.179	1.678	1.423	27.950	594,596	1.167	1.971	1.689	28.850	1.010	0.843
rck05_rdxb	369,325	0.979	1.561	1.595	28.460	369,325	0.990	1.894	1.913	28.460	0.988	0.834
rck06_rdxl	137,641	1.369	1.351	0.987	17.450	137,641	1.365	1.553	1.138	24.670	1.003	0.867
rck07_shale	3,728,838	0.220	0.882	4.000	22.660	3,728,838	0.216	0.978	4.534	22.660	1.022	0.882
rck08_gwk	1,085,041	0.176	0.829	4.701	24.910	1,085,041	0.180	0.928	5.168	24.920	0.982	0.910
All	**6,365,020**	**0.436**	**1.180**	**2.705**	**28.460**	**6,365,020**	**0.436**	**1.333**	**3.062**	**28.850**	**1.001**	**0.884**

Table 17-24: Comparison of Estimated Sulphur Grade (su_use) to Nearest Neighbour (psu_use) Model

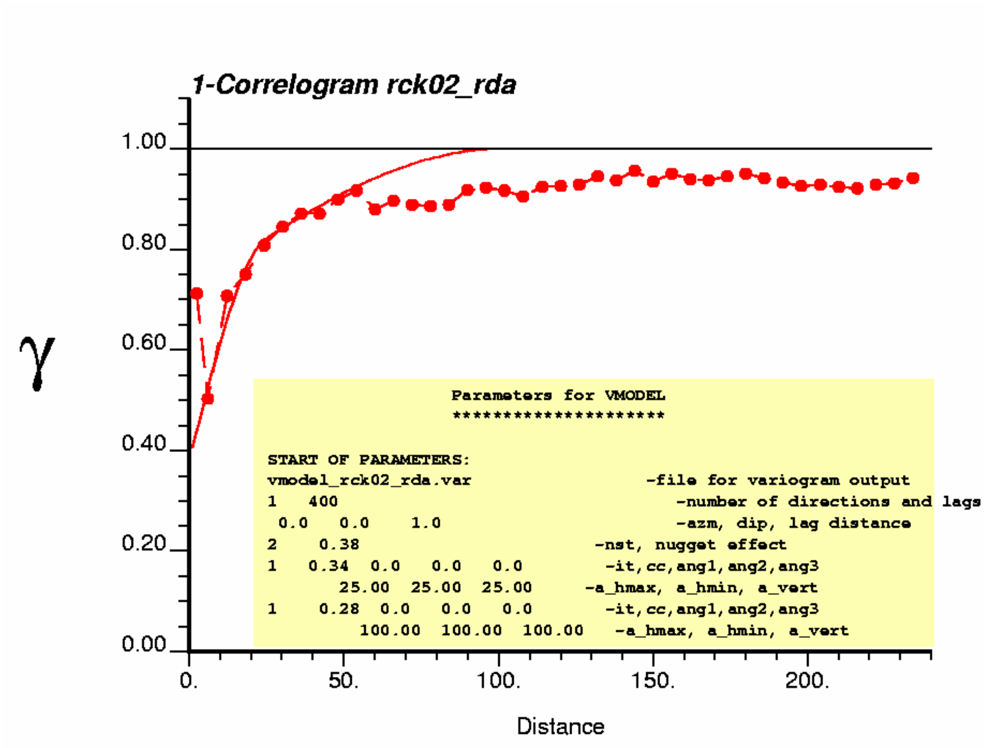
	Su_use % (S)					Psu_use % (NN)					S/NN	S/NN
Rock_mine	**no blks**	**mean**	**std dev**	**cv**	**max**	**no blks**	**mean**	**std dev**	**cv**	**max**	**mean**	**CV**
rck01_other	4,408	0.939	0.408	0.434	2.656	4,408	0.921	0.469	0.509	2.953	1.020	0.853
rck02_rda	415,741	0.838	0.467	0.558	6.072	415,741	0.843	0.560	0.664	6.072	0.994	0.840
rck03_rdf	11,108	1.198	0.431	0.360	2.914	11,108	1.152	0.559	0.485	4.473	1.040	0.742
rck04_rdx	575,595	0.849	0.539	0.635	8.519	575,595	0.860	0.601	0.699	8.519	0.986	0.909
rck05_rdxb	336,631	0.753	0.485	0.645	5.680	336,631	0.757	0.547	0.722	5.680	0.993	0.893
rck06_rdxl	127,460	0.816	0.365	0.448	5.130	127,460	0.814	0.446	0.549	5.130	1.002	0.816
rck07_shale	3,038,497	0.452	0.557	1.231	5.886	3,038,497	0.450	0.601	1.336	5.886	1.004	0.922
rck08_gwk	982,375	0.420	0.508	1.209	5.997	982,375	0.422	0.556	1.320	6.012	0.997	0.916
All	5,491,815	0.544	0.556	1.021	8.519	5,491,815	0.544	0.608	1.117	8.519	0.999	0.915

Figure 17-13: Trend Plot of RCK02_RDA Gold Grades ID3 (au_use) versus NN (pau)



To determine the spatial continuity, correlograms were computed for each rock type that has sufficient 6 m composites. Rock types *rck01_other* and *rck03_rdf* only have 60 and 69 6m composites, respectively, so correlograms were not calculated. The correlograms were computed in the GSLIB program *gamv.exe* and fit using *vmodel.exe*. An example correlogram provided in Figure 17-14 shows the fitted model developed for rck02_rda.

Figure 17-14: Correlogram of 6 m Composite Gold Grade for RCK02_RDA Intrusive



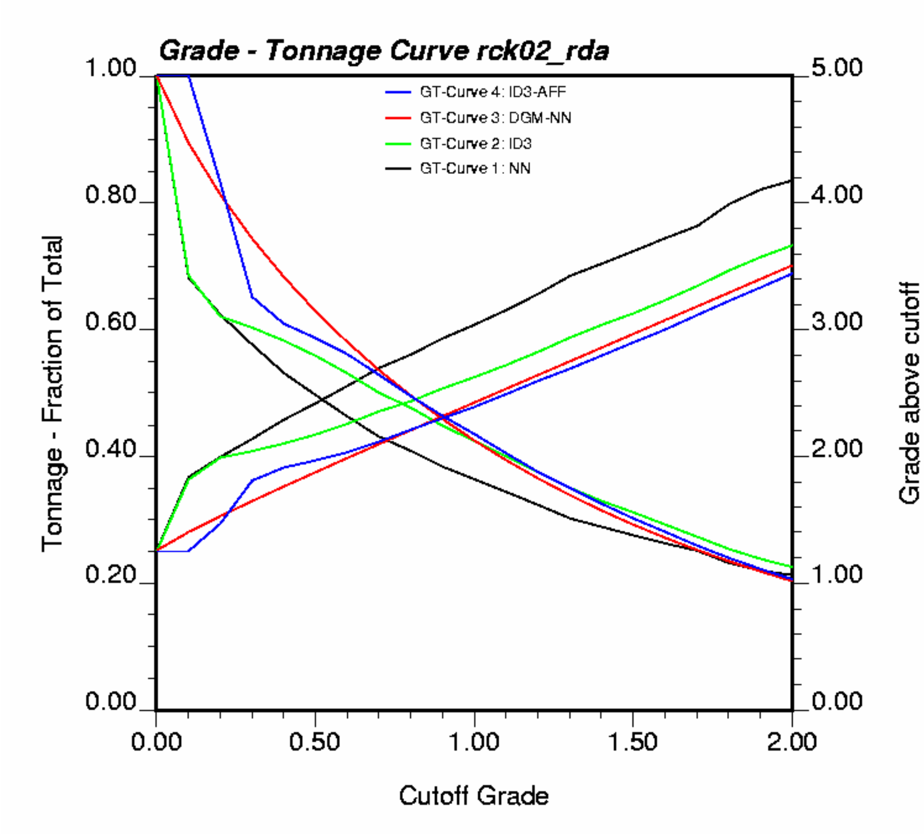
The fitted model is then used to calculate the variance of a point within a block using the program *gammabar.exe*. Using Krige's relationship, the variance of a block within the deposit can be determined along with the block variance. The block variance is then used as a parameter for the DGM program *dgm.exe*.

The ID3 grade estimates have been placed into a block model with a grid of 6 m by 6 m by 6 m and then an economic cut-off is applied to the model. In theory then, the SMU-sized ID3 block estimates should have the same distribution as the DGM-NN. If this were true, then the CV ratios should be close to 1.0. None of these ratios are close to 1.0; in fact, they are greater than 1.0, indicating that the ID3 gold grade estimation model is more variable than it should be for a model reflecting the appropriate change of support.

The impact of having a grade estimation model that is too variable is that for any given cut-off, the tonnes and grade will not reflect the tonnes and grade of the SMU blocks and will likely overestimate the grade and underestimate the tonnes. A block model that is too variable is therefore conditionally biased.

Figure 17-15 displays grade-tonnage curves for NN (black), ID3 (green) and the DGM-NN (red). The black line for tonnes and grade is equivalent to separating ore from waste at 6 m drill core SMU, not the block SMU. The DGM-NN lines show the distribution of the SMU blocks as determined from the change of support method. The ID3 green line is directly from the ID3 grade estimates.

Figure 17-15: Grade – Tonnage Curve for Gold Estimate for RCK02_RDA Intrusive



Using the 1.2 g/t Au cut-off, the ID3 model predicts a proportional tonnage of 0.375 grading 2.83 g/t Au, while the DGM-NN predicts a tonnage of 0.375 with a grade of 2.60 g/t Au. While the tonnage is the same at this cut-off, the grade is about 7% more for the ID3 model than the DGM-NN. This is entirely due to the difference in the variability of the ID3 model.

Grade-tonnage curves for the other rock types show a similar behaviour to that of the grade-tonnage curve of rck02_rda (see Appendix E). For all rock types, the ID3 model overestimates the grade and underestimates the tonnes at the range of economic cut-offs.

It is possible to adjust the variance of the ID3 grade estimates to better match those predicted by DGM-NN model. The blue lines shown in Figure 17-15 are variance-corrected ID3 grade estimates for rck02_rda. As the amount of the adjustment is small (for the intrusive), the affine correction method was used.

In NovaGold's opinion, the gold and sulphur grade estimates are under-smoothed relative to the expected variability of perfect selection. It is possible that a mine plan generated from this block model will overestimate the gold grade. The Donlin Creek LLC is aware of the issue and is working internally and with outside experts to quantify the materiality and assess the impact of alternate estimation plans that utilize more composite samples. NovaGold agrees with this approach and suggests that mine planning and production scheduling regard this model as being an optimistic case.

A Review of Vulcan "Script Files"

NovaGold inspected the "script-files" created for each gold estimation run to ensure that the correct search, composite and block selection, and parameters were used to estimate grades in the block model. No errors were found in the run-files.

A Summation Check

NovaGold independently checked the summations for the grade and tonnage figures. No errors were found.

17.14 Resource Classification and Summary

The logic for mineral resource classification of Donlin Creek is consistent with the CIM definitions referred to in NI 43-101. The current level of drilling allows a reasonable assumption of geologic and grade continuity for the indicated mineral resource category. The Indicated mineral resource category is supported by the present drilling grids over the ACMA and Lewis deposits (nominal 25 m to 35 m). The measured mineral resource category is supported only in blocks pierced by exploration drill holes. Inferred mineralization is limited to a reasonable expectation of mining by a conceptual open pit shell using a metal price of US$650 per ounce of gold and recent estimates of mining, geotechnical and metallurgical parameters.

17.14.1 Resource Classification

The resource model was classified using distance to nearest composite as stored in the model blocks during the nearest-neighbour grade estimate. Classification distances are based on the 80% and 90% of variance from the omni-directional indicator variogram model described in Section 17.12. The classification methodology is provided in Table 17-25.

Table 17-25: Donlin Creek Resource Classification

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

17.14.2 Resource Tabulation

The mineralization of the Donlin Creek project, effective date as of February 5, 2008, is classified as measured, indicated and inferred mineral resources. The classified mineral resources are shown in Table 17-26. The mineral resource has been constrained within a conceptual pit based on US$650 per ounce of gold and using recent estimates of mining, geotechnical and metallurgical parameters. Barrick's Technical Services Evaluations Group estimated a variable net smelter return (NSR) cut-off grade based on recent estimates of mining costs, processing costs (dependent upon sulphur content), selling costs and royalties, rather than gold grade alone. The

NSR cut-off equates to approximately 0.8 g/t gold at the average estimated sulphur grade of 1.12%.

The mineral resource estimates for Donlin Creek project show an increase in resources over the July 2006 mineral resource estimates. This increase is the result of additional drilling.

Table 17-26: Donlin Creek Project Mineral Resource Summary(1)(2)(3)(4)
Effective Date February 5, 2008

	Tonnes (M)	Au (g/t)	Contained Au (Million oz)
US$0.01 NSR/t Cut-off			
Measured Mineral Resource	4.3	2.73	0.38
Indicated Mineral Resource	367.4	2.46	29.0
Measured + Indicated Mineral Resources	**371.7**	**2.46**	**29.38**
Inferred Mineral Resource	**46.5**	**2.31**	**3.46**

(1) Mineral resources that are not mineral reserves do not have demonstrated economic viability
(2) Rounding differences may occur.
(3) Resources are constrained within a Lerchs-Grossman (LG) open-pit shell using the long-term metal price assumption of US$650/oz of gold. Assumptions for the LG shell included pit slopes variable by sector and pit area: mining cost is variable with depth, averaging US$1.57/t mined; process cost is calculated as the percent sulphur grade x US$2.09 + US$10.91; general and administrative costs, gold selling cost and sustaining capital cost are reflected on a per tonne basis. Average estimated sulphur grade is 1.12%. Based on metallurgical testing, gold recovery is assumed to be 89.5%. Blocks with a cost margin of US$0.01/t or higher above the variable cut-off were reported.

NovaGold reviewed in detail the implementation of the classification criteria, and subsequent flagging of blocks. NovaGold also reviewed the tabulation of the resource blocks within the various domains using an independent computer program. No errors or omissions were found. In the opinion of the author, resource classification criteria are reasonable. NovaGold is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issue that would materially impact the estimation of resources. NovaGold is also not aware of any mining, metallurgical, infrastructure or other relevant factors that would materially impact the estimation of resources.

18.0 OTHER RELEVANT DATA AND INFORMATION

There are no other relevant data and information available at the time of this report.

19.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

A Preliminary Economic Assessment by AMEC was completed in March 2002 (Juras and Hodgson, 2002). NovaGold commissioned SRK to complete a Preliminary Economic Assessment in September 2006 (Dodd et al., 2006). Both of these studies are not reliable due to changes in mineral resources. No other studies have been finalized for the Donlin Creek project.

The project is not presently a development or production property.

20.0 INTERPRETATION AND CONCLUSIONS

- The geology of the Donlin Creek project and mineralization controls are well understood.
- The mineralization has been delineated by core and RC samples collected from drill holes. Key quality assurance and control steps have ensured the validity of the assay database. Auditing of the drill hole database has verified the integrity of the data used to estimate resources.
- Metallurgical testing completed to date appears to support the selected processing sequence.
- Probability shells were constructed to partition intrusive and sediments into mineralized and unmineralized domains during gold and sulphur estimation.
- Gold and sulphur grades were estimated by ID3. NovaGold has validated the resource estimate and believes it to have been completed to industry standard.
- The mineral resource at Donlin Creek is classified according to the CIM definitions referred to in National Instrument 43-101.
- In the opinion of the author the underlying data are adequate to define the mineral resources that are the subject of this report.

21.0 RECOMMENDATIONS

The project is of sufficient quality to warrant a feasibility study. The author is not aware of the exact cost of such a study, but assumes it could exceed two million dollars.

22.0 REFERENCES

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22.1 Glossary

Adit: An opening driven horizontally into the side of a mountain or hill for providing access to a mineral deposit

Aeromagnetic survey: A geophysical survey using a magnetometer on board or towed behind an aircraft.

Airborne survey: A survey made from an aircraft to obtain photographs, or measure magnetic properties, radioactivity, and so on.

Anomaly: Any departure from the norm which may indicate the presence of mineralization in underlying bedrock.

Anticline: an area of rock deformation that involves a downward slope to either side. In an exposed eroded anticline the oldest rock layers are in the center and the rocks on either side dip or slope away from the center of the structure. The rock at the center of the anticline is higher than the same stratum elsewhere. Anticlines typically form during crustal warping as the result of compression concurrent with orogenic mountain building.

Assay: A chemical test performed on a sample of ores or minerals to determine the amount of contained metal.

Base metal: A base metal is a common or at least inexpensive metal. Frequently, the term is used to refer to those that oxidize or corrode relatively easily, and react variably with dilute hydrochloric acid to form hydrogen. Examples include iron, nickel, copper, lead and zinc.

Bedding: The arrangement of sedimentary rocks in layers.

Breccia: A rock in which angular fragments are surrounded by a mass of fine-grained minerals.

Channel sample: A sample composed of pieces of vein or mineral deposit that have been cut out of a small trench or channel, usually about 10 cm wide and 2 cm deep

Cordillera: The continuous chain of mountain ranges on the western margins of North and South America.

Diamond drill: A rotary type of rock drill that cuts a core of rock that is recovered in long cylindrical sections, two cm or more in diameter.

Electromagnetic Survey: A geophysical survey method which measures the electromagnetic properties of rocks.

Footwall: The rock on the underside of a vein or ore structure.

Fold: geological process that causes a bend in a stratum of rock

Geochemical Survey: a study or sampling program undertaken to measure the quantities of selected elements in a certain area

Grab sample: A sample from a rock outcrop that is assayed to determine if valuable elements are contained in the rock. A grab sample is not intended to be representative of the deposit, and usually the best-looking material is selected.

Grade: Percentage of a metal or mineral composition in an ore or processing product from mineral processing.

Hangingwall: The rock on the upper side of a vein or ore deposit.

Host Rock: The rock within which the mineralization or ore occurs

Induced polarization: A method of ground geophysical surveying employing an electrical current to determine indications of mineralization.

Partnership: a business undertaking entered into by two or more parties which is intended to terminate upon the completion of a specific project

Landsat: The generic name for a series of natural resource scanning satellites launched by the United States beginning in 1972.

Landsat TM: Landsat Thematic Mapper. Earth observation satellite with seven bands at 30 m spatial resolution

Mapping: Recording geological features on a map

Mine: An opening or excavation in the earth for the purpose of extracting minerals.

Mineral: A naturally occurring, solid, inorganic element or compound, with a definite composition or range of compositions, usually possessing a regular internal crystalline structure.

Mineral Resource: A concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a mineral resource (deposit) are known, estimated or interpreted from specific geological evidence and knowledge.

Mineral Reserve: A mineral reserve is the economically mineable part of a measured or indicated mineral resource demonstrated by at least a preliminary feasibility study. This study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A mineral reserve includes diluting materials and allowances for losses that may occur when the material is mined.

Occurrence: The area where a particular mineral is found

Ore: A natural deposit in which a valuable metallic element occurs in high enough concentration to make mining economically feasible. The term is proscribed under NI 43-101.

Overburden: Material of any nature, consolidated or unconsolidated, that overlies a deposit of ore that is to be mined.

pH: The negative logarithm of the hydrogen ion concentration, in which pH = -log [H+]. Neutral solutions have pH values of 7, acidic solutions have pH values less than 7, and alkaline solutions have pH values greater than 7.

Prospect: A mining property, the value of which has not been determined by exploration

Reconnaissance: a general examination or survey of a region with reference to its main features, usually preliminary to a more detailed survey

Resistivity survey: A geophysical technique used to measure the resistance of a rock formation to an electric current.

Rift: a zone between two diverging tectonic plates

Royalty: a percentage interest in the value of production from a lease that is retained and paid to the mineral rights owner

Scarp: A steep cliff or steep slope, formed either as a result of faulting or by the erosion of inclined rock strata

Schist: Metamorphic rock dominated by fibrous or platy minerals. Schist has a schistose plain of cleavage, and is product of regional metamorphism

Sedimentary Rock: A rock formed from the consolidation of loose sediment or from chemical precipitation, such as sandstone and limestone

Shear Zone: A planar zone of weakness, similar to a fault, but consisting of several parallel displacement zones usually over a greater width than a single fault

Tectonic: pertaining to the rock structures and external forms resulting from the deformation of the Earth's crust.

Trench: long, narrow excavation dug through overburden or blasted out of rock to expose a vein or ore structure

Vein: A mineralized zone having a more or less regular development in length, width, and depth to give it a tabular form.

Zone: An area of distinct mineralization.

22.2 Abbreviations and Units of Measure

Acid mine drainage	AMD
Annum (year)	a
Average	AV
Best value	BV
Canadian Securities Administration	CSA
Centimetre	cm

Check Samples	CS
Coefficient of determination	R^2
Confidence interval	CI
Copper	Cu
Cubic centimetre	cm^3
Cubic feet per minute	cfm
Cubic metre	m^3
Copper Equivalent	CuEQ
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Degree	°
Diameter	ø
Dry metric tonne	dmt
Elevation (metres)	el
Environmental Evaluation	EA
Environmental Impact Assessment	EIA
Gram	g
Global positioning system	GPS
Gold	Au
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inductively-coupled plasma (Chemical Analysis Instrument)	ICP
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilogram per year	kg/a
Kilometre	km
Lead	Pb
Less than	<
Litre	L
Litres per minute	L/m
Mass spectrometer (Analysis Instrument)	MS
Mass submerged in water	Mw
Measure of acidity or alkalinity of a solution	pH
Metre	m
Metres above sea level	masl
Metres per minute	m/min
Metres per second	m/s
Micrometre (micron) 10^{-6}m	µm
Milliamperes	mA
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm

Million	M
Million Dollars (US)	US$M
Million tonnes	Mt
Ministry of Energy and Mines	MEM
Minute (plane angle)	'
Minute (time)	min
Month	mo
Natural Source Audio Magnetotelluric	NSAMT
Ounce	oz
Optical Emission Spectroscopy (Analysis Instrument)	OES
Overall bias	OABias
Parts per billion	ppb
Parts per million	ppm
Percent	%
Preliminary Economic Assessment	PEA
Probability Assisted Constrained Kriging	PACK
Quality Assurance and Quality Control	QA/QC
Reverse circulation	RC
Rock mass rating	RMR
Rock quality designator	RQD
Second (plane angle)	"
Second (time)	s
Silver	Ag
Standard deviation	SD
Specific gravity	SG
Square centimetre	cm^2
Square kilometre	km^2
Square metre	m^2
Thermal Imaging	TM
Thousand tonnes	kt
Tonne (1,000 kg)	t
Tonnes (1,000 kg) per annum	t/a
Tonnes (1,000 kg) per day	t/d
Underground	UG
US dollar	US$
Universal Transverse Mercator (co-ordinate system)	UTM
X-Ray diffraction	XRD
Year (annum)	a

23.0 DATE AND SIGNATURE PAGE

The effective date of this Technical report, entitled "Donlin Creek Project NI 43-101 Technical Report" is February 5, 2008.

Signed,

Kevin Francis

NovaGold Resources Inc.

Date: February 22, 2008

APPENDIX A

Core Logging Manual

Compilation of 2007 Core Logging, Sampling And Sample Processing Procedures

Including QA/QC Protocols

Brian Flanigan

August 16th 2007

Table of Contents

Core Logging

2007 Donlin Creek Project

Core Logging Manual

Brian Flanigan

April 19, 2007 revision

Introduction

Core logging is the process of documenting geologic features in a format useful for interpretations to be used for various related projects. In our case, this information will be used for resource calculations, metallurgical studies, geotechnical analysis and a host of related mine development activities.

Prior to the mid 1980's, core logging was nearly as much art as it was science, similar in many respects to cartography (which is rapidly becoming a lost art as well). The advent of computers and computer software capable of plotting geologic information into visual representations, such as maps and cross-sections, has made core logging more of a data-recording driven process than an artistically appealing one. Although the early artistic practices of core logging have nearly faded from existence, the scientific recording of observations has remained; albeit the format in which data is recorded has changed dramatically. *The key purpose of core logging today is the same as it was in the past: The objective recording of observations without interpretive influence from previously formed conclusions or biases. The best interpretations stem from the accurate recording of non-tainted observations.*

Many "old school" core loggers shudder either in disgust or in anguish at what core logging has become. This decade's geologists will likely never know the sense of pride and accomplishment from creating a visually appealing and accurate graphic log from which interpretations can be readily shaped. Fortunately, there are many other gratifying aspects of core logging and related geological work.

Although "old school" geologists might despair, current logging methods and practices offer many advantages; Efficiency is greatly increased and consistency between geologists is enhanced. Also, tools of the digital age make creating maps, cross-sections, and graphic logs almost instantaneous. It is much easier to test different hypotheses and draw conclusions when a complete data set is as close as fingertips on a keyboard. Thus, it should come as no surprise that the Donlin Project has adopted more modern core logging methods. Visual representations of the collected data are now created by geologists carefully directing high-end geologic modeling computer software.

The following procedures are a distillation of the work of the author and many other geologists including: Scott Petzel, Joe Piekenbrock, Stan Dodd, Chris Gierymski, Fawn Glassburn, Heidi Drexler; and the 2007 core loggers: Kyle Linebarger, Leif Bailey, Alejandro Ly, Jennifer Hansom, Orion George, Gabe Kassos, and Scott Cereghino. Thanks also to Rich Harris and Kerry Alder for the editorial comments on the earlier drafts.

Core Logging Procedural Outline (The skinny—modified from Drexler, 2006)

This is a step-by-step approach to core logging, all steps included in this list are mandatory for accurate data recording.

1. Core should be brought into the core logging area and placed on tables with the labeled end of the box facing outward and organized so that the deepest footage in Box 1 is next to the shallowest footage in Box 2 and so on. Therefore, footage increases toward the rear of the box in each row and footage increases each row to the right (Figure 1).
2. A geotechnician will work in each core logging tent. This technician must perform several tasks before geological logging can begin. The geotechnical tasks include but are not limited to the following:
   - A) Convert the run blocks from feet to meters and write the meterage on the two blank sides of the blocks. All logging measurements are done in metric. This should be the last time you deal with feet. Complete this process before core is washed, its difficult to write on wet run blocks.
   - B) Wash core with a scrub brush and water, drill muds are slimy, more than one washing may be required. Clean core is much better to log because the textures and sulfide mineral grains are more apparent. Washing also makes for better photographs, which are kept on file and the only record of whole core.
   - C) Core is pieced back together as much as possible. Although the core comes out of the ground as a complete cylinder inside the drill tube, it can become mixed-up when it is placed into boxes or during transport to the core tents. As the geotechnicians perform their work, inconsistencies or mixed core are typically resolved. On occasion core is loaded in the box the wrong way, or the helper drops a box or a tube on the ground and the core becomes disorganized. If you cannot piece the core back together, all of the spatial data from this interval is suspect and should be noted in the log.
   - D) Using the core blocks as starting points, determine from/to distance for each box and note it on the top of the box.
   - E) Write the hole number on metal tag (one for each box) and the box number with from/to on second metal tag. Staple the tags onto the box front.
   - F) Measure core recovery and rock quality determination (RQD) for each drill run. RQD is an aggregate of intact core >12cm in length as defined by natural breaks in a given block run. Make sure to account for mechanical breaks (i.e. a piece of core broken to fit into the box or due to the drilling process is counted as one piece). Each geotechnician should enter their data into the database.
   - G) Measure specific gravity samples and enter them into the database.
3. The geologist then records the geological information.
4. Determine sample intervals and assign sample numbers from the sample books. Write the sample number on a metal tag. Staple the metal tag and the paper sample tag from the sample book to the box at the **<u>beginning</u>** of the sample interval. Sample breaks are typically at 2-3 meters except at lithologic contacts and at obvious mineralization changes. Geologists are also responsible for inserting QA/QC samples into the sample stream (see section on QA/QC samples).
5. PIMA data is taken from every intrusive sample interval; including mafic dikes (see section on PIMA for specific PIMA machine usage). A sample is also measured by PIMA when it contains both sediments and intrusive units. In reality we use a later

generation instrument, a Terra Spectrometer, but have continued to use the "PIMA" name for simplicity.

6. A 5-10 cm sample temporarily removed from every rock type (or one for every 20 m if the unit is >20m, see section on specific gravity for details).
7. Generate a sample sheet (see section on Data Entry) for each batch of core delivered to the core cutters. Print three copies, one for the drill log file and two (one each) for core cutting and the prep lab.
8. Complete data entry (see section on Data Entry). **This should be done prior to the core leaving the shack.**
9. Stack core outside on pallets or in plastic bins located at the northeast end of the core shack for pick up by one of the core cutters.
10. At the end of the process the drill hole file should contain the following: Check list, collar info sheet, abandonment form, down hole survey, geological summary, logging sheets (lithology, alteration, mineralization, and structure), sample sheets, specific gravity, and RQD forms.

Figure 1: Typical layout and orientation of core as it is logged.

"Starting a hole" preliminaries (Modified from Drexler, 2006)

1. Create a manila folder, label DC07-xxxx…,DGT07-xxxx…, etc…
2. Fill out the Donlin Creek 2007 Collar Info sheet
    a. Locate location of hole on large map on wall, note drill pad number or proposed ID (e.g. ACINF07-018, ROCH-003, etc)
    b. Note location, which is generally abbreviated in the proposed location prefix (Acma, Rochilieu, etc…)
    c. The actual hole number is associated with the proposed ID number, it is on spreadsheet posting to the left of the wall map.
        i. Note the easting and northing, planned depth, azimuth and dip.
    d. Start and ending dates are drilling dates, *not logging* dates. These, along with rig numbers can be found on the network K drive:

        i. K:\DonlinJV\.....
    e. Note the hole type/core size: HQ, NQ, PQ, etc.
    f. Target can be found on the sections:
        i. Look at section map for section number
        ii. Find section on table.
        iii. Using a ruler indicate where the intrusions are and the expected grade. Put this info in a notebook too.
    g. An Ashtech survey is not always available at the time of logging – keep an eye out for when it does become available.
        i. K:\DonlinJV\2007 AshtechProjects\
        ii. 2007 AshtechSuccessSurveys
        iii. 2007 AshtechSurveys
3. Enter the Collar sheet info into Acquire
    a. File, open recent workspaces, choose your assigned workspace
    b. 1. Data entry
        i. A. New Collar – choose proposed ID appropriate to your hole number and fill in the blank cells (ie hole prefix and number, area, target, dates, etc)

Logging Format

The logging format at Donlin is subdivided into 4 fields: 1) Lithology, 2) Structure, 3) Alteration, and 4) Mineralization. Logging also includes breaking out sample intervals, Terra Spec ("PIMA") analysis, specific gravity measurements, etc. The intricacies and details of these are provided in other sections of this manual.

Although there are no strict rules as to how a person goes about recording information for each of these fields, accuracy is paramount and efficiency a very close second. The order of what needs to be done for each hole depends on the completion of various steps; thus, a systematic, orderly procedure is strongly encouraged. The end product format, however, relies on some conformity so that consistency remains.

But, where does one begin? Again, there are no strict rules. Nevertheless, I suggest the following procedure as a guideline from which each new core logging geologist may find their own tune to dance to. Every geologist has subtle to not-so-subtle variations to the following procedure, depending on how they were taught, years of experience, and the way that they think. The following procedure IS NOT required, but is strongly recommended for the first few holes while you are getting your feet wet and accustomed to the process and desired result. Feel free to modify, change, and improve the procedure as you go to better suit your individual preferences and working styles. As long as you are logging effectively, accurately and efficiently, the Donlin core logging police will not arrest you for violation of logging codes.

RECOMMENDED PROCEDURE (ACTUALLY DOING IT!!!)

Typically, you will have an entire core shack to yourself, which amounts to about 100m of laid out HQ size core. After the core has been washed and geoteched (run blocks converted to meters, and box length, block runs, and RQD measured), the core is ready for the geologist to log. For simplicity, this section does not reveal every intricacy and detail. Details, such as lithologic descriptions, structural angles, codes, modifiers, etc., will be outlined in later sections. The following is in procedural order:

LITHOLOGY

Walk the length of the core a couple of times acquainting yourself with the lithologic contacts and major structures. With a china marker, write directly onto the core a line at each lithologic contact and what type it is. E.g.: Bedding contact (BC), Faulted contact (FC), Igneous contact (IC), Veined contact (VC), etc. Once each lithologic contact has been marked, go back to each and measure their down hole distance from the nearest run block, measure the contact angles, and note the nature of each contact [bedding parallel (BDP), undulating (UN), gradational or sharp (GD or SHP), etc]. I write all of these things directly onto the core at each contact. You may also want to write the rock type abbreviation (code) onto the core on either side of the contact—this may come in handy if the core photos are used later for interpretive work, which you may be called upon to do at some point.

Now, all that needs to be done is to transfer your contact data written on the core to the lithology sheet, and to thoroughly describe each lithologic interval (rock type, color, grain size, texture, etc.). Remember to use modifiers to document features that are consistently distributed throughout the interval.

You may find it useful to start at the top of the hole and describe each lithology singly on paper completely before moving on to subsequent intervals. Doing so will keep your lithologic intervals nicely separated on your page and will make it easier when it comes time for data entry into the database. I tend to leave two or three blank lines between each lithologic interval description so I can easily add to my descriptions if I later see something I missed, such as a general description of sulfide abundance.

STRUCTURE

Well, wouldn't you know it—you already have your rock type contact structures written on the core! Off to a good start, but there is still more structure to do. I typically walk the core a few times marking faults, shears and joints in one pass, then bedding, ash layers, folds, and other textural point features in another pass, and lastly I mark veins. When I mark veins I classify type (V1, V2, V3, V4) and I typically write down the percentage of each sulfide species onto the core as well. Again, don't be afraid to write on the core… down hole distance, alpha angle, thickness, modifiers, sulfide species and concentrations, etc.

It's on the core—put it on paper, use the remarks column to describe, gouge percentages, contact characteristics, etc. Quick and easy!

DEFINE SAMPLE INTERVALS

Okay, in the strictest sense, defining sample intervals isn't quite logging core, but it has to be done before you can log sulfide concentrations, and pull samples for PIMA spectra. You might be thinking… "Shouldn't we log sulfide concentrations before we define our sample intervals? Aren't sample intervals defined, in part, by sulfide species and abundances and not simply lithology?" Well, by golly you would be right! So, let's do that.

Well, what do you know, you've already logged mineralization for your veins when you did structure! And… if you are sneaky you might have already written py, as, or, re, etc., on the core when you came across it logging both lithology and structure. If so, simply do one quick pass and note any mineralization intensity breaks where you might want to start/stop a sample interval. For disseminated mineralization it is not usually necessary to be too picky about breaking samples out based on mineralization intensity, unless changes are dramatic. However, veins should be given more consideration (see below).

Now, mark on the core sample interval breaks abiding by the following guidelines:

SULFIDE MINERALIZED rock, whether it is rhyodacite, mafic dike, or sedimentary, has a MAXIMUM sample length of 2 meters, **no exceptions**. It is okay to have a sample interval less than 2 meters, in some cases it is even desirable, a 0.5 m sample length is enough material for the sample lab to process. It is much better to have a mineralized sample smaller than 2 meters than larger. Small samples can always be mathematically composited, but larger samples cannot be reduced for more detail. A MAXIMUM of 3 m is allowable for sample intervals only if it is in NON-MINERALIZED SEDIMENTS.

A 10 to 15 meter transition zone of two-meter sized samples is used both at the beginning and end of non-mineralized runs. This is equivalent to 5-8 two meter samples.

Interval breaks: Sample intervals should not cross lithology contacts that are documented on the log. There may be times when there a lithology is not documented as a lithologic interval; for example ash beds, small intervals of intrusive (<1 m), or small intervals of sediments within a large intrusive body. For these instances, you can cross the lithologic contact (because it is not documented as its own lithologic interval), just make sure it falls within a single sample interval—do not divide it between two samples. You may have to shorten prior sample intervals to do this, again 2 meters is a maximum, you can go smaller, but not larger. You may break out 1 m intrusive as a sample interval, but avoid taking samples smaller than 0.5 m.

Similarly, avoid splitting sulfide mineralized veins between two samples. Adjust your prior sample widths smaller so that you can fit the entire vein into a single sample. If you find a vein greater than 2m, say 2.23m, then make 2 samples out of it (1.0m and 1.23m, for example). Veins of this width are rare, but they have been observed as in the case of hydrothermal vein breccias. This is especially important for very well-mineralized veins.

At this point there should be abundant writing on the core from the previous logging. So, when marking sample intervals you may want to use a different color marker, or use an asterisk to make the sample interval marking stand out. Once the sample intervals are marked it is time to staple on the aluminum and paper sample tags—Remember to include QAQC samples (see section on QAQC), writing them in the Sample book along with "from" and "to."

(If you start logging first thing in the morning, you should usually be at or near this point by noon.)

LUNCH BREAK

MINERALIZATION

As stated earlier, if this is your adopted method, the actual logging of sulfide minerals should already be completed on your core. At this stage it is simply a matter of transferring that data onto your mineralization sheet. Sulfide species and abundances are to be logged on a by-sample-interval basis to insure maximum detail at the resolution of sample intervals. Time spent filling out the mineralization sheet is typically <20min (sometimes much less). Remember to get the carbonate (V4) veins too—they are nearly always barren of sulfide, but are still documented.

Sulfide concentrations are on a **1-4** scale: **1**--trace to 0.5%, **2**--0.5% to 2%, **3**--2% to 5%, **4**-->5%.

ALTERATION

Alteration logging is a three part process. Part one is clay alteration and involves use of the PIMA. Parts two and three are simply a qualitative designation of carbonate and Fe-oxidation in the rock. Carbonate alteration at Donlin is somewhat of a misnomer as much of the carbonate present is likely sedimentary in origin or due to meteoric water surficial transport and deposition of calcite. Nevertheless, it is very important that we document calcite effervescence for purposes of acid reducing potential and stratigraphic studies, more detail will be provided on this later. Fe-oxide alteration is due to surficial weathering processes of mafic minerals and sulfides when meteoric waters are introduced through faults and various other permeable aspects of the rocks.

PIMA PREPARATION

Before PIMA spectra measurement can be taken, samples must be collected from each intrusive bearing sample interval. Write the last three digits of the corresponding sample number onto each respective sample. You do not need to run PIMA analyses for sedimentary sample intervals. As PIMA samples must be dry before analysis, efficiency can be increased by collecting the samples and setting them aside to dry while logging Fe-ox and carbonate alteration.

FE-OX and CARBONATE ALTERATION LOGGING

Although it is not necessary to log Fe-ox and carbonate on a by-sample-interval basis, I find it easier and more efficient, so I present it this way. Others may disagree and that is fine. I find that logging on a by-sample interval basis keeps me going in a systematic, consistent, and accurate fashion after months of logging (you begin to get a little "punchy" after awhile)—this way I know my alteration logging detail/resolution is always adequate for any acid reducing potential and stratigraphic studies. And I also know that my alteration will be detailed enough to be compared with mineralization and assay data for any correlative comparisons. All rock types should be checked for acid effervescence, not just the sediments. I suggest dripping a two inch line of acid within each core divider. Where there are abundant interbeds or variations in the intrusive clay alteration, you will want to spot check different areas. After dripping the acid, qualitatively average the effervescence for that sample interval and write it down on your log. Time spent by this method is typically <30 minutes.

Both Fe-ox and carbonate are logged on a **1-4** scale: **1**-weak, **2**-moderate, **3**-strong **4**-extreme.

CLAY ALTERATION LOGGING

For most people, the subtleties of clay alteration make it extremely difficult to log visually with any amount of consistent accuracy. For this reason, the PIMA replaces qualitative visual estimates from geologists with a spectral analysis of the clays.

At this stage your samples for PIMA which have been set aside should be dry. If they are not, you may want to consider doing the specific gravity analysis at this time.

No data other than "From", "To", and "Sample Number" need to be written out for the PIMA. A check mark is to be placed on the PIMA sheet for each corresponding sample/interval as it is analyzed and designated as "light" or "dark" colored. You may want to first enter your sample intervals into acquire. This way you can copy and modify your sample sheet for the prep lab as a PIMA sheet. This can save you a lot of time as you will not have to individually write out each sample interval and sample number by hand for the PIMA sheet.

A procedure for using the PIMA analyzer is provided in the appendices. Disregarding the drying process, actual work time for PIMA (which includes sample collecting and filling out the PIMA sheet) can be 2 hours or longer. Although the process is simple enough, it is very time consuming when there are long runs of intrusive.

When you are finished, remember to put the samples back into their corresponding sample intervals.

SPECIFIC GRAVITY MEASUREMENTS

Specific gravity analysis is important for tonnage and grade estimates and related mining process issues. While it is not necessary to measure every sample, the data should be collected so that they are representative of the mineralized zones we encounter in the drill holes. Thus, the procedure in place is to collect a minimum of one sample every 20 meters, and at least one sample per lithologic interval if it less than 20 meters in length. Each sample should be 5 to 10 cm in length (a little longer is okay as precision is improved for larger samples), although samples do not have to be uniformly shaped as this method does not require a volume measurement. (Not sure of the name of the method, but it seems like something Archimedes would have done.) The procedure is simple enough and is as follows:

Write on each sample a point measurement representing its distance down hole. Measure and document the length of each sample on your sheet. Also document each sample's down hole distance. First, weigh each sample dry, and then weigh each sample wet using the tin can weighing pan. When weighing wet, make sure the tin can and sample are completely submerged, and that the apparatus is hanging freely. Both dry and wet weights are to be documented. Place samples back into their corresponding vacancies in the core boxes. That's it, pretty simple. Total time involved is usually <30 minutes, depending on the number of lithology breaks.

SPECIFICS, INTRICACIES AND DETAILS

Total Mineralization

Total mineralization is based on the combination of vein sulfide intensities and disseminated mineralization intensities. Disseminated mineralization is dominated by pyrite and arsenopyrite. Other disseminated minerals (e.g. realgar, orpiment) occur locally and are not typically pervasive. Pyrite is common in altered intrusive, in sediments adjacent to intrusive contacts, associated with fault structures (hydrothermal) and as bands in sedimentary rocks (syngenetic). Disseminated arsenopyrite can occur in halos adjacent to veins and fractures, in coalesced halos and in discrete zones amongst intrusive.

It can also be difficult to estimate percentages of disseminated sulfides, particularly arsenopyrite, which can be highly variable throughout an individual sample interval but can also have a major influence on grade. A more qualitative approach, as suggested on the "cheat sheet" is recommended.

Structures

Structure data is captured as point data for entry into the database. Interval data (i.e., a broad fault zone, ~3m) can be entered as a remark or as a modifier. However, be sure to include some representative point measurements from within these broad zones.

Structure data can be entered as both non-oriented and oriented data (see below). The amount of data collected is up to the logging geologist, but must be sufficient enough for computer modeling and interpretive purposes. Contacts should be noted and described (i.e. fault, bedding parallel, etc) as well as fault zones select veins and local bedding. Depth, structure type and modifiers, alpha (angle to core axis) and width (where applicable) should be recorded for each chosen structural feature. The amount of structure to record is determined on a per hole basis, when the there are no major structures the small structures are representative, and are recorded.

The 2007 season is focusing on contact orientation and behavior as well as bedding orientation—right side up or up side down and changes in bedding angles. This information is critical for pit stability design as well as stratigraphic correlation.

Alpha angles do not require the core to be spatially oriented, and thus can be measured on both oriented and non-oriented core. The alpha angle is simply the angle relative to the core axis. Hold the protractor parallel to core axis with the arrow pointing at the structure and read the strip protractor.

Oriented core measurements rely on beta angles. Oriented core will have a line running parallel to core axis throughout most of the core; highly broken areas are difficult to connect the line through. Before logging make sure the line matches on either side of the core blocks. At each new run a new line is started, which is suppose to be checked with the previous line (run), sometimes this is not so. Any line that is misplaced cannot be used for oriented data.

A beta angle is the angle between the line drawn on the core by a geotechnician (the line originates as a scribe when the core comes out of the tube) and the farthest **down-hole** point of a structural feature about the center of the core. Structural features include bedding planes, fractures, veins, faults, contacts, etc. Beta angles are measured using a tool called a linear protractor, which is a strip of paper with a scale from 0 to 360 (representing the degrees of a circle). There are different linear protractors for each size of core HQ, HQTT, NQ, NQ3, etc.). Before taking a measurement with a linear protractor, make sure you have the correct size. Test the linear protractor by wrapping it around the core and seeing if 0 and 360 line up.

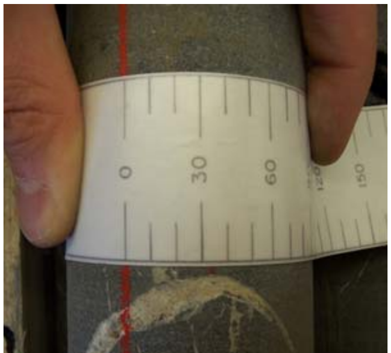


Figure 2: Kyle measuring the beta angle of a V4 vein. This beta angle is 33 degrees.

To take a beta angle measurement, first line up the zero of the linear protractor with the orientation line. Wrap the linear protractor around the core in a **counterclockwise** direction (make sure the 0 is on the left side of the liner protractor before wrapping it around the core) until the furthest **down-hole** point of the core is reached by the linear protractor. Read off the number; this is your beta angle. Try to measure your beta angles to within a degree.

Beta angles are extremely helpful for modeling purposes. However, a wrong beta angle is worse than none. Therefore, if you are unsure of the accuracy of the orientation line or your beta measurement, don't try to guess to take the measurement.

Sampling and sample series

Some of the intricacies of sampling have been described under the recommended procedure section. More detail is provided here. Each core logger and their cross shift will be assigned a series of a 1000 sample tags. When one geologist leaves camp to go on break the resuming geologist for that hole will continue with the sample book. This is so that the groups of 78 will remain as drill hole consistent as possible—in no instance should a sample book be switched to another hole before the prior hole is completely sampled. When you come to the end of the series, simply start a new sample series, but clearly indicate on the sample sheet (excel sheet) that you are switching sample series.

QA/QC Samples

For year 2007, the analytical lab assay fusion batch size is comprised of 78 samples. Thus, each group of 78 is to have <u>up to</u> 12 (twelve) QAQC samples inserted into the sample sequence which includes 3 of each of the following: a standard STD, a granite blank (Blank-Granite), a Crushed Duplicate, and a Field Duplicate* (special treatment, see below). The field duplicate is "new" for 2007 and is simply the other half of the sawed core.

For simplicity you may want to think of the group of 78 as 3 groups of 26, but as far as the prep lab is concerned each batch is a group of 78. This of course makes it difficult to break up the sample numbering sequence into readily predictable sample number groupings. Thus, after exporting your sample sheet from acquire, add a new column, "Group No". We MUST add this column or the prep lab will have no idea how to split up and process sample batches. An example is appended. Because we are not beginning/ending on easily predictable sample numbers, I recommend the following procedure to help keep things straight:

On the front of each sample book write # 78 starts at sample number "xxxxxx" and ends at sample number "xxxxxx". Inside the sample book write "#78 begin" on the page with the beginning sample number, and “#78 end” on the last (ending) sample number. I know that this means there will be two consecutive tag pages with #78 written on them, but it will make it clear which is the beginning and which is the end. At this point, my method is to write the QAQC in the sample book(s), dividing the 78’s into three 26’s (with my finger as a place holder) and put a std, dup, blnk-gran randomly spaced into each “26”. **For Field Dupes, make sure that they are for sulfide mineralized rock only and no more than 1 per 26 samples. Field duplicates are taken at the discretion of the geologist. In non-sulfide mineralized rock they are not taken.*

> If one enters their QAQC into their sample books as a means to determine where QAQC samples should be inserted there is no need to create a “handwritten” sample/PIMA sheet. As long as one enters their sample intervals into acquire prior to when the PIMA measurements are taken, a PIMA sheet may be converted from a copy of the core cutting/prep-lab sample sheet.

Additionally, core loggers will now be assigning submittal numbers for each group of 78. As each submittal is comprised of only one group of 78, we have no need to use the old designation of batch number as it would simply be the same as is used for the submittal. In order to clarify the submittal beginnings and endings for the prep lab, insert a blank row above the beginning of each new submittal, a spreadsheet tracking these numbers is on the K-drive and a posting is on the geology office cork board.

It is essential that we keep a good and complete record within our sample books for hole begins/ends, #78 begins/ends, QAQC inserts, submittal numbers and of course sample intervals. To keep from duplicating submittal numbers, a tracking sheet will be kept in the geology office to be filled out for each submittal/“batch”. Each submittal number must be a COMPLETE 78.

Completed/used sample books are to be filed in the geology office--DO NOT THROW THEM IN THE TRASH.

Rock Type Descriptions

Sedimentary Units

Greywacke (GWK)—The greywackes at Donlin are typically medium (1-2mm) to fine (<1mm) sand and silt sized lithified sediments. The color is typically light to medium-dark gray. When scratched with a nail or scribe it is very gritty compared to the other dominant Donlin sedimentary rocks, siltstone and shale. Calcareous cement is common but typically not visually apparent, although readily apparent by testing with acid for effervescence. Whether the calcareous cement is a product of sedimentary processes or secondary precipitation out of ground water is at times a matter of some debate. The fossil (shell fragment) calcareous intervals are almost certainly limestone layers.

Limestone at Donlin generally occurs as thin layers <0.5m interbedded within the siltstone and shale units. Occasionally, fossiliferous limestone layers (with shell fragments) occur as short intervals within the coarser grained greywacke. The limestones are almost entirely CaCO3, which differs from areas that have an overprint of carbonate alteration or secondary carbonate by meteoric water transport and precipitation. Due to the thin nature of the limestone layers is has not been broken out as a logging unit, and thus, has no database classification. When encountered, however, it should be mentioned in your remarks.

Claystone is comprised primarily of clay-sized particles (less than 0.004 mm in diameter). It does not refer to those rocks that are laminated or easily split into thin layers (clay shales). At Donlin, claystone is most easily identified by its peculiar waxy property when wet. When wet, the claystone is difficult to write on with a china marker, but when dry, this diagnostic property vanishes. Claystone also breaks much the same as siltstone, in a blocky, semi-concoidal fashion. Although claystone is markedly different than siltstone, in 2007 we are not differentiating it from Siltstone (SLT). Nevertheless, when encountered in drill core, note it in your remarks.

Siltstone (SLT) is comprised of both clay and silt size particles. In many respects it looks similar to the Shale at Donlin (as it is comprised of the same material) where it is very fine grained and dark gray to black in color, however it lacks the well-developed fissility (platy cleavage). Like claystone, it typically has a semi-concoidal to concoidal fracture and when broken has a "blocky" appearance. When the rock contains an appreciable amount of clays and it is wet, the rock takes on the characteristic "waxy" property of claystone. Technically, when the rock contains >50% clays it should be classified as a claystone.

Shale (SHL) is comprised of the same sedimentary material as the Siltstone (and claystone) listed above, except that it has well developed fissility and easily breaks into platy laminated pieces. For engineering purposes it is very important to distinguish shale from the other Donlin sedimentary units. Our designation of shale also includes the rare occurrences of higher grade metamorphic rocks slate and phyllite.

Conglomerate (CGL) is a sedimentary rock comprised of rounded lithic and/or other rock fragments/clasts >2mm within a finer grained supporting matrix. Conglomerate beds are believed to be sporadic occurrences at Donlin and are typically interbedded within the greywacke unit, and may contain rounded clasts of the Donlin siltstone and/or shale within a matrix comparative to the Donlin coarse grained greywacke through the finer grained siltstone.

Argillite (ARG) is a fine-grained sedimentary rock comprised predominantly of indurated clay particles, but may contain variable amounts of silt-sized particles. The argillites grade into shale when the fissile layering typical of shale is developed. At Donlin the argillite is typically very hard and comprised mainly of clays.

Ash (ASH)—Fine grained volcanic ash. The ash beds at Donlin are light gray in color, very fine grained, and typically non-lithified. The ash occurrences make excellent marker horizons in the stratigraphic column. They are usually relatively thin, 1 to 5 cm, but may be up to >0.5m in some areas. Due to the lack of lithification the ash beds pose significant stability concerns for pitwall design and are to be logged as discrete points with a defined thickness and angle under the Structure field.

Igneous Units (apparent oldest to apparent youngest)
(Modified after Piekenbrock, 2002 and 2003)

MD is the apparent oldest of the intrusions and occur as a series mafic dikes and sills. These dikes are thin generally 1 to 3 meters in width and are typically a buff brown-green color and fined grained. They are often carbonate rich (effervesce easily) and contain accessory greenish mariposite/fuchsite, and altered biotite. Margins are occasionally flow banded. A darker brown (biotite-rich) phase has been encountered on occasion and is believed to represent less alteration.

RDF follows emplacement of the mafic dikes and is a very distinct fine-grained porphyry dike called RDF. Striking roughly ENE to E-W the RDF dike dips shallowly 10 to 25 degrees to the north. At ACMA, the RDF is typically a fine-grained felsic unit with distinctive small feldspar phenocrysts set in a grey fine-grained matrix. The unit is typically 5 to 10 meters in width and appears to fill a pre-existing structural orientation often filled by the early mafic dikes. This structural corridor is a critical element in the ACMA zone and acts as a continual focus of intrusion and post-mineral faulting.

> Diagnostic features: Fine grained matrix (not aphanitic), with small feldspar phenocrysts typically <0.5cm. Commonly medium to dark gray with abundant sulfide fracture fillings (V1) <1mm. Phenocrysts are much less abundant than the more crowded "X" phases.

RDX has been used as the catch-all term for any coarser-grained or phenocryst-rich (crowded) porphyry. Volumetrically the most common intrusive phase present, it is characterized by sharp intrusive contacts without chill margins. It is relatively homogenous with a distinctive crowded feldspar texture but shows widely varying alteration from texturally enhanced illite-rich alteration of the crowded phenocrysts to broad texturally destructive alteration dominated by grey/green smectite and often strong ankerite/carbonate alteration. The ankerite alteration often forms distinctive haloes rimming feldspars. In some instances, the older core has already developed significant orange-colored oxidation of the ankeritic component (occasionally confused with orpiment) making it in some ways easier to log than fresh core. Believed to be closely related to the RDXB, but without pervasive disseminated graphite in the matrix, however can contain graphite/feldspar blebs and knots up to several centimeters in diameter. Large dikes of RDX encountered under Lewis (to date) have revealed unimpressive mineralization.

> Diagnostic features: Crowded feldspar phenocrysts with medium grained (phaneritic) matrix.

RDXB is a coarsely porphyritic unit with very large blocky feldspars set in a graphite/sulfide-rich matrix which gives the unit a distinctively dark appearance. This is the so-called 'blue porphyry' of the early Placer years. In Acma it is a very important mineralizer and contains impressive high-grade as disseminated mineralization. RDXB occurs as a series of discrete sill like bodies often along strike from RDX sills. Indeed, the spatial distribution of the 'XB' suggests they could be highly fluidized, coarsely quenched RDX dikes. (Previously logged as RDX and RDBP)

This rock type was originally divided out from RDX and named RDBP (Rhyodacite Blue Porphyry) due to the deep blue color resulting from the disseminated graphite within the matrix. Later, it was renamed RDXB to more accurately describe the crowded feldspar phenocryst texture. As it commonly has large blocky feldspars, many geologists have assumed the "B" to mean "blocky", which is technically incorrect, as RDXB must contain disseminated graphite within the matrix.

> Diagnostic features: MUST contain disseminated graphite throughout the matrix. Commonly contains large blocky feldspars and disseminated sulfides, but these characteristics are not definitive.

RDXL is also roughly emplaced within the ENE structural zone generally lying immediately below the RDF unit as a dike-like, moderately NNW-dipping unit. It thickens rapidly to the NE along section where it is poorly defined by drilling. The unit is defined by distinctive large plagioclase laths in a population of smaller K-spar phenocrysts. It has significant coarser-grained biotite and seems to be distinctly less altered or more texturally enhanced by alteration than the other units. It rarely shows strong textural destruction. It has generally been lumped into RDX though compositionally and texturally it is much more similar to a coarse-grained RDA. (Once logged as RDX)

> Diagnostic features: Abundant lath-shaped feldspar phenocrysts. Some areas exhibit a light blue color similar to that of RDXB, but without the presence of disseminated graphite.

RDA is a very distinctive unit with a fine-grained salt and pepper texture of fine biotite phenocrysts and variable quartz and K-spar phenocrysts set in an aphanitic matrix. It is often easily distinguished by a flow-banded (possible chill) margin with other units. The finer-grained less phenocryst-rich units tend to have the more obvious flow-banding (chill?) than the coarser-grained units. Texturally it is highly variable compared to RDX which is largely homogenous. Conversely, it seems to display less alteration variability than RDX. Two important sills of RDA have been recognized in the model and serve as excellent markers traceable across the entire area currently reviewed at ACMA. (Previously logged as RDA or RDX depending on the abundance of phenocrysts). On occasion, this unit is confused with highly altered mafic dike, but MD contains a much, much lower concentration of quartz phenocrysts (if any at all).

> Diagnostic features: Aphanitic matrix, non mafic. Commonly contains large >0.5cm quartz phenocrysts, flow banded margins, and is comparatively unaltered. Usually has a distinctive creamy color. In photographs it is difficult to distinguish from the mafic dikes.

Vein Type Descriptions and Classifications

Type 1 (V1) - Thin discontinuous sulfide (as pyrite with trace arsenopyrite) veinlets often with little to no quartz and a broad disseminated selvage of pyrite. They often show minor amounts of ankerite in the veinlets as well as ankerite intermingled with adjacent disseminated pyrite, pervasive in the adjacent matrix and as distinct overgrowths or rims around adjacent plagioclase phenocrysts. Such veins are typically low grade and show broad pervasive illite alteration.

Type 2 (V2) - Thin discontinuous quartz sulfide veinlets often with variable pyrite and arsenopyrite contents and broad often pervasive selvages of fine-grained needle-like arsenopyrite. Open space vuggy textures become common and trace amounts of stibnite occur in some veinlets. There is some suggestion that a broad pyrite front surrounds the arsenopyrite selvage itself. Such veins show moderate grade and intensifying illite with variable but overall decreased ankerite.

Type 3 (V3) Can be subdivided into two types, but in 2007 we are making no distinctions between types "a" and "b" for our logging of V3 veins.

Type 3a - Thicker, more continuous open space quartz veinlets often with pyrite, arsenopyrite, and native arsenic as irregular late spheroids and variable amount of stibnite. Such veins often show broad arsenopyrite-rich selvages with little to no ankerite. In some instances, minor amounts of calcite are present in the veins and in many instances though not always small amounts of kaolinite fill the open space interstices within the veinlets. Such veinlets often exhibit elevated grades.

Type 3b - Thick, continuous quartz veinlets with open space textures and complex sulfide mineralogy including pyrite, arsenopyrite, stibnite, native arsenic, realgar, orpiment and trace sphalerite in intensely illite altered material. The veinlets often have thin fillings or late fracture partings of kaolinite. Realgar sometimes occurs within adjacent feldspar sites giving a 'punkin patch texture' to the surrounding rock. Grades are often very high.

Type 4 (V4) - Discrete carbonate veinlets as both ankerite and calcite are not yet adequately evaluated. Ankerite veinlets are probably related to the earlier sulfide rich veins while calcite/dolomite veinlets seem to be preferentially distributed as a halo in the surrounding sediments. Calcite also occurs as trace amounts in Type 3 veins.

Here's an excellent diagnostic flow-chart method for vein classification modified after Scott Petzel in 2002:

1. Is the structure mineralized? i.e. contains minerals associated with hydrothermal activity?
    - If Yes, then continue to 2…
    - If No, then select appropriate non-mineralized structure code.

2. Does the structure contain realgar, orpiment or stibnite?
    - If Yes, then the vein is Type 3 - select appropriate codes and modifiers
    - In No, then continue to 3 …

3. Does the structure contain >30% quartz?
    - If Yes, then the vein is Type 2 - select appropriate codes and modifiers
    - In No, then continue to 4 …

4. Does the structure contain >50% carbonate?
    - If Yes, then the vein is Type 4 carbonate vein - select appropriate codes and modifiers.
    - In No, then the vein is Type 1 sulfide vein - select appropriate codes and modifiers.

Logging Codes and Modifiers

Each logging field contains a primary/principle classification and modifiers. Lithologic modifiers are generally consistently distributed throughout the interval, whereas structural modifiers are generally more constrained (local) and considered as "point" data. Here is a cheat sheet for the 2007 logging codes:

CHEAT SHEET—Donlin Creek 2007 Geology Logging Codes

Cheat Sheet Version 6.2 adopted Updated March 20, 2007

Rock Type Primary Fields

Sedimentary Rocks	**Intrusive Rocks**	**Metamorphic Rocks**
CGL: Conglomerate	GDR: Granodiorite	BHF: Biotite Hornfels
GWK: Greywacke	IBX: Intrusive Breccia	CHF: Calcsilicate Hornfels
	MD: Mafic Dike	HFL: Hornfels
	MZD: Monzodiorite	SKN: Skarn
SHL: Shale	QLT: Quartz Latite Porphyry	
SLT: Siltstone	RD: Rhyodacite	
ARG: Argillite	RDA: Aphanitic matrix Rhyodacite	
GRN: Ground / Missing Core	RDF: Fine Grained Rhyodacite	
OVB: Overburden	RDX: Rhyodacite (crowded)	
OVR: Channel Deposits	RDXB: Rhyodacite (crowded w/disseminated graphite-Blue & Blocky)	
NS: No Sample	RDXL: Rhyodacite (crowded with lath-shaped phenocrysts)	
	RHY: Rhyolite	

Rock Type Secondary Fields

In mixed intervals primary rock type codes can be used as modifiers.

Sedimentary Features	**Intrusive Features**	**Metamorphic features**	**Structural**	**Alteration**
BD: Bedded	AH: Aphanitic	BN: Banded	BK: Broken	BL: Bleached
BN: Banded	IBX: Intrusive Breccia	FO: Foliated	BX: Brecciated	OX: Oxidized
CHT: Chert	EG: Equigranular	MY: Mylonitic	CA: Cataclastite	
LM: Laminated	XE: Xenolithic	RC: Recrystallized	LE: Lineated	
MX: Massive	MX: Massive	SC: Schistose	SH: Sheared	
	PH: Phenocrystic	SP: Spotty	SL: Slickensided	
Grain Size	PP: Porphyritic		VU: Vuggy	
CG: Coarse Grained			PL: Polished	
FG: Fine Grained				
MG: Medium Grained				

Alteration

Clay:	**Fe Oxides:**	**Carbonate:**	**Other:**
CL: Clay unidentified	HE: Hematite	AN: Ankerite	SI: Silica
KA: Kaolinite	LI: Limonite	CB: Carbonate	BI: Biotite
IL: Illite	OX: Oxidized	DO: Dolomite	KF: K- Feldspar
IK: Mixed Illite/Kaolinite	MG: Magnetite		FU: Fuchsite
SE: Sericite			MA: Mariposite

Pima Alteration Logging

Pervasive:		**Veinlet Alteration:**	
IL: Illite	BE: Smectite	VI : Veinlet Illite	VC: Veinlet Carbonate
KA: Kaolinite	B: Dickite	VK: Veinlet Kaolinite	
IK: Illite/Kaolinite mix	AL: Albite	VD: Veinlet Dickite	

Structure Codes

Vein and Vein Modifiers can be recorded as point structure data.

Contacts	**Sedimentary Features**	**Intrusive Features**	**Structural**	**Mineral**
Level One > CT: Contact	BC: Bedding Contact	IC: Intrusive Contact	FC: Faulted Contact	VC: Veined Contact
Level Two >	GC: Gradational Contact	CM: Chilled Margin	BX: Brecciated	
	U: Unconformity	BN: Banded	SL: Slickensided	
			SH: Sheared	
			BDP: Bedding Parallel	
Modifiers >	SHP: Sharp	UN: Undulating		
	IR: Irregular			
	BK: Broken			

Faults Level One >	FLT: Fault (brittle)	SH: Shear (ductile)	FR: Fracture (open)	JN: Joint (parting)
Level Two >	GO: Gouge	MY: Mylonitic	MF: Mineral filled	
	BX: Breccia	SHZ: Shear Zone	CF: Clay Filled	
	FLZ: Fault zone			
	CA: Cataclastic			
Modifiers >	SL: Slickensides	BK: Broken		

Miscellaneous Structures	FL: Fold	**Miscellaneous Modifiers**	BDP: Bedding parallel	FP: Fault parallel
ASH: Ash layer	KL: Cleavage	UN: Undulating	CVP:Cleavage parallel	CTP: Contact parallel
BD: Bedding	LN: Lineation	PL: Polished	FOP: Foliation parallel	SH: Shear surface
BN: Banding	FO: Foliated	PU: Pulverized	GB: Graded Bedding	
BDC: Cross Bedding				

Mineralization

Primary Vein Codes	V1: Type One Veins = *	V2: Type Two Veins =	V3: Type 3 Veins =	V4: Type 4 veins =
*** If a structure contains sulfides on less than 50% of its surface and there is no quartz or carbonate it is coded as a fracture (FR).**	Quartz and/or carbonate veins with >50% sulfide (but without ST, OR, RE or NA)	Veins of greater than 30% quartz, < 50% sulfide, (but without ST, OR, RE or NA)	Any mineralized vein structure containing ST, OR, RE, NA.	Veins containing >50% carbonate material.

Modifiers (ABC's)

Primary Mineralogy	**Physical feature mineralogy**	**Secondary Mineralogy**	**Location related**		
AN: Ankerite		HE: Hematite	AV: Anastomosing	AB: Albite	BDI
AS: Arsenopyrite	LI: Limonite	BN: Banded	BI: Biotite	CVP: Cleavage parallel	
AU: Gold	MA: Malachite	BX: Brecciated	CH: Chlorite	FOP: Foliation parallel	
CA: Calcite	MG: Magnetite	BK: Broken	CL:Clay (unidentified)	FP: Fault parallel	
CB: Carbonate	NA: Native Arsenic	FD: Faulted	EP: Epidote	CTP: Contact parallel	
CC: Chalcocite	OR: Orpiment	LM: Laminated	FU: Fuchsite		
CA: Calcite	PO: Pyrrhotite	MY: Mylonitic	GA: Garnet		
CI: Cinnabar	PY: Pyrite	OX: Oxidized	KF: K-Feldspar		
CO: Covellite	RE: Realgar	RC: Recrystallized	MU: Muscovite		
CY: Chalcopyrite	SB: Stibconite	VU: Vuggy	PG: Plagioclaise		
DO: Dolomite	SC: Scorodite		SE: Sericite		
GL: Galena	SP: Sphalerite		TO: Tourmaline		
GR: Graphite	ST: Stibnite		TA: Talc		

Intensity Codes

Vein Material Abundance Codes	**Sulfide/Ore mineral Intensity Codes**	**Alteration Intensity Codes**
1: Weak (approx 1-2, 1cm vns/m)	1: Weak (Approx trace to 0.5%)	1: Weak
2: Moderate (approx. 3-5, 1cm vns/m)	2: Moderate (approx. 0.5% to 2%)	2: Moderate
3: Strong (approx. 6-10, 1cm vns/m)	3: Strong (approx. 2% to 5%)	3: Strong
4: Very Strong (>10, 1cm vns/m)	4: Very Strong (>5%)	4: Extreme

Memo:

April 12, 2007

From: Brian Flanigan

To: Core logging geologists (VIPs of Donlin)

Subject: Submittal number tracking and usage (and other stuff).

In order to keep things somewhat coherent for the prep lab, assay lab, and database, we will using and tracking submittal numbers in the following fashion:

1) Keep submittal number and hole number continuity as much as possible.

2) We will not have sample series dedicated to individual people anymore. The sample series will be dedicated to logging tents and transferred from an outgoing geologist to the incoming geologist. So, keep your sample books marked accurately as to hole number and groups of 78.

3) Submittal numbers will be transferred in the same manner as sample series. Two geologists should not be using the same submittal number simultaneously. There is a tracking sheet on:

   K:\ _donlinjv\2007\2007 Sample Submittal Sheets

   named 2007_Submittal_numbers.xls.

   Please keep it up to date and put your name in the Misc Notes column.

4) Core is to be down stacked onto pallets/bins in the in the usual fashion. Large numbers on the bottom and small numbers on the top. Do not mix drill holes on a single pallet/bin. It is okay to have two submittal numbers on a pallet/bin—this will happen frequently, but should not pose a problem for the pre lab.

5) Be courteous to your incoming cross-shift. Leave them in good shape to continue the hole you started. Have all of your data entry completed for what you have logged—clearly note on each log sheet what has been entered into acquire. Tidy up your logging area before you leave. The geologist name in the database should go to the person who **enters** the most data for that hole.

Also… A couple of changes... I have spent a lot of today coordinating with core cutting, the prep lab and Glen Marshall (the consultant who set-up the new saw and prep labs).

Anyway, in order to streamline things with core cutting and the prep lab we will be color-coding the QAQC on our sample sheets.

Color coding will be standardized as follows:

Start group of 78 submittal no DC07-nnn = Lavender

All standards = Rose

Field duplicates = Light orange

Crush duplicates = Sky blue

Blanks = Light Yellow

These are exact colors and show up under the fill pull-down menu.

I know this seems petty (if not downright silly), but it will really help core cutting and the prep lab keep things organized.

See attached example.

I have also attached and email concerning sample series and submittal number and other things including shift change-outs. Just so we are all on the same page.

Brian

Hole ID	Group #	Sample #	From	To	Std/Blank	DUP	Weight	Photo	Saw	Bucket
Continue Submittal # DC07-362										
DC07-1615	18	798612	220.43	222						
DC07-1615	19	798613	222	224						
DC07-1615	20	798614	224	226						
DC07-1615	21	798615	226	228						
DC07-1615	22	798616	228	230						
DC07-1615	23	798617	230	232						
DC07-1615	24	798618	232	234						
DC07-1615	25	798619	234	236						
DC07-1615	26	798620	236	238						
DC07-1615	27	798621	238	240						
DC07-1615	28	798622	240	242						
DC07-1615	29	798623	242	242.75						
DC07-1615	30	798624	242.75	244						
DC07-1615	31	798625	244	246						
DC07-1615	32	798626	246	248						
DC07-1615	33	798627	248	250						
DC07-1615	34	798628	250	252						
DC07-1615	35	798629	252	254						
DC07-1615	36	798630	254	256						
DC07-1615	37	798631	256	257.5						
DC07-1615	38	798632	257.5	258.5						
DC07-1615	39	798633	258.5	259.8						
DC07-1615	40	798634	259.8	261						
DC07-1615	41	798635	261	263						
DC07-1615	42	798636	263	265						
DC07-1615	43	798637	265	267						
DC07-1615	44	798638			Blank-Granite					
DC07-1615	45	798639	267	269						
DC07-1615	46	798640	269	271						
DC07-1615	47	798641	271	273						
DC07-1615	48	798642			Std-IM					
DC07-1615	49	798643	273	275						
DC07-1615	50	798644	275	277						
DC07-1615	51	798645	275	277		DUP				
DC07-1615	52	798646	277	279						
DC07-1615	53	798647	279	281						
DC07-1615	54	798648	281	283						
DC07-1615	55	798649	283	284.68						
DC07-1615	56	798650	284.68	286						
DC07-1615	57	798651	286	288						
DC07-1615	58	798652	288	290						
DC07-1615	59	798653	290	292						
DC07-1615	60	798654	292	294						
DC07-1615	61	798655	294	296						
DC07-1615	62	798656	296	298						
DC07-1615	63	798657	296	298		DUP				
DC07-1615	64	798658	298	300						
DC07-1615	65	798659	300	301.74						
DC07-1615	66	798660	301.74	302.92						
DC07-1615	67	798661	302.92	304.5						
DC07-1615	68	798662	304.5	306.5						
DC07-1615	69	798663			Std-IL					
DC07-1615	70	798664	306.5	308.5						
DC07-1615	71	798665	308.5	310.5						
DC07-1615	72	798666	310.5	312.5						
DC07-1615	73	798667			Blank-Granite					
DC07-1615	74	798668	312.5	314.5						
DC07-1615	75	798669	314.5	316.5						
DC07-1615	76	798670	316.5	318.5						
DC07-1615	77	798671	318.5	320.5						
DC07-1615	78	798672	320.5	322.5						

Example sample sheet to go to the prep lab

Hole # DGT07-1446 **Mineralization** **Geologist Name:**

Depth		Vein Type					Vein Modifiers								Total Mineralization							
From	To	V1	V2	V3	V4	FeO	Py	As	St	Re	Min1	Amt	Min2	Amt	Py	As	St	Re	Min1	Amt	Min2	Amt
0	5.94																					
5.94	8																					
8	11				1																	
11	14				1																	
14	17				1																	
17	20				1																	
20	23				1																	
23	26				1																	
26	29				1																	
29	32				1																	
32	35																					
35	38				1																	
38	41				1																	
41	44				1																	
44	45.48				1																	
45.48	46.94				3																	
46.94	48				1																	
48	49.55				1																	
49.55	51																					
51	53				2																	
53	54.25				1										1	1						
54.25	55.47														1	1						
55.47	55.82																					
55.82	57		1				4	1							1	1						
57	59														1							
59	61				1										1	1						
61	62.64				1										1							
62.64	63.45				1																	
63.45	64.53				1																	
64.53	65.77				1																	
65.77	67																					
67	69																					
69	71		1		1		1								1							
71	73				1										1							
73	75		1				2								1							
75	77														1							
77	79														1							
79	81														1							
81	83																					
83	85														1	1						
85	87														1							
87	89														1	1						
89	91														1	1						
91	93																					
93	95														1							
95	97																					
97	98																					
98	98.65																					
98.65	98.85																					
98.85	100				1																	
100	102				2																	
102	103.72																					
103.72	104.35															1	1					
104.35	104.52																					
104.52	106.5				1																	
106.5	108				1																	
108	110				1																	
110	112				1																	
112	114				1																	
114	115.04				1																	
115.04	116.13				1																	
116.13	118				1																	
118	119.5				2											1						
119.5	120.75			2	2		1	1	3							1	1	2				
120.75	122.75				2																	
122.75	123.94				2											1	1					
123.94	125.94				2											2						
125.94	127.94				1																	
127.94	130				1																	
130	133				1																	
133	136				1																	
136	139				1																	
139	140.51				1																	
140.51	143				1																	
143	146				1																	
146	149				1																	
149	152				1																	
152	154				1																	
154	156				1																	
156	157.12				1																	
157.12	158.65				2										1							
158.65	159.87														2	1						
159.87	160.47																					

Bold lines indicate mineralization interval for Acquire data entry. Colors: Blue for vein data; Yellow for total mineralization data.

The Terra spectrometer, generally called “PIMA” is an instrument that measures the wavelengths of light reflected from lighter colored minerals for purposes of identification.

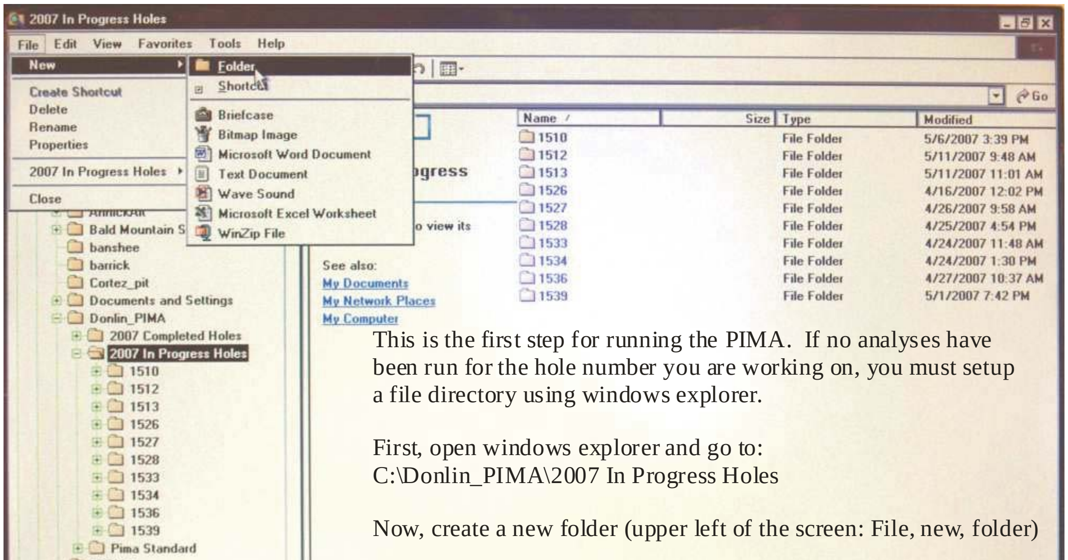
At Donlin Creek, it is believed that many of the clay minerals are associated with the gold as it was deposited from hydrothermal processes. As these clay minerals are very difficult to conclusively identify by the naked eye, the PIMA provides a much more accurate and consistent diagnosis.

The following instructions are quite detailed, and thus, may seem complex, but after you have processed a few holes’ worth of samples, it will become a simple process.

Because the minerals we are identifying are microscopic in size, it is very important that the sample be clean and dry. Residual water on the sample will cause the instrument to give a poor spectra, and thus, samples should be completely dry.

It is also important to note that although the individual grains may be microscopic, they generally occur as aggregates from alteration of a larger mineral grain. These aggregates are loosely called phenocrysts by the geologists and they are quite distinctive when compared with the bulk of the rock. It is these phenocrysts that are the “target” of our analyses. Some photos are provided below for reference.

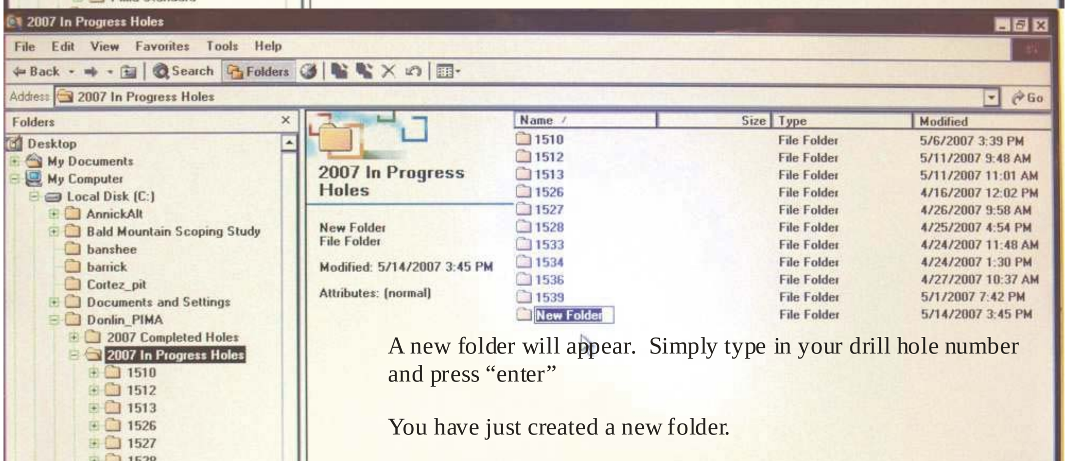
Final note: Turn on the PIMA instrument at least 20 minutes before use.



This is the first step for running the PIMA. If no analyses have been run for the hole number you are working on, you must setup a file directory using windows explorer.

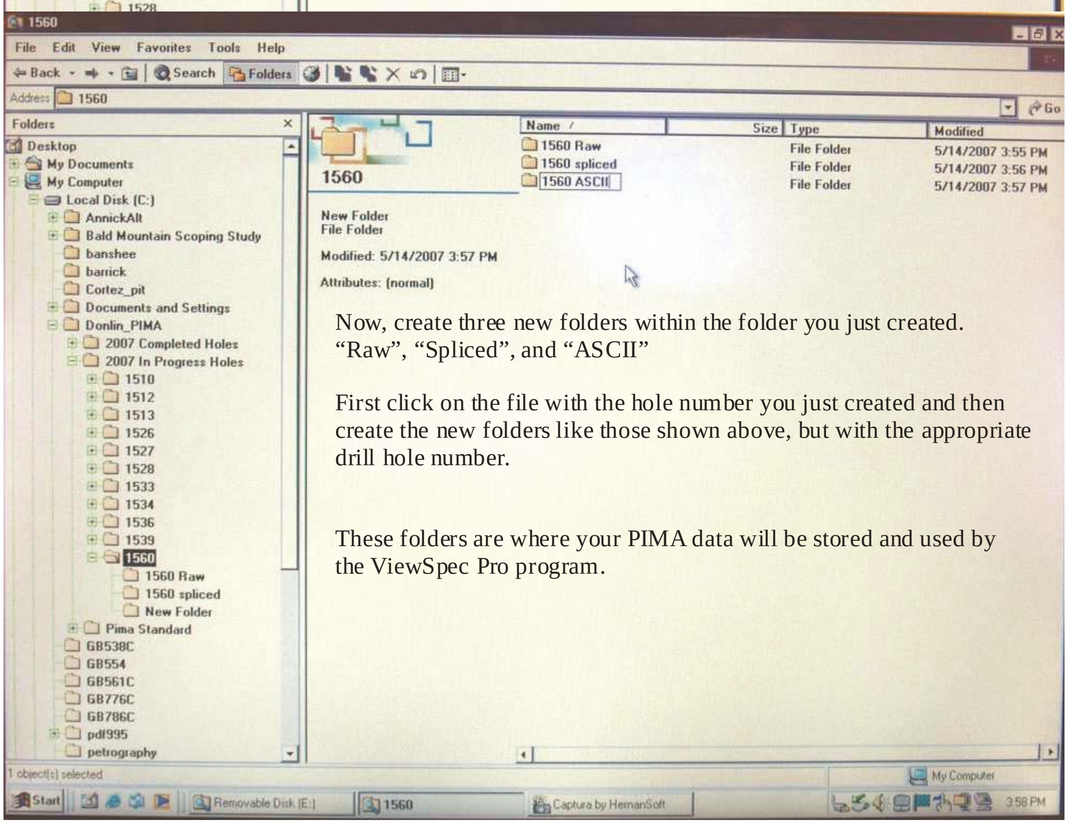
First, open windows explorer and go to:
C:\Donlin_PIMA\2007 In Progress Holes

Now, create a new folder (upper left of the screen: File, new, folder)



A new folder will appear. Simply type in your drill hole number and press "enter"

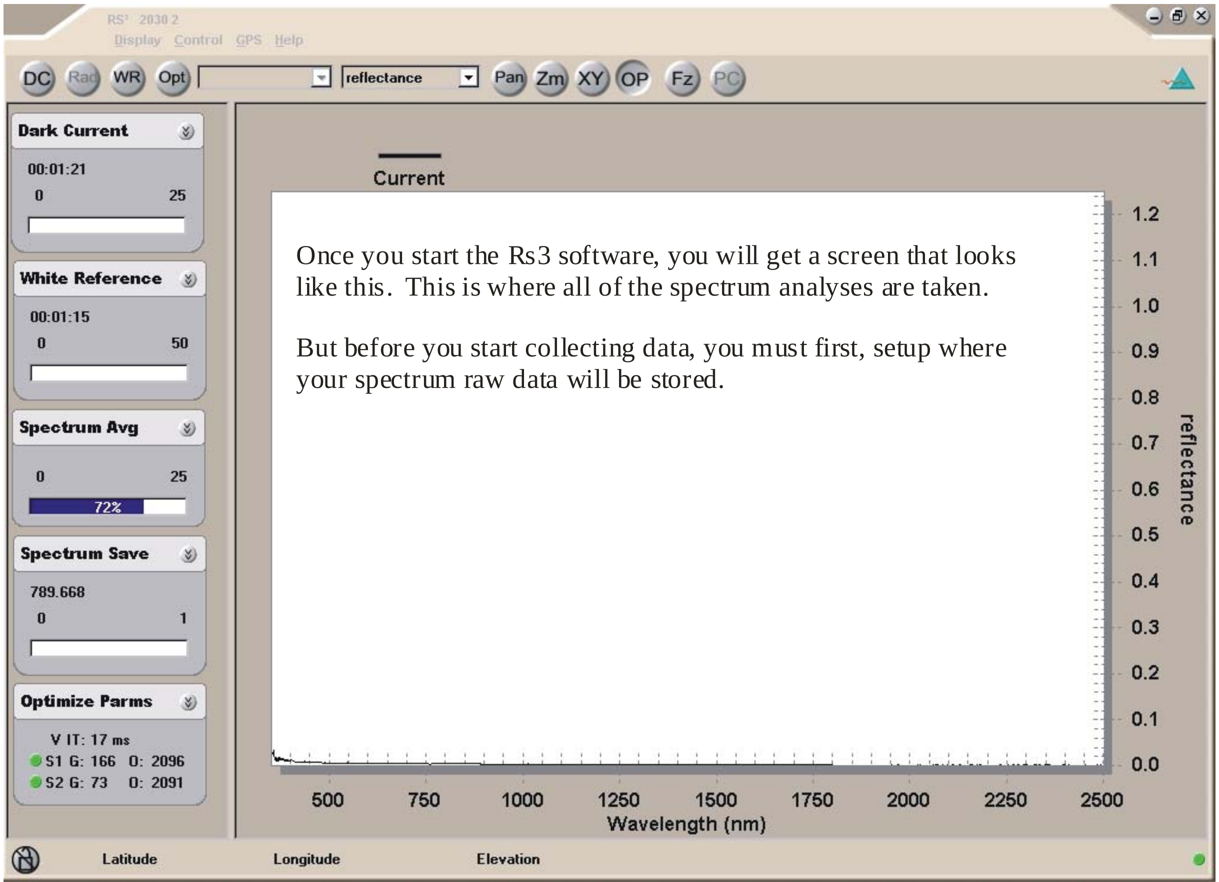
You have just created a new folder.



Now, create three new folders within the folder you just created. "Raw", "Spliced", and "ASCII"

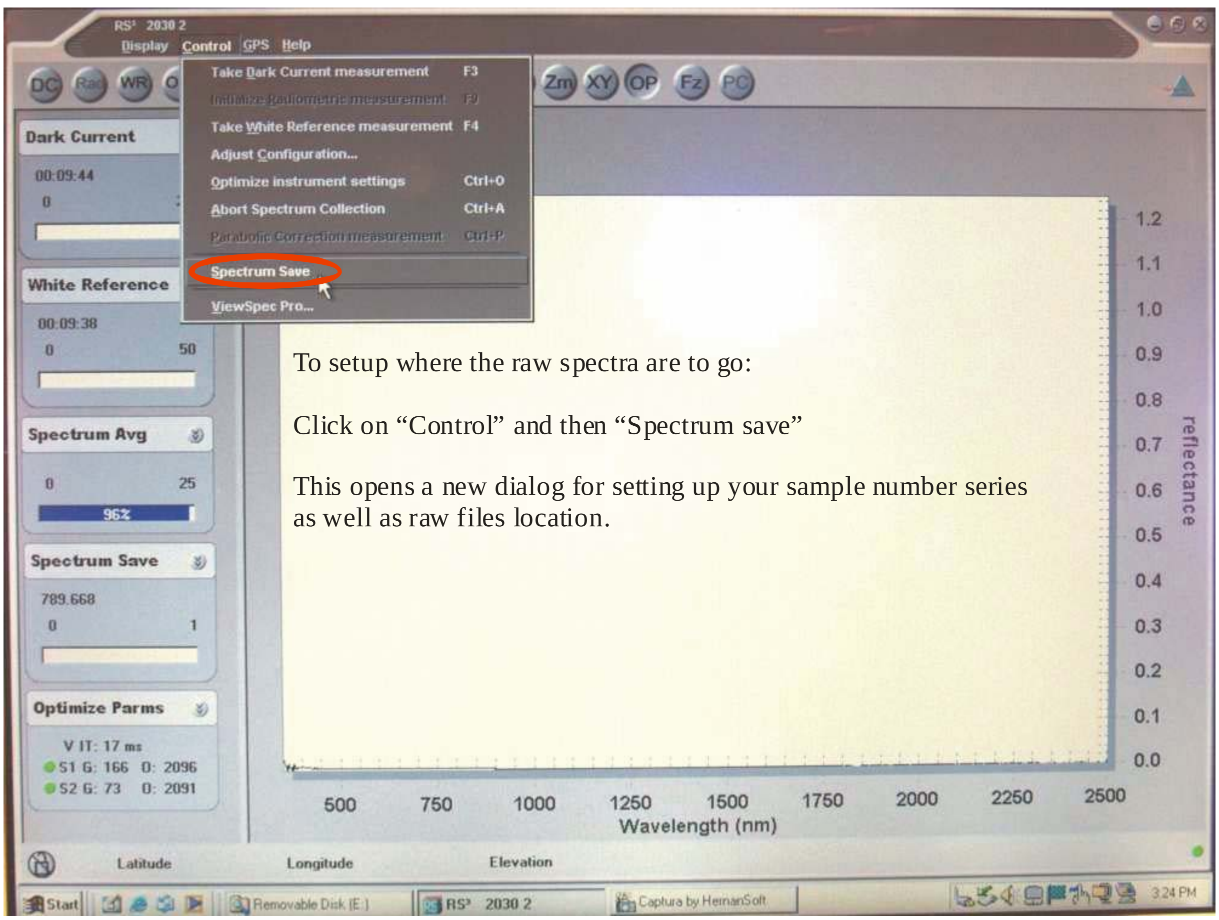
First click on the file with the hole number you just created and then create the new folders like those shown above, but with the appropriate drill hole number.

These folders are where your PIMA data will be stored and used by the ViewSpec Pro program.



Once you start the Rs3 software, you will get a screen that looks like this. This is where all of the spectrum analyses are taken.

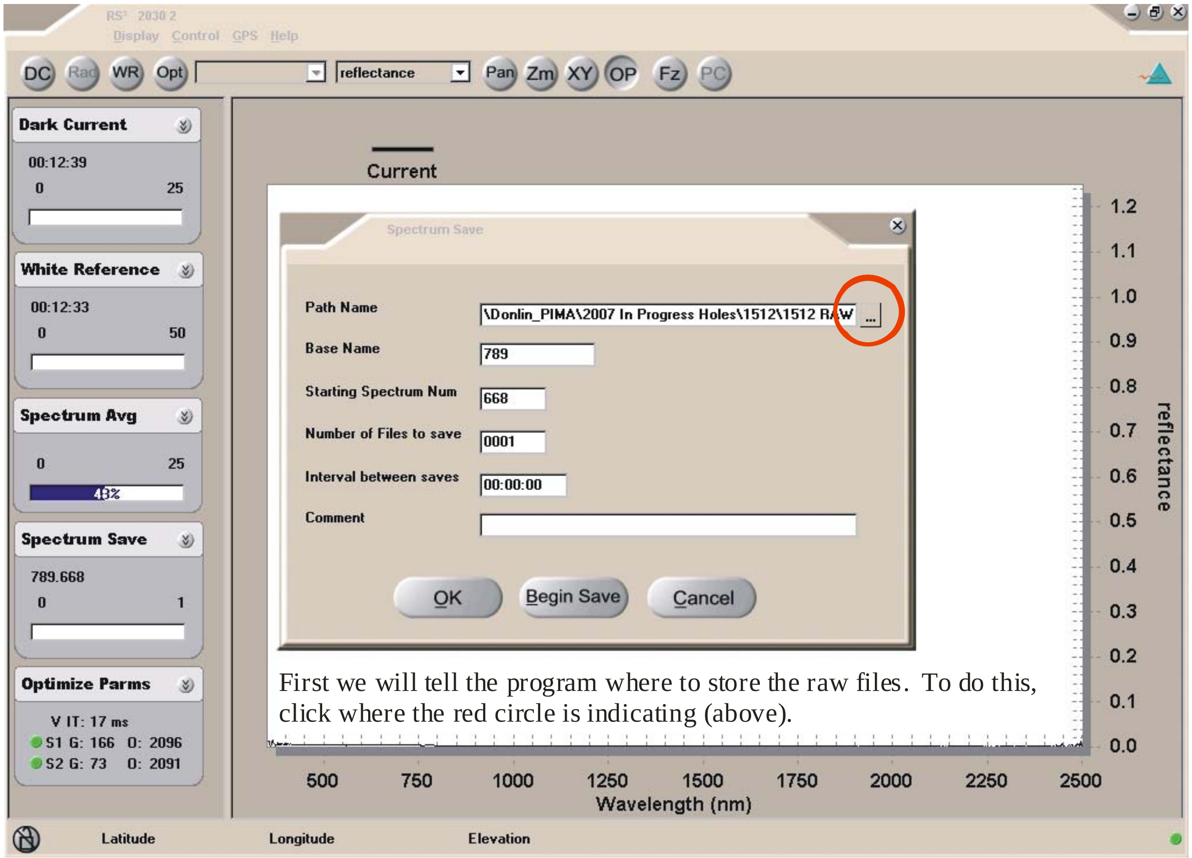
But before you start collecting data, you must first, setup where your spectrum raw data will be stored.



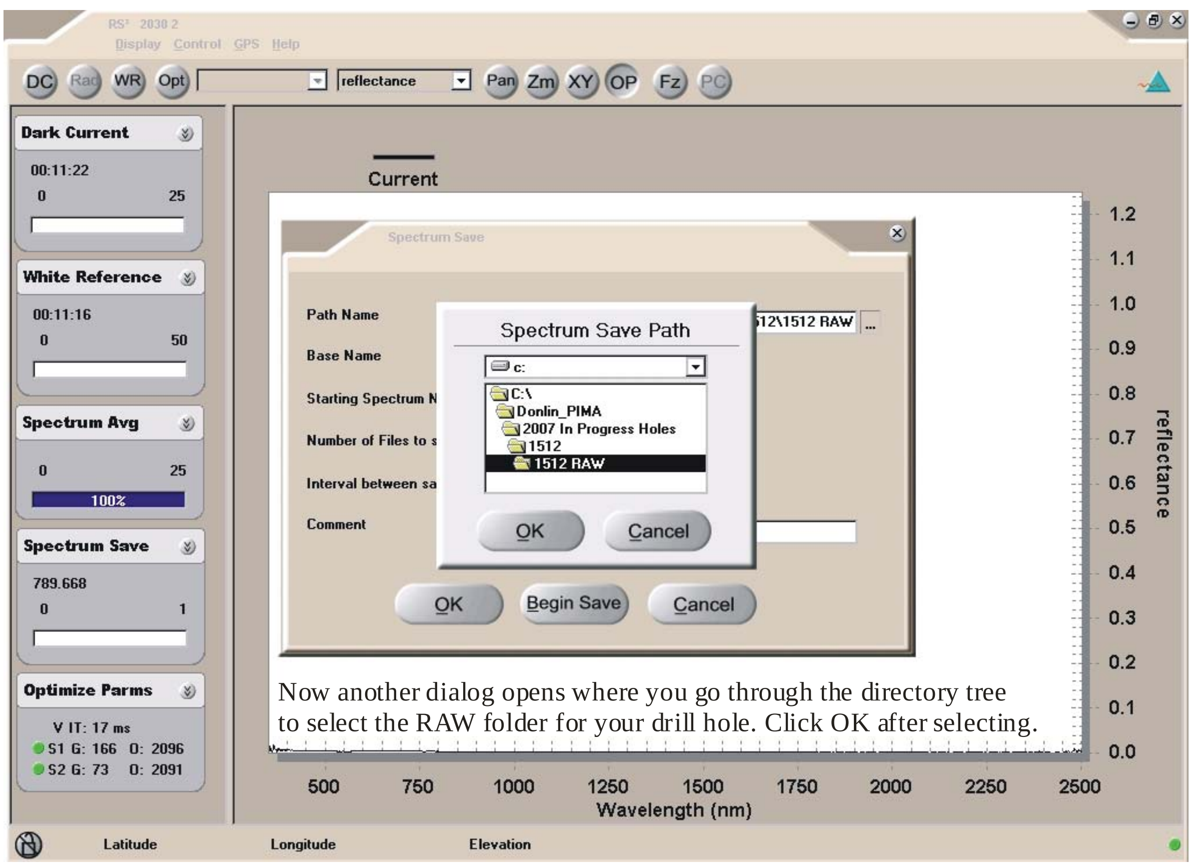
To setup where the raw spectra are to go:

Click on “Control” and then “Spectrum save”

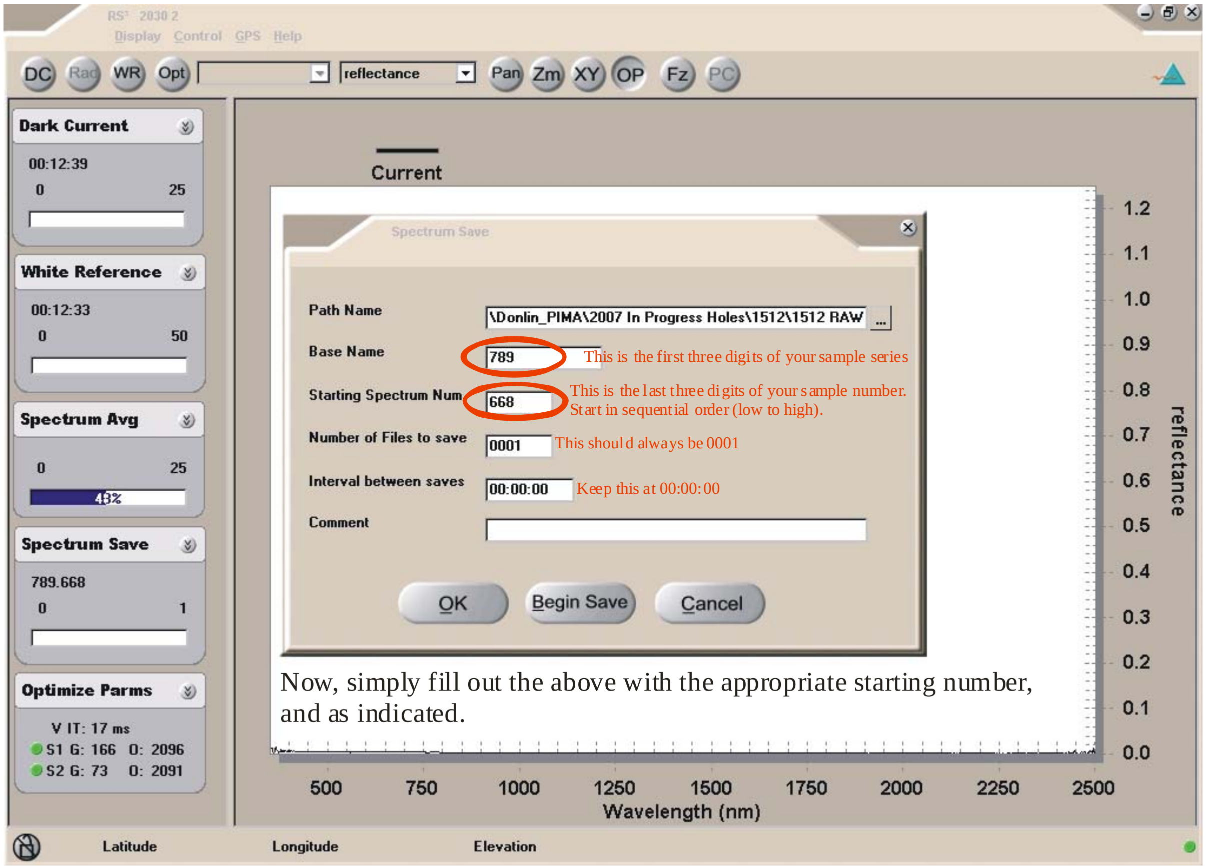
This opens a new dialog for setting up your sample number series as well as raw files location.

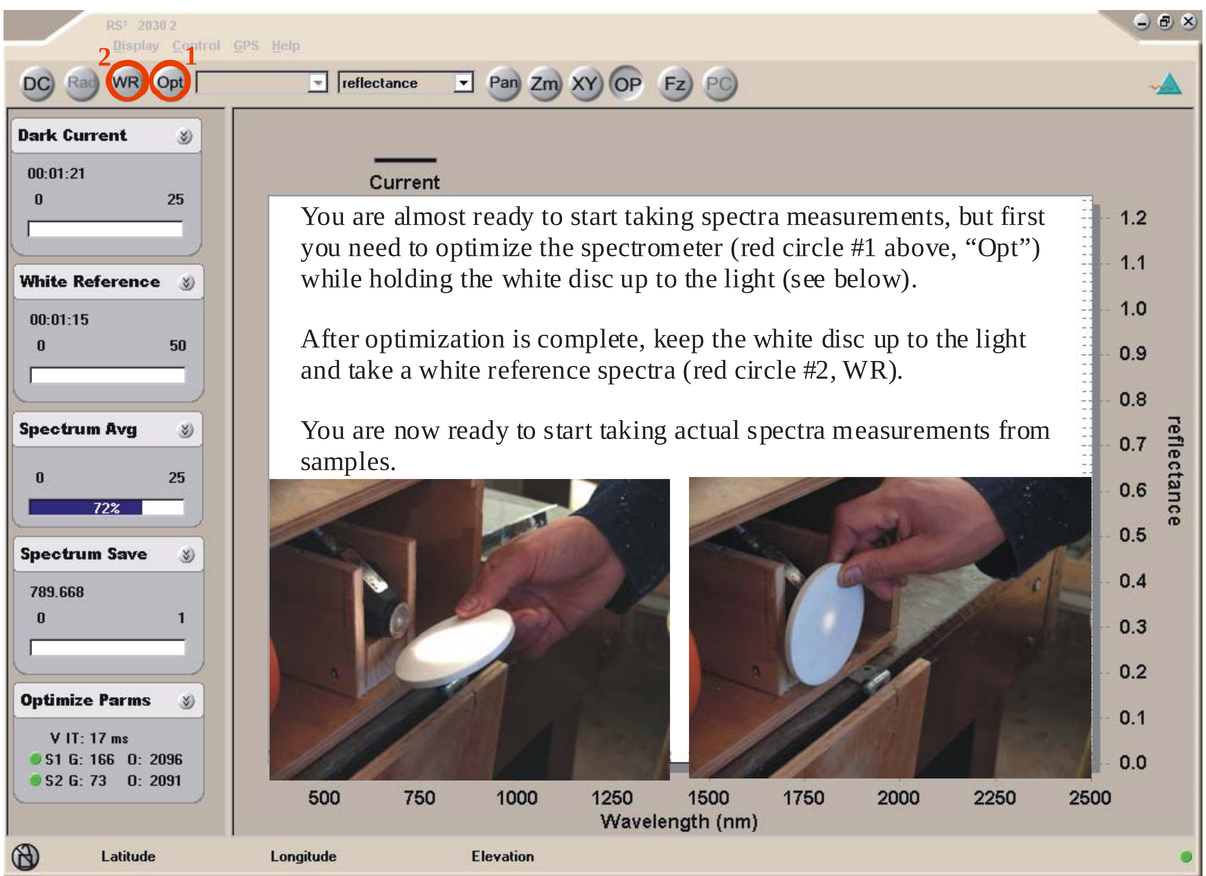


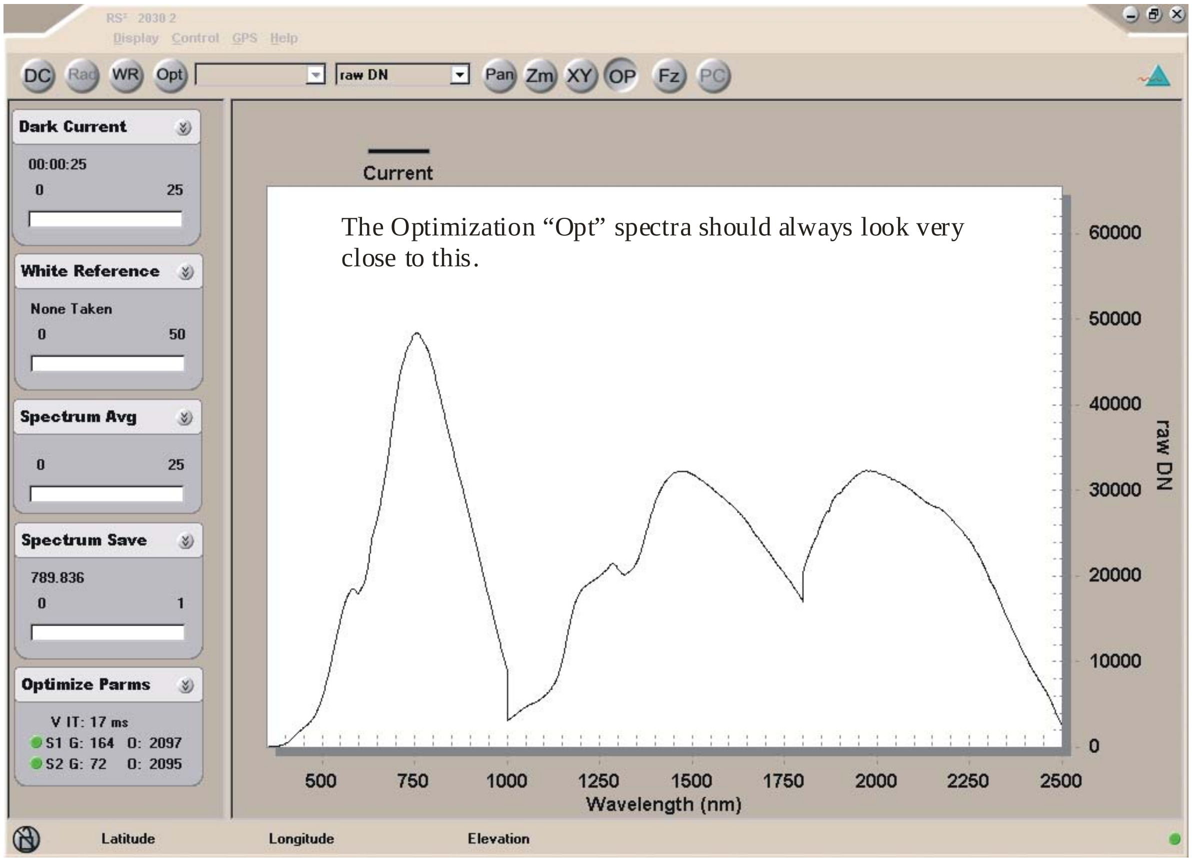
First we will tell the program where to store the raw files. To do this, click where the red circle is indicating (above).

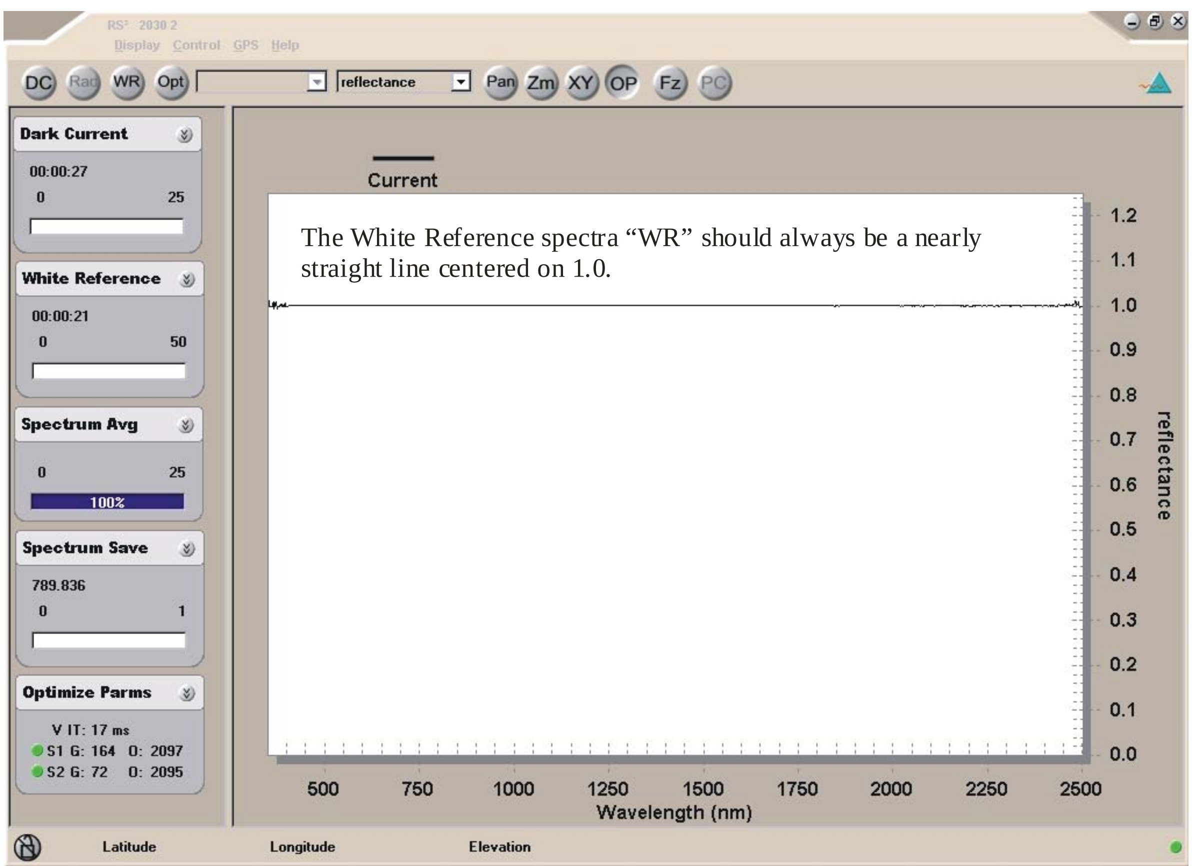


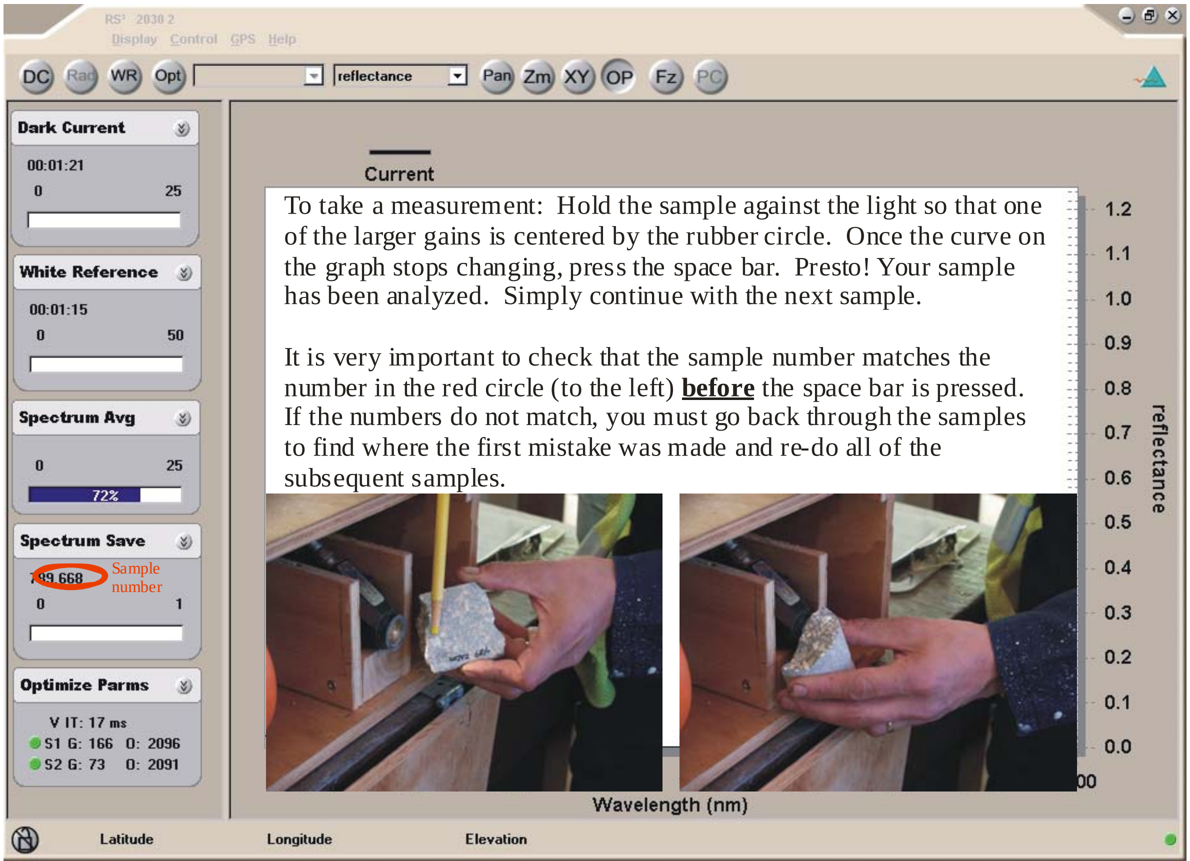
Now another dialog opens where you go through the directory tree to select the RAW folder for your drill hole. Click OK after selecting.


Display Control GPS Help
DC Rad WR Opt reflectance Pan Zm XY OP Fz PC
Dark Current
00:12:39
0 25
White Reference
00:12:33
0 50
Spectrum Avg
0 25
43%
Spectrum Save
789.668
0 1
Optimize Parms
V IT: 17 ms
S1 G: 166 O: 2096
S2 G: 73 O: 2091
Current
Spectrum Save
Path Name
\Donlin_PIMA\2007 In Progress Holes\1512\1512 RAW
Base Name
789
This is the first three digits of your sample series
Starting Spectrum Num
668
This is the last three digits of your sample number. Start in sequential order (low to high).
Number of Files to save
0001
This should always be 0001
Interval between saves
00:00:00
Keep this at 00:00:00
Comment
OK
Begin Save
Cancel
Now, simply fill out the above with the appropriate starting number, and as indicated.
1.2 1.1 1.0 0.9 0.8 0.7 0.6 0.5 0.4 0.3 0.2 0.1 0.0
reflectance
500 750 1000 1250 1500 1750 2000 2250 2500
Wavelength (nm)
Latitude
Longitude
Elevation


Display Control GPS Help
2
1
DC Rad WR Opt reflectance Pan Zm XY OP Fz PC
Dark Current
00:01:21
0 25
White Reference
00:01:15
0 50
Spectrum Avg
0 25
72%
Spectrum Save
789.668
0 1
Optimize Parms
V IT: 17 ms
S1 G: 166 O: 2096
S2 G: 73 O: 2091
Current
You are almost ready to start taking spectra measurements, but first you need to optimize the spectrometer (red circle #1 above, "Opt") while holding the white disc up to the light (see below).
After optimization is complete, keep the white disc up to the light and take a white reference spectra (red circle #2, WR).
You are now ready to start taking actual spectra measurements from samples.
1.2 1.1 1.0 0.9 0.8 0.7 0.6 0.5 0.4 0.3 0.2 0.1 0.0
reflectance
500 750 1000 1250 1500 1750 2000 2250 2500
Wavelength (nm)
Latitude
Longitude
Elevation

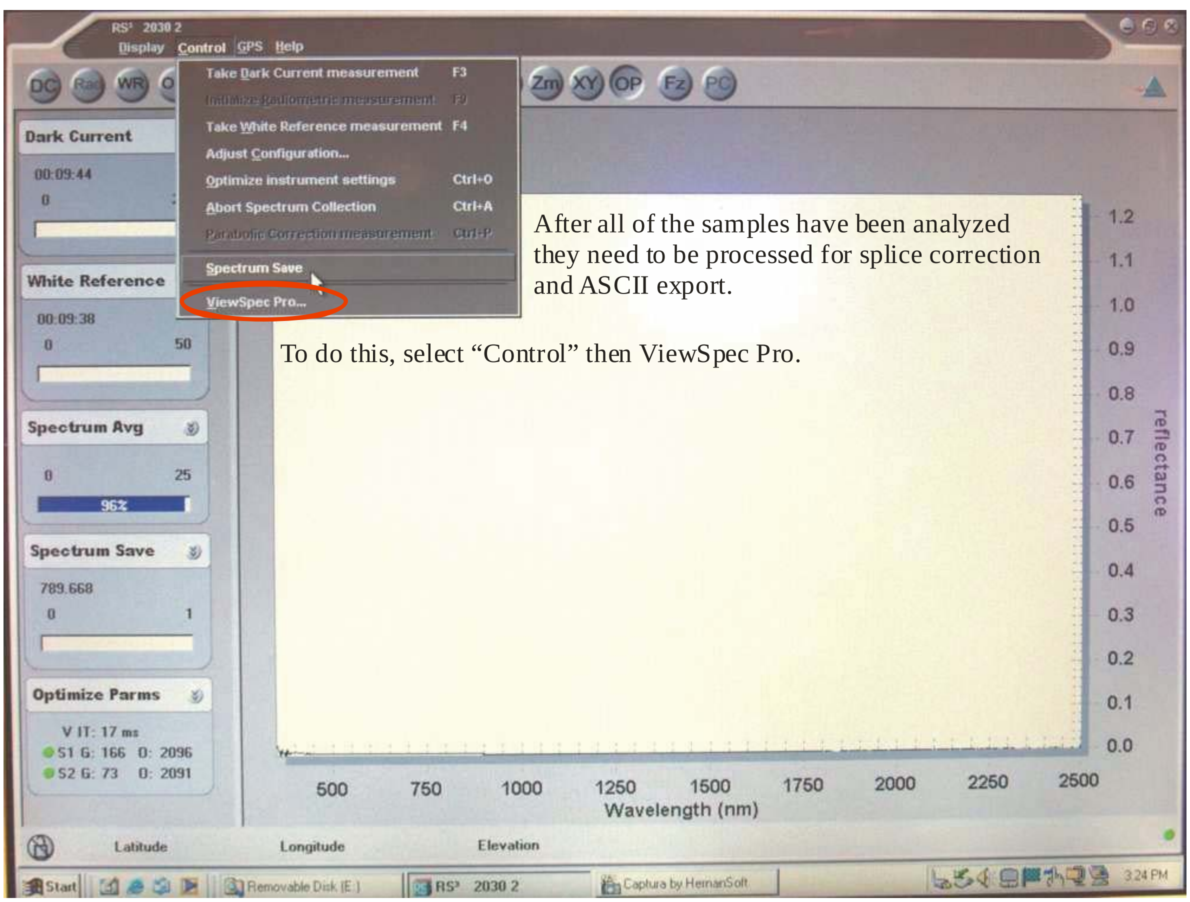

RS³ 2030 2
Display Control GPS Help
DC Rad WR Opt
raw DN
Pan Zm XY OP Fz PC
Dark Current
00:00:25
0 25
White Reference
None Taken
0 50
Spectrum Avg
0 25
Spectrum Save
789.836
0 1
Optimize Parms
V IT: 17 ms
S1 G: 164 O: 2097
S2 G: 72 O: 2095
Current
The Optimization "Opt" spectra should always look very close to this.
60000
50000
40000
30000
20000
10000
0
raw DN
500 750 1000 1250 1500 1750 2000 2250 2500
Wavelength (nm)
Latitude
Longitude
Elevation


RS³ 2030 2
Display Control GPS Help
DC Rad WR Opt
reflectance
Pan Zm XY OP Fz PC
Dark Current
00:00:27
0 25
White Reference
00:00:21
0 50
Spectrum Avg
0 25
100%
Spectrum Save
789.836
0 1
Optimize Parms
V IT: 17 ms
S1 G: 164 O: 2097
S2 G: 72 O: 2095
Current
The White Reference spectra "WR" should always be a nearly straight line centered on 1.0.
1.2
1.1
1.0
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0.0
reflectance
500 750 1000 1250 1500 1750 2000 2250 2500
Wavelength (nm)
Latitude
Longitude
Elevation



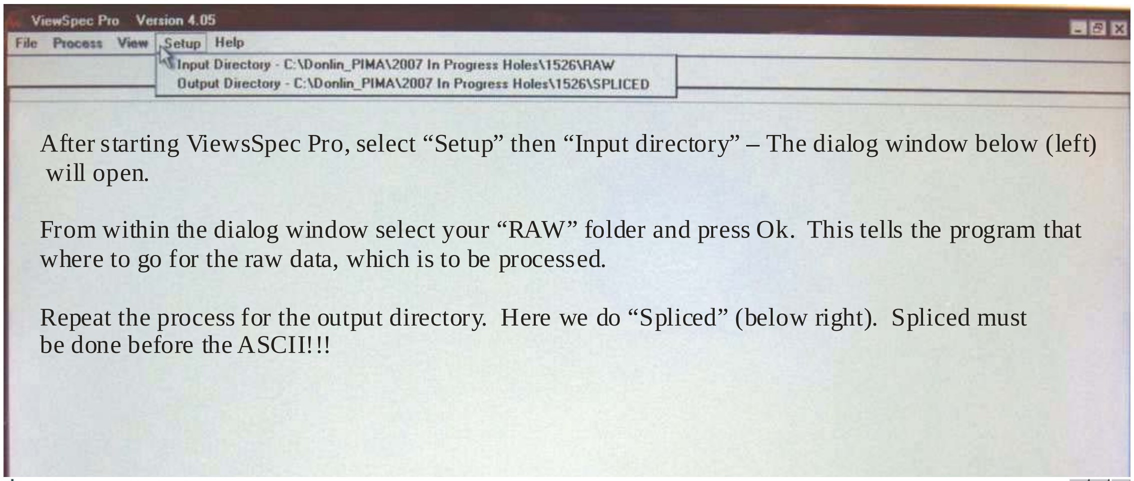
To take a measurement: Hold the sample against the light so that one of the larger gains is centered by the rubber circle. Once the curve on the graph stops changing, press the space bar. Presto! Your sample has been analyzed. Simply continue with the next sample.

It is very important to check that the sample number matches the number in the red circle (to the left) **before** the space bar is pressed. If the numbers do not match, you must go back through the samples to find where the first mistake was made and re-do all of the subsequent samples.



After all of the samples have been analyzed they need to be processed for splice correction and ASCII export.

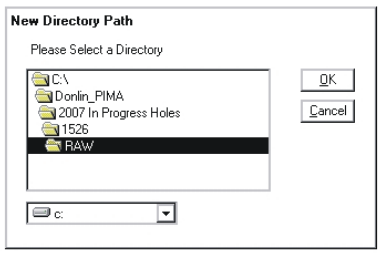
To do this, select "Control" then ViewSpec Pro.

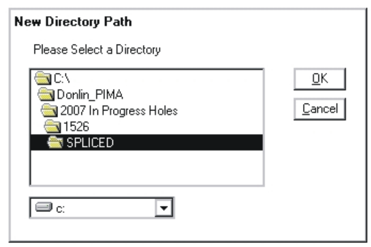


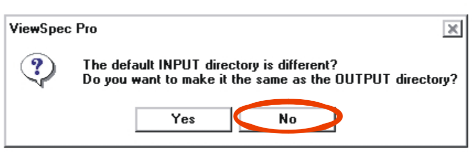
After starting ViewsSpec Pro, select "Setup" then "Input directory" – The dialog window below (left) will open.

From within the dialog window select your "RAW" folder and press Ok. This tells the program that where to go for the raw data, which is to be processed.

Repeat the process for the output directory. Here we do "Spliced" (below right). Spliced must be done before the ASCII!!!





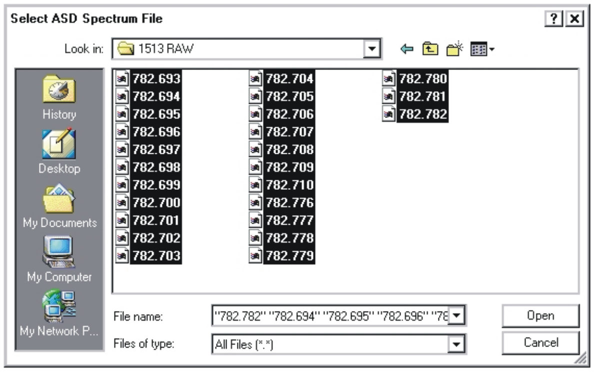


You may get this dialog. If so, select "No."

As our input directory is "Raw" and our output directory is either "Spliced" or "ASCII," our input and output directories are NEVER the same. So, ALWAYS select "No."

After setting up the input and output directories, you need to open and select the raw data to be processed. Simply select “File” then “Open” (red circle) to open the dialog below.

From within this dialog window, open your “Raw” folder.



Make sure the dialog window is in the correct “RAW” directory and showing sample numbers for the analyses you have taken. Click on the first sample number and while holding down on the shift key, click on the last sample--now let up on the shift key. This selects all of your data (now in bold highlight). Press the open button.

Your list of selected raw sample analyses are now open. To process them, simply select “Process” then “Splice Correction.”

Now, that the samples have been splice corrected, you must process them for ASCII Export. BUT, before doing so you must reset the output file to the ASCII folder. To do this simply go “Setup” then select the output directory to be “ASCII”.

Now, go back to “Process” and select “ASCII Export”

ALL DONE!!!

WHEN YOU FINISH, MAKE SURE YOU COPY YOUR MAIN FOLDER ONTO THE BACKUP DRIVE (ask a geologist where).

May 19, 2007

MEMO

From: Brian Flanigan

To: Fawn Glassburn, Heidi Drexler, Kerry Adler, Kyle Linebarger, Leif Bailey, Jennifer Hansom, Orion George, Gabe Kassos, Scott Cereghino, and all other core logging geologists.

Cc: Rich Harris, Chris Valorose, Penny Hobbie, Aziz Jalloh

Subject: Strategy and procedure for turning PIMA over to the geotechs.

Everyone:

I have finally finished the PIMA step-by-step instruction manual. It is incomplete in the sense that there is no good comprehensive explanation for sample selection and sample setup. In other words, among other things, I have not explained in the instruction manual that the last three numbers need to be written on their samples, thing like this will have to be shown. So there are still a few minor things that they will need to be taught that are not written out. So, this will fall on you and me in the beginning—have patience and do not do the work for them. Show them for a couple of samples and then walk them through it, but they should be doing the steps, not you. If you simply show them by writing on the samples and working the mouse yourself, it is going to take you a long, long time to free yourself from the pima duties.

In the beginning there will likely be some confusion as to which holes will need to be analyzed and which ones don't. This is because core logging will have holes that have been and have not been "pima-ed." As time passes core cutting will have no already "pima-ed" holes.

In order to combat confusion in core cutting, we will now TYPE OR WRITE AT THE TOP OF EACH SAMPLE SHEET "STAGE FOR PIMA." This will let core cutting know that they do not have to shrink wrap the pallet after cutting as it needs to be staged at TENT ZERO for the geotechs to pima.

To eliminate confusion during this transition, paletted holes TO BE "pima-ed" should be flagged on the top boxes with blue flagging by core cutting. We should have no problem getting core cutting to comply with this as flagging boxes is much easier than shrink-wrapping them.

At some point we will be getting a "PIMA sheet query" from Penny for a PIMA sheet export from acquire. The export sheets will only show sample intervals that are intrusive (only pima compatible samples). We will file these "PIMA sheets" in a manila folder named "HOLES FOR PIMA." Once we have these sheets back from the geotechs (with check marks by each sample), we will file them in another folder named "PIMA COMPLETED." Later, we can let the geotechs file them, but only after everyone is comfortable with the process.

Until we get a PIMA export from acquire set up, we will have to create a PIMA sheet for the geotechs. I suggest copying the xls sample sheet and converting it into a pima sheet. This is less than ideal as it will show ALL samples instead of simply intrusive samples. Thus, we will have to be available to tell the geotechs which intervals to analyze. So, hopefully we will be getting the query/export utility from Penny soon.

Once the geotechs are finished analyzing a hole they are to shrink wrap the pallets for that hole and clearly mark the hole number on each so that they may be taken down to the bone-yard.

Finally… as it turns out, I will be on break during this transition. No, I did not plan it this way, but take heart, it will probably take at least a week for the first of the holes to make it through core cutting. So, I will probably be back before things can get too chaotic.

Order of flow:

1) Drill hole is logged as per the usual fashion (minus PIMA)

2) Sample Sheets go to core cutting with "STAGE FOR PIMA" written on the top of each.

3) PIMA SHEET exported/created and filed in the geo office.

4) Core is cut and sampled.

5) Cut core is blue flagged and staged for PIMA at tent zero (big door side).

6) Geotechs get the PIMA SHEET from the geo office.

7) Geotechs PIMA the hole.

8) Geotechs finish the hole: label pallets by hole number and shrink wrap for destiny in the bone yard.

07/06/2007

MEMO

From: Brian Flanigan

To: Geotechs

Subject: Modifications to Pima Procedure

PIMA STEPS:

1) START WITH COMPLETELY CUT HOLES FIRST

2) PIMA HOLE IN ITS ENTIRETY (IF POSSIBLE)

3) LABEL PALLETS, SHRINK WRAP AND FLAG ORANGE THE FINISHED PALLETS.

4) ONLY COMPLETED HOLES GO TO THE BONEYARD, PARTIAL HOLES WAIT UNTIL COMPLETION. (THIS IS SO THAT THE PALLETS STAY TOGETHER BY HOLE NUMBER.)

5) AFTER PROCESSING RAW FILES COPY THEM INTO THE *2007 COMPLETED HOLES* FOLDER, AND REMOVE THEM FROM THE *2007 IN PROGRESS HOLES* FOLDER.

6) COPY THE FILES IN THE COMPLETED HOLES FOLDER ONTO THE THUMB DRIVE.

7) COPY ALL HOLE FOLDERS ON THE THUMB DRIVE ONTO THE K-DRIVE: K:\2007\2007 ASD (PIMA)\Spectra

8) GO TO: K:\2007\2007 ASD (PIMA)\PIMASHEETS\To_Be_PIMAed_sheets FIND YOUR HOLE .XLS FILE AND MOVE IT INTO: K:\2007\2007 ASD (PIMA)\PIMASHEETS\PIMA_Completed_sheets.

9) EDIT ANY CHANGES YOU MADE TO THE *.XLS SHEET TO REFLECT ADDITIONAL AND/OR DELETED INTERVALS/SAMPLE NUMBERS. ALSO CHECK (x) THE BLANK CELLS UNDER “PIMAED” FOR THE INTERVALS PROCESSED.

10) CLEAN THE THUMB DRIVE (DELETE FILES) FOR THE NEXT USER.

11) ON THE PIMA LAPTOP: CLEAN THE FILES YOU BACKED UP FROM THE *HOLES COMPLETED* FOLDER.

12) FILE PAPER COPY IN THE FILING CABINET IN THE GEOLOGY OFFICE

This file is located on: K:\2007\2007 ASD (PIMA)\ PIMA STEPS.DOC

May 16, 2007

From: Brian Flanigan
Acting Quality Control Manager

To: Bill Bieber, Rich Harris, Glenn Marshall

Cc: Samantha, Elena, John and Memo
Sample Lab Supervisors

Subject: Organizational changes in Core Cutting and the Prep Lab

Everyone:

Samantha, Elena and I have made some minor changes in the organization of how uncut/unprocessed core is to flow through the Sample Processing Facility. These changes were made to accommodate both quality as well as efficiency. Both John and Memo will have to be brought up to speed. I will meet with them before I depart for break on the 22nd.

The new process is as follows:

1) The sequence in which holes are sawed now rely on the sample sheets to guide from submittal number to submittal number as effectively as possible.

   The result is that drill holes are still cut in their entirety, but when a hole is finished being sawed, the last submittal number guides the supervisor to what hole should be photographed and ready prior to the end of the "previous" hole with the same submittal number. For example, saw #2 is cutting hole number 1528, when this hole is finished it is in the middle of submittal #71, which continues in hole number 1537. Thus, 1537 should be already photographed and ready to saw, before 1528 is finished. See flow sheet below:

SAW #1	SAW #2	SAW #3	SAW #4	SAW #5	SAW #6
DC07-1499	**DC07-1528**	**DC07-1529**	**DC07-1527**	**DC07-1531**	**DC07-1526**
S#45	S#54	S#48	S#51	S#57	S#43
DC07-1527(already thru SAW #4)	S#55	S#52(end)	S#53	S#58	S#44
	S#62		S#60	**DC07-1534**	S#56
	S#67		**DC07-1536**	S#58	**DC071531**
	S#71		S#60	S#68	S#56
	DC07-1537		S#66	S#69	S#57
	S#71		S#75	S#74	S#58
	S#76		S#79	**DC07-1538**	**DC071534**
	S#80		S#85	S#74	S#58
	S#86		**DC07-1510**		
	wait n see		S#85		

The above flow sheet is an example from early last week.

   The organization of drill hole sequence through each saw is the responsibility of the core cutting supervisors. In order for the supervisors to accomplish this task they must get a copy of the Submittal Tracking Sheet periodically from the geologists as needed (THEY WILL HAVE TO ASK FOR IT). The Submittal Tracking Sheet provides a synopsis of

Submittal numbers and related hole numbers along with sample number sequences. The Sheet also shows which submittals have already been processed/shipped, so that we know which submittal numbers are of immediate concern and which are not.

2) As the core is cut, core boxes are organized by hole number on pallets just as before on past years projects. However, samples are put onto carts by submittal number. This is very important so that any later core pulls by the geologists can be easily retrieved in the bone yard by hole number. And… Having samples organized on carts by submittal number enables much easier flow of samples through the prep lab with fewer incomplete submittals taking up precious space in waiting.

   On the prep lab side, Memo and Elena have suggested that crush rejects for each sample go into bins by submittal number. Although this is different than in the past, when rejects were organized by hole number, we have found this to be a crucial change to greatly enhance efficiency, and thus, the change has been made. As each bin holds 78 samples adequately, with little room left over, the change works out well. For later reference, all bins are labeled with corresponding submittal number and any related drill hole numbers.

PRODUCTIVITY TRACKING

I have instituted more comprehensive and ongoing productivity and progress tracking than was used in the past. I found this important for the planning of the required workforce, materials and tools.

With assistance of the core cutting supervisor I have been tracking the daily output of core cut in meters, rather than in number of boxes or samples. I felt the addition of meters was necessary as not all boxes are of equal length, particularly when comparing NQ and HQ size core. As I also track the daily output of logged core by the geologists, I can more easily judge the productivity/progress in context with the project needs. For instance, right now stability in the core cutting workforce, and patience with new hires as they are getting up to speed needs to be our primary focus before any dramatic changes (e.g. night shift or additional saws) are put in force.

Productivity of the prep lab is very much dependent on core cutting. Right now their output is easily matching that of core cutting. I am providing productivity graphs for each for your viewing. We have had some set-backs in core cutting personnel numbers and a new crew since the 12th of May, as the decline in productivity reflects.

Final note: I am open to any suggestions, but please realize there have been a lot of recent changes and we are still getting our feet wet and settling in.

Sincerely,

Brian Flanigan

2007 SUBMITTAL/BATCH NUMBER FOR EACH GROUP OF 78								
SUBMITTAL NO	HOLE IDs	SAMPLE # BEGIN	SAMPLE # END	count	#78 Filled?	Misc Notes	SHIPPED?	Date Shipped
DC07-308	DC07-1593	795971	796000	30	Y	Terry	Y	8/11/2007
		798001	798007	7				
	DC07-1598	798008	798048	41				
DC07-309	DC07-1592	812305	812350	46	Y	Orion		
	DC07-1611	812351	812382	32				
DC07-310	DC07-1598	798049	798126	78	Y	Terry	Y	8/13/2007
DC07-311	DC07-1600	796731	796796	66	Y	Gabe/Jen H.	Y	8/13/2007
	DC07-1603	796797	796808	12		Leif		
DC07-312	DC07-1602	797419	797496	78	Y	Orion	Y	8/6/2007
DC07-313	DC07-1602	797497	797574	78	Y	Orion	Y	8/9/2007
DC07-314	DC07-1602	797575	797652	78	Y	Orion	Y	8/8/2007
DC07-315	DC07-1601	801327	801404	78	Y	Jennifer H.	Y	8/11/2007
DC07-316	DC07-1601	801405	801482	78	Y	Jennifer H. and Scott	Y	8/10/2007
DC07-317	DC07-1597	791483	791537	55	Y	Jennifer H.		
	DC07-1607	791538	791560	23				
DC07-318	DC07-1603	796809	796886	78	Y	Leif	Y	8/13/2007
DC07-319	DC07-1603	796887	796938	52	Y	Leif		
	DC07-1604	796939	796964	26				
DC07-320	DC07-1602	797653	797730	78	Y	Orion	Y	8/10/2007
DC07-321	DC07-1602	797731	797736	6	Y	Orion	Y	8/13/2007
	DC07-1606	797737	797808	72				
DC07-322	DC07-1598	798127	798204	78	Y	Terry		
DC07-323	DC07-1599	793883	793960	78	Y	Jennifer G.	Y	8/13/2007
DC07-324	DC07-1606	797809	797886	78	Y	Orion	Y	8/13/2007
DC07-325	DC07-1601	801483	801560	78	Y	Scott	Y	8/13/2007
DC07-326	DC07-1604	796965	797000	36	Y	Leif		
		814001	814042	42				
DC07-327	DC07-1598	798205	798237	33	Y	Terry		
	DC07-1612	798238	798282	45				
DC07-328	DC07-1604	814043	814120	78	Y	Leif		
DC07-329	DC07-1599	793961	794000	40	Y	Jennifer G.		
		799001	799017	17				
	DC07-1605	799018	799038	21				
DC07-330	DC07-1606	797887	797964	78	Y	Orion	Y	8/13/2007
DC07-331	DC07-1606	797965	797981	17	Y	Orion		
	DC07-1611	797982	798000	19				
	DC07-1611	800001	800042	42				
DC07-332	DC07-1601	801561	801567	7	Y	Scott	Y	8/13/2007
	DC07-1608	801568	801638	71				
DC07-333	DC07-1612	798283	798360	78	Y	Terry		
DC07-334	DC07-1604	814121	814198	78	Y	Leif		
DC07-335	DC07-1604	814199	814200	2	Y	Leif		
	DC07-1613	814201	814276	76				
DC07-336	DC07-1608	801639	801716	78	Y	Scott		
DC07-337	DC07-1611	812383	812460	78	Y	Orion		
DC07-338	DC07-1612	798361	798438	78	Y	Terry		
DC07-339	DC07-1605	799039	799116	78	Y	Jennifer G.		
DC07-340	DC07-1609	800043	800120	78	Y	Orion		
DC07-341	DC07-1613	814277	814354	78	Y	Leif		
DC07-342	DC07-1613	814355	814432	78	Y	Leif		
DC07-343	DC07-1611	812461	812492	32	Y	Orion		
	DC07-1610	812493	812538	46		Gabe		
DC07-344	DC07-1610	812539	812616	78	Y	Gabe		
DC07-345	DC07-1605	799117	799155	39	Y	Jennifer G.		
	DC07-1614	799156	799194	39				
DC07-346	DC07-1612	798439	798473	35	Y	Terrell		
	DC07-1615	798474	798516	43				
DC07-347	DC07-1609	800121	800198	78	Y	Orion/Gabe		
DC07-348	DC07-1608	801717	801794	78	Y	Scott		
DC07-349	DC07-1608	801795	801827	33	Y	Scott		
	DC07-1618	801828	801872	45				
DC07-350	DC07-1614	799195	799272	78	Y	Jennifer G.		
DC07-351	DC07-1607	791561	791638	78	Y	Jen H.		
DC07-352	DC07-1607	791639	791650	12	Y	Jen H.		
	DC07-1616	791651	791716	66				
DC07-353	DC07-1613	814433	814493	61	Y	Leif		
	DC07-1622	814494	814510	17				
DC07-354	DC07-1609	800199	800200	2	N	Gabe		
	DC07-1621	800201	800254	54				
DC07-355	DC07-1614	799273	799279	7	Y	Jennifer G.		
	DC07-1620	799280	799350	71				
DC07-356	DC07-1615	798517	798594	78	Y	Terry		
DC07-357	DC07-1610	812617	812694	78	Y	Gabe		
DC07-358	DC07-1622	814511	814588	78	Y	Leif		
DC07-359	DC07-1622	814589	814666	78	Y	Leif		

Example of the Submittal tracking sheet

PHOTO APPENDIX A

ROCK UNITS


SHL/GWK, ~70/30
GWK/SHL, ~60/40
GWK/SHL, ~80/20
GWK/SHL, ~80/20
GWK with V4's
GWK
GWK
GWK/SHL, ~60/40


MD
MD
md
MD: Dark and light phase
md
md
gwk
gwk
md
MD
MD
RDA
SHL
RDA
RDA


SHL
RDX with Fe-oxidation
RDX with Fe-oxidation
RDX with Fe-ox and possible
ankerite alteration of Felds.
RDX
RDX
RDX
RDX
RDF with thin V1 sulfide veins

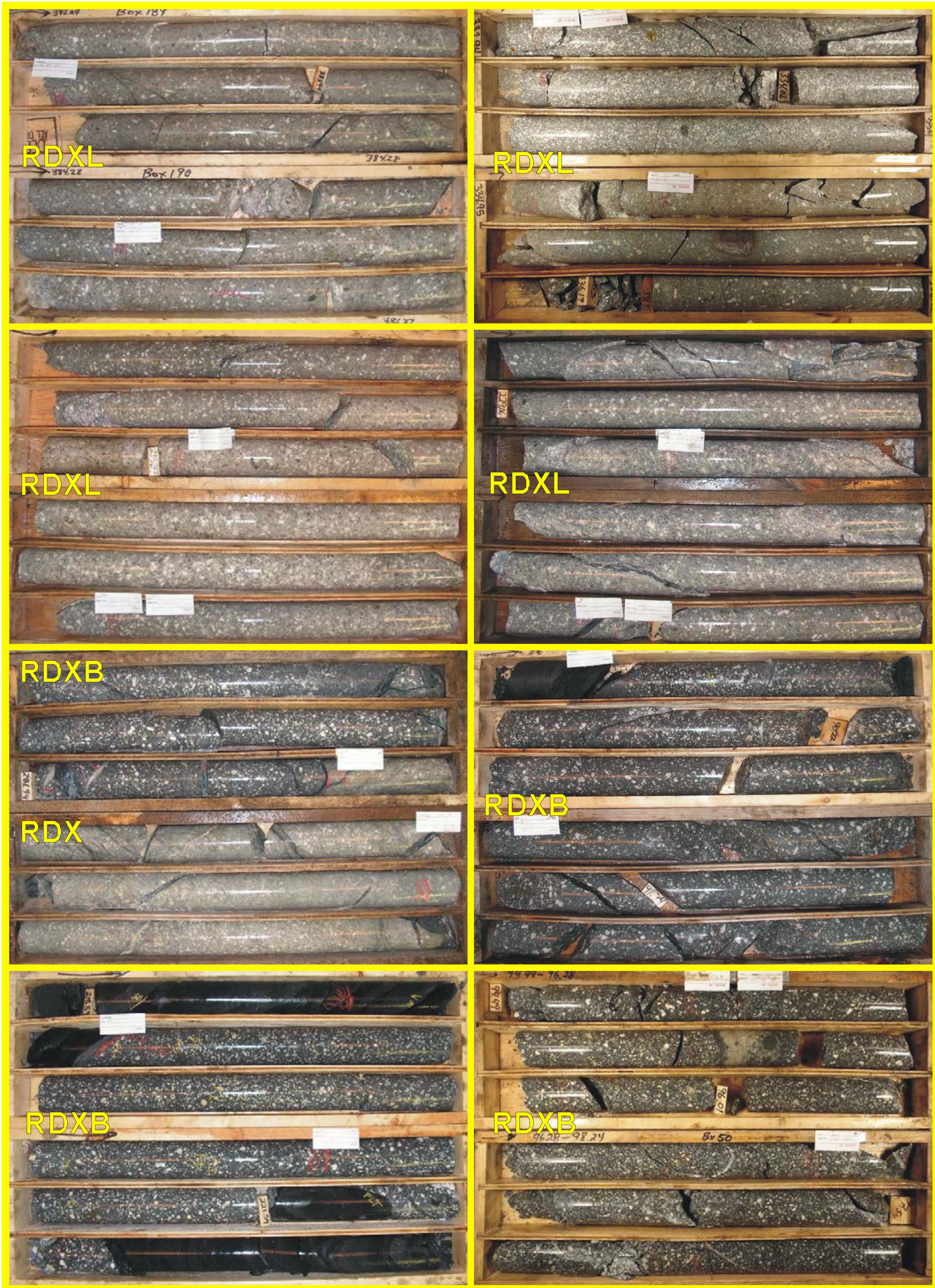

RDXL
RDXL
RDXL
RDXL
RDXB
RDX
RDXB
RDXB
RDXB

PHOTO APPENDIX B

SULFIDE AND VEIN MINERALIZATION


Hydrothermal vein breccia with stibnite (V3)
Hydrothermal vein breccia with stibnite (V3)
Disseminated (syngenetic?) pyrite
Hydrothermal vein breccia with stibnite +/- native arsenic (V3, BX)
Hydrothermal vein breccia with stibnite +/- native arsenic (V3, BX)
Typical barren carbonate veins (V4)
Hydrothermal vein breccia with stibnite +/- native arsenic (V3, BX)
Hydrothermal vein with native arsenic, arsenopyrite and sphalerite


Vein opiment, realgar,
+/- stibnite and native arseinc
(V3)
Vein orpiment and realgar
(V3)
Vein (V3) and disseminated
orpiment and realgar
Vein orpiment and realgar (V3)
Disseminated and vein (V3)
orpiment and realgar
Disseminated orpiment
Massive sulfide vein
Massive sulfide vein: pyrite,
pyrrhotite, arsenopyrite and
chalcopyrite (V1)

APPENDIX B

6 m Composites Exceeding 4 g/t

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
AT99-2-1	6	12	6	5.88	5.88	RDX
AT99-2-1	18	20	2	7.90	7.90	RDX
AT99-2-2	18	24	6	5.42	5.42	SED
AT99-2-4	18	22	4	4.02	4.02	RDX
DC01-586	52.88	58.88	6	7.90	7.90	RDX
DC01-586	58.88	64.88	6	13.98	13.98	RDX
DC01-586	94.88	100.88	6	8.65	8.65	SHL
DC01-586	124.88	130.88	6	4.70	4.70	RDA
DC01-586	142.88	148.88	6	4.01	4.01	RDA
DC01-586	148.88	154.88	6	4.54	4.54	RDA
DC01-586	244.88	250.88	6	11.17	11.17	RDA
DC01-586	256.88	262.88	6	5.40	5.40	RDA
DC01-587	0	6	6	4.98	4.91	OVB
DC01-587	18	24	6	5.77	5.77	RDXB
DC01-587	48	54	6	4.02	4.02	RDX
DC01-587	54	60	6	4.28	4.28	RDX
DC01-587	66	72	6	7.10	7.10	RDA
DC01-587	72	78	6	7.79	7.79	RDA
DC01-587	90	96	6	5.01	5.01	RDX
DC01-587	102	108	6	6.22	6.22	RDX
DC01-587	108	114	6	9.27	9.27	RDX
DC01-587	126	132	6	10.74	10.74	RDX
DC01-587	132	138	6	7.41	7.41	RDX
DC01-587	138	144	6	13.48	12.71	RDX
DC01-587	144	150	6	4.99	4.99	RDX
DC01-587	168	171.6	3.6	6.14	6.14	GWK
DC01-588	63.96	69.96	6	4.14	4.14	RDX
DC01-589	21.66	27.66	6	5.65	5.65	RDXB
DC01-589	63.66	69.66	6	7.33	7.33	RDX
DC01-589	75.66	81.66	6	4.91	4.91	RDX
DC01-590	12.4	18.4	6	5.98	5.98	RDXB
DC01-591	10.57	16.57	6	9.21	9.21	SHL
DC01-591	16.57	22.57	6	4.51	4.51	RDXB
DC01-592	68	74	6	6.74	6.74	RDXB
DC01-593	67.44	73.44	6	4.03	4.03	RDX
DC01-593	73.44	79.44	6	10.83	10.83	RDX
DC01-593	85.44	91.44	6	5.51	5.51	SHL
DC01-594	60	66	6	4.04	4.04	RDA
DC01-595	29.18	35.18	6	4.58	4.58	RDA
DC01-595	35.18	41.18	6	9.25	9.25	RDA
DC01-595	41.18	47.18	6	10.92	10.92	RDX
DC01-595	131.18	137.18	6	6.51	6.51	RDA
DC01-595	143.18	149.18	6	13.95	13.95	RDX
DC01-595	149.18	155.18	6	18.09	12.79	GWK
DC01-595	209.18	215.18	6	6.00	6.00	RDA
DC01-595	215.18	221.18	6	4.70	4.70	RDA
DC01-595	221.18	227.18	6	4.16	4.16	RDA
DC01-595	227.18	233.18	6	5.04	5.04	RDA
DC01-596	2.3	8.3	6	9.12	9.12	RDXB
DC01-596	8.3	14.3	6	8.47	8.47	RDXB
DC01-596	14.3	20.3	6	6.07	6.07	RDXB
DC01-596	20.3	26.3	6	5.22	5.22	RDXB
DC01-596	26.3	32.3	6	7.90	7.90	RDXB
DC01-596	68.3	74.3	6	6.27	5.96	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC01-597	8.44	14.44	6	5.49	5.49	RDA
DC01-597	14.44	20.44	6	4.27	4.27	RDA
DC01-597	20.44	26.44	6	18.70	17.99	RDX
DC01-597	26.44	32.44	6	4.25	4.25	RDX
DC01-597	50.44	56.44	6	4.50	4.50	SLT
DC01-597	56.44	62.44	6	7.91	7.91	RDX
DC01-597	62.44	68.44	6	7.06	7.06	RDX
DC01-597	68.44	74.44	6	45.13	22.62	RDX
DC01-597	74.44	80.44	6	26.19	10.95	RDX
DC01-597	86.44	92.44	6	6.02	6.02	GWK
DC01-597	98.44	104.44	6	4.29	4.29	GWK
DC01-597	146.44	152.44	6	6.08	6.08	RDA
DC01-598	15.35	21.35	6	4.67	4.67	RDX
DC01-599	6	12	6	7.29	7.29	RDXB
DC01-599	12	18	6	4.13	4.13	SHL
DC01-599	198	204	6	4.23	4.23	RDA
DC01-599	228	234	6	4.44	4.44	RDX
DC01-600	29.33	35.33	6	5.83	5.83	GWK
DC01-600	41.33	47.33	6	6.62	6.62	RDXB
DC01-600	71.33	77.33	6	5.85	5.85	RDXB
DC01-600	95.33	101.33	6	5.43	5.43	GWK
DC01-600	101.33	107.33	6	41.16	20.20	RDXB
DC01-600	107.33	113.33	6	16.49	16.49	RDXB
DC01-600	113.33	119.33	6	12.72	8.24	SHL
DC01-601	11.18	17.18	6	4.85	4.85	RDXB
DC01-601	17.18	23.18	6	4.30	4.30	RDXB
DC01-601	35.18	41.18	6	4.67	4.67	RDX
DC01-602	129.56	135.56	6	7.47	7.47	RDXB
DC01-602	147.56	149.96	2.4	6.04	6.04	GWK
DC01-603	45.96	51.96	6	8.79	8.79	RDA
DC01-603	63.96	69.96	6	8.59	8.59	RDX
DC01-603	69.96	75.96	6	7.44	7.44	RDX
DC01-603	75.96	81.96	6	9.14	9.14	RDX
DC01-603	87.96	93.96	6	4.10	4.10	RDX
DC01-603	123.96	129.96	6	5.86	5.86	RDA
DC01-603	129.96	135.96	6	9.14	9.14	RDA
DC01-604	42	48	6	5.16	5.16	GWK
DC01-604	48	54	6	19.75	10.87	SHL
DC01-604	72	78	6	5.75	5.75	GWK
DC01-605	34.57	40.57	6	4.23	4.23	GWK
DC01-605	76.57	82.57	6	5.08	5.08	GWK
DC01-605	94.57	100.57	6	6.46	6.46	RDX
DC01-605	100.57	106.57	6	13.34	13.34	RDXB
DC01-605	106.57	112.57	6	5.36	5.36	RDXB
DC01-606	6	12	6	4.62	4.16	OVB
DC01-607	39.25	45.25	6	6.28	6.28	GWK
DC01-607	69.25	75.25	6	6.66	6.66	RDA
DC01-607	93.25	99.25	6	7.37	7.37	RDA
DC01-607	99.25	105.25	6	4.70	4.70	RDA
DC01-607	105.25	111.25	6	16.34	16.34	RDX
DC01-607	111.25	117.25	6	17.97	17.97	RDX
DC01-607	117.25	123.25	6	12.15	11.04	RDA
DC01-607	123.25	129.25	6	55.07	28.85	RDX
DC01-607	129.25	135.25	6	27.81	14.05	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC01-607	135.25	141.25	6	6.10	6.10	RDX
DC01-608	42	48	6	8.53	8.53	RDXB
DC01-608	180	186	6	4.78	4.78	RDX
DC01-608	198	204	6	5.28	5.28	RDX
DC01-609	51.25	57.25	6	6.14	6.14	RDA
DC01-609	57.25	63.25	6	13.91	13.91	RDA
DC01-609	69.25	75.25	6	4.10	4.10	RDA
DC01-609	123.25	129.25	6	5.69	5.69	RDA
DC01-609	129.25	135.25	6	7.87	7.87	RDA
DC01-609	135.25	141.25	6	5.73	5.73	RDA
DC01-609	165.25	171.25	6	7.11	7.11	SHL
DC01-610	27.33	33.33	6	4.74	4.74	GWK
DC01-610	39.33	45.33	6	7.23	7.23	RDX
DC01-610	45.33	51.33	6	4.80	4.80	RDX
DC01-610	87.33	93.33	6	4.16	4.16	SLT
DC01-611	14.47	20.47	6	4.54	4.54	RDX
DC01-611	32.47	38.47	6	4.63	4.63	RDX
DC01-611	80.47	86.47	6	4.01	4.01	GWK
DC01-611	110.47	116.47	6	19.63	17.45	GWK
DC01-611	128.47	134.47	6	5.38	5.38	RDXL
DC01-611	140.47	146.47	6	5.22	5.22	RDX
DC01-611	146.47	152.47	6	4.69	4.69	RDX
DC01-611	152.47	158.47	6	9.68	9.68	RDX
DC01-611	158.47	164.47	6	9.63	9.63	RDX
DC01-611	170.47	176.47	6	7.07	7.07	RDX
DC01-611	182.47	188.47	6	6.25	6.25	RDX
DC01-611	188.47	194.47	6	4.56	4.56	RDX
DC01-611	218.47	224.47	6	6.24	6.24	RDX
DC01-611	248.47	254.47	6	5.51	5.51	RDXL
DC01-612	111.35	117.35	6	4.54	4.54	RDXL
DC01-612	129.35	135.35	6	5.56	4.82	RDXL
DC01-613	9.05	15.05	6	6.20	6.20	RDX
DC01-613	27.05	33.05	6	9.46	9.46	GWK
DC01-613	45.05	51.05	6	6.62	6.59	RDX
DC01-613	51.05	57.05	6	18.77	17.76	RDX
DC01-613	123.05	129.05	6	4.24	4.24	RDX
DC01-613	213.05	219.05	6	7.19	7.19	RDX
DC01-615	60	66	6	9.06	9.06	RDX
DC01-615	66	72	6	5.08	5.08	RDX
DC01-615	114	120	6	5.43	5.43	RDX
DC01-615	120	126	6	13.23	13.23	RDX
DC01-615	126	132	6	10.27	10.27	RDX
DC01-617	108	114	6	4.39	4.39	RDX
DC01-617	132	138	6	9.32	9.32	RDX
DC01-617	138	144	6	6.47	6.47	RDA
DC01-618	24.9	30.9	6	4.18	4.18	RDX
DC01-618	30.9	36.9	6	4.40	4.40	RDX
DC01-619	30	36	6	6.32	6.32	RDXB
DC01-619	96	102	6	5.14	5.14	RDX
DC01-619	102	108	6	10.52	10.52	RDX
DC01-619	108	114	6	14.40	14.40	RDX
DC01-619	114	120	6	9.39	9.39	RDX
DC01-619	210	216	6	7.11	7.11	RDA
DC01-619	222	228	6	5.74	5.74	RDA

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC01-619	228	234	6	16.04	16.04	RDA
DC01-619	234	240	6	15.06	15.06	RDA
DC01-619	240	246	6	4.43	4.43	RDA
DC01-619	258	264	6	5.07	5.07	RDA
DC01-619	264	270	6	6.58	6.58	RDA
DC01-619	270	276	6	7.31	7.31	RDA
DC01-619	276	282	6	12.17	12.17	RDA
DC01-619	282	288	6	12.30	11.83	GWK
DC01-620	22.5	28.5	6	5.79	5.79	RDX
DC01-620	28.5	34.5	6	5.30	5.30	RDX
DC01-620	34.5	40.5	6	4.10	4.10	RDX
DC01-621	90	96	6	4.57	4.57	RDX
DC01-622	109.98	115.98	6	4.49	4.49	RDA
DC01-625	141	147	6	8.90	8.90	RDXB
DC01-625	147	153	6	5.67	5.67	RDXB
DC01-625	243	249	6	7.52	7.52	GWK
DC01-625	303	309	6	6.08	6.08	RDA
DC01-625	309	315	6	14.58	14.58	RDA
DC01-625	315	321	6	5.47	5.47	RDA
DC01-625	333	339	6	14.27	14.27	RDXB
DC01-625	339	345	6	5.50	5.50	SHL
DC01-625	345	351	6	11.12	11.12	RDA
DC01-625	363	369	6	5.97	5.97	RDA
DC01-625	369	375	6	7.75	7.75	RDA
DC01-625	375	381	6	6.45	6.45	RDA
DC02-673	100	106	6	6.99	6.99	RDXB
DC02-673	106	112	6	11.95	11.95	RDXB
DC02-673	112	118	6	6.71	6.71	RDXB
DC02-675	78	84	6	6.83	6.83	RDA
DC02-675	84	90	6	6.26	6.26	RDA
DC02-675	102	108	6	4.89	4.89	RDA
DC02-675	108	114	6	10.28	10.28	RDA
DC02-675	114	120	6	4.97	4.97	RDA
DC02-675	120	126	6	4.98	4.98	RDX
DC02-675	126	132	6	4.03	4.03	RDX
DC02-675	144	150	6	4.17	4.17	RDX
DC02-675	156	162	6	4.16	4.16	RDX
DC02-675	186	192	6	11.46	11.46	RDA
DC02-677	4	10	6	4.22	4.22	RDA
DC02-677	10	16	6	4.31	4.31	RDX
DC02-677	58	64	6	6.97	6.97	RDA
DC02-677	82	88	6	6.87	6.87	RDX
DC02-677	88	94	6	4.54	4.54	RDX
DC02-677	94	100	6	10.11	10.11	RDX
DC02-677	100	106	6	5.76	5.76	RDX
DC02-677	112	118	6	5.29	5.29	RDX
DC02-677	118	124	6	13.47	13.47	RDX
DC02-677	124	130	6	11.75	8.87	RDX
DC02-677	136	142	6	6.79	6.79	RDX
DC02-677	154	160	6	4.02	4.02	RDX
DC02-677	160	166	6	5.05	5.05	RDX
DC02-677	166	172	6	5.71	5.71	RDX
DC02-677	178	184	6	8.03	8.03	RDX
DC02-677	184	190	6	5.87	5.87	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC02-677	190	196	6	11.12	11.12	RDX
DC02-679	6.37	12.37	6	7.16	7.16	RDX
DC02-679	24.37	30.37	6	4.24	4.24	RDX
DC02-679	48.37	54.37	6	4.55	4.55	RDX
DC02-679	54.37	60.37	6	5.90	5.90	RDX
DC02-679	60.37	66.37	6	4.93	4.93	RDX
DC02-679	78.37	84.37	6	5.03	5.03	RDX
DC02-679	84.37	90.37	6	12.58	12.58	RDX
DC02-681	36	42	6	4.10	4.10	RDA
DC02-681	48	52.75	4.75	5.20	5.20	RDA
DC02-681	54.75	56.75	2	5.11	5.11	RDA
DC02-681	57	59	2	6.49	6.49	RDA
DC02-681	96.75	102.75	6	4.05	4.05	RDX
DC02-681	132.75	138.75	6	4.43	4.43	RDX
DC02-681	138.75	144.75	6	11.63	11.57	RDX
DC02-681	144.75	150.75	6	11.12	10.34	GWK
DC02-681	150.75	156.75	6	4.41	4.41	RDA
DC02-681	174.75	180.75	6	5.43	5.43	GWK
DC02-681	180.75	186.75	6	4.65	4.65	RDA
DC02-681	186.75	192.75	6	26.72	19.24	GWK
DC02-683	12	18	6	6.12	6.12	RDA
DC02-683	18	24	6	4.31	4.31	RDA
DC02-683	30	36	6	6.14	6.14	RDX
DC02-683	84	90	6	6.53	6.53	RDX
DC02-683	138	144	6	9.03	9.03	SHL
DC02-683	192	198	6	7.29	7.29	RDA
DC02-684	13.3	14	0.7	4.26	4.26	RDXB
DC02-687	170.6	176.6	6	4.61	4.61	RDXL
DC02-687	188.6	194.6	6	4.04	4.04	RDXL
DC02-688	69.14	75.14	6	8.65	8.65	RDX
DC02-689	56.91	62.91	6	6.26	6.26	RDA
DC02-689	62.91	68.91	6	4.03	4.03	RDA
DC02-691	75	81	6	4.15	4.15	RDXL
DC02-693	78	84	6	5.31	5.31	RDX
DC02-697	79.85	85.85	6	6.17	6.17	RDA
DC02-697	85.85	91.85	6	4.75	4.75	RDA
DC02-698	48	54	6	5.18	5.18	RDA
DC02-698	54	60	6	7.21	7.21	RDA
DC02-700	104.56	110.56	6	5.32	5.32	RDXB
DC02-703	36	42	6	5.87	5.87	RDA
DC02-703	42	48	6	4.69	4.69	RDA
DC02-705	112	118	6	6.21	6.21	RDA
DC02-711	92.47	98.47	6	4.02	4.02	RDA
DC02-711	98.47	104.47	6	9.53	9.53	RDA
DC02-712	152	158	6	7.05	7.05	RDX
DC02-715	64.75	70.75	6	6.04	6.04	RDX
DC02-715	94.75	99.97	5.22	9.77	8.19	GWK
DC02-722	144	150	6	4.75	4.75	RDXB
DC02-723	113.47	119.47	6	77.75	22.66	MD
DC02-723	119.47	125.47	6	7.89	7.89	GWK
DC02-723	137.47	143.47	6	8.08	6.54	GWK
DC02-727	116.84	122.84	6	7.95	7.95	GWK
DC02-727	128.84	134.84	6	7.92	7.92	GWK
DC02-727	152.84	158.84	6	6.11	6.11	GWK

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC02-729	25.65	31.65	6	13.60	13.60	GWK
DC02-730	153.69	159.69	6	4.72	4.72	RDX
DC02-733	70.88	76.88	6	4.42	4.42	RDX
DC02-734	154.8	160.8	6	5.73	5.73	GWK
DC02-735	96	102	6	6.89	6.89	RDX
DC02-735	108	114	6	4.71	4.71	RDX
DC02-739	25.8	31.8	6	5.83	5.83	SHL
DC02-739	85.8	91.8	6	10.78	10.08	GWK
DC02-740	59.4	65.4	6	5.46	5.46	RDXL
DC02-740	65.4	71.4	6	14.19	8.51	RDXL
DC02-741	107.16	113.16	6	10.56	10.56	RDA
DC02-741	113.16	119.16	6	9.11	9.11	RDA
DC02-743	314	320	6	5.15	5.15	RDXL
DC02-743	320	326	6	5.52	5.52	RDXL
DC02-744	101	107	6	5.20	5.20	RDXL
DC02-744	107	113	6	14.48	8.10	RDXL
DC02-744	113	119	6	5.55	5.55	RDXL
DC02-745	126	132	6	5.55	5.55	GWK
DC02-745	156	162	6	4.06	4.06	RDA
DC02-745	192	198	6	5.08	5.08	RDA
DC02-749	120	126	6	7.38	7.38	GWK
DC02-749	144	150	6	5.85	5.85	GWK
DC02-750	33.55	39.55	6	4.49	4.49	GWK
DC02-750	39.55	45.55	6	5.86	5.86	SHL
DC02-851	39	45	6	5.21	4.37	RDXL
DC02-851	51	57	6	6.75	6.75	RDXL
DC02-854	166.86	172.86	6	14.07	13.47	GWK
DC02-855	274	280	6	4.90	4.90	RDX
DC02-855	310	316	6	10.21	10.21	RDX
DC02-855	316	322	6	5.65	5.65	RDX
DC02-855	349.07	355.07	6	13.36	12.53	GWK
DC02-855	385.07	391.07	6	6.22	6.22	RDXL
DC02-855	391.07	397.07	6	5.89	5.89	RDXL
DC02-855	409.07	415.07	6	5.66	5.66	RDXL
DC02-856	134	140	6	6.66	6.44	MD
DC02-856	140	146	6	17.46	13.13	GWK
DC02-857	178	184	6	5.42	5.42	RDA
DC02-857	202	208	6	4.11	4.11	RDA
DC02-858	89.67	95.67	6	8.20	8.20	SHL
DC02-859	80	86	6	5.21	5.21	RDXB
DC02-859	116	122	6	6.10	6.10	SHL
DC02-859	268	274	6	6.35	6.35	RDA
DC02-859	274	280	6	8.79	8.79	RDA
DC02-859	280	286	6	9.33	9.33	RDA
DC02-859	286	292	6	10.69	10.69	RDA
DC02-859	352	358	6	8.66	8.66	RDA
DC02-859	358	364	6	12.35	12.35	RDA
DC02-859	406	412	6	6.22	6.22	RDA
DC02-859	436	442	6	5.30	5.30	RDA
DC02-859	448	454	6	4.15	4.15	RDA
DC02-859	454	460	6	4.26	4.26	RDA
DC02-860	152	158	6	4.40	4.40	RDXB
DC02-860	176	182	6	11.96	11.96	RDXB
DC02-860	182	188	6	7.00	7.00	RDXB

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC02-860	188	194	6	6.98	6.98	RDXB
DC02-861	180	186	6	6.43	6.43	RDXB
DC02-861	186	192	6	8.75	8.75	RDXB
DC02-861	388	394	6	9.13	9.13	RDA
DC02-861	394	400	6	5.17	5.17	RDA
DC02-861	400	406	6	11.93	11.93	RDA
DC02-861	406	412	6	15.97	15.97	RDA
DC02-861	496	502	6	6.01	6.01	RDA
DC02-862	234	240	6	4.58	4.58	RDX
DC02-863	160	166	6	8.30	8.30	RDXB
DC02-863	166	172	6	5.13	5.13	RDXB
DC02-863	272.51	278.51	6	7.34	7.34	GWK
DC02-863	290.51	296.51	6	4.28	4.28	GWK
DC02-863	302.51	308.51	6	5.20	5.20	GWK
DC02-863	332.51	338.51	6	8.40	8.40	RDA
DC02-863	338.51	344.51	6	9.20	9.20	RDA
DC02-863	344.51	350.51	6	8.12	8.12	RDA
DC02-863	374.51	380.51	6	5.55	5.55	GWK
DC02-863	380.51	386.51	6	7.30	7.30	RDXB
DC02-863	410.51	416.51	6	15.39	15.39	RDXB
DC02-863	416.51	422.51	6	4.05	4.05	RDXB
DC02-863	482.51	488.51	6	8.46	8.46	RDA
DC02-864	24.1	30.1	6	6.06	6.06	RDXB
DC02-864	96.1	102.1	6	5.74	5.74	RDX
DC02-864	102.1	108.1	6	4.24	4.24	RDX
DC02-864	108.1	114.1	6	4.09	4.09	RDX
DC02-864	198.1	204.1	6	4.21	4.21	RDX
DC02-864	204.1	210.1	6	6.30	6.30	RDX
DC02-864	210.1	216.1	6	5.78	5.78	RDX
DC02-864	252.1	258.1	6	4.61	4.61	RDA
DC02-865	195.15	201.15	6	4.20	4.20	GWK
DC02-865	363.15	369.15	6	16.56	15.89	GWK
DC02-865	369.15	375.15	6	7.57	7.01	GWK
DC02-866	59.2	65.2	6	4.57	4.57	RDA
DC02-866	77.2	83.2	6	8.50	8.50	RDA
DC02-866	83.2	89.2	6	5.50	5.50	RDA
DC02-866	89.2	95.2	6	5.93	5.93	RDA
DC02-866	95.2	101.2	6	6.50	6.50	RDA
DC02-866	113.2	119.2	6	4.26	4.26	GWK
DC02-866	161.2	167.2	6	23.04	17.81	RDX
DC02-866	167.2	173.2	6	4.05	4.05	RDX
DC02-866	173.2	179.2	6	5.07	5.07	GWK
DC02-866	311.2	317.2	6	5.19	5.19	RDA
DC02-866	317.2	323.2	6	4.25	4.25	RDA
DC02-866	371.2	377.2	6	5.26	5.26	RDA
DC02-866	431.2	437.2	6	7.41	7.41	RDA
DC02-867	303.3	309.3	6	7.05	7.05	RDA
DC02-867	345.3	351.3	6	5.18	5.18	RDX
DC02-867	351.3	357.3	6	16.52	16.52	RDX
DC02-868	95.1	101.1	6	4.63	4.63	RDA
DC02-868	107.1	113.1	6	4.03	4.03	RDX
DC02-868	131.1	137.1	6	6.99	6.99	RDX
DC02-868	239.1	245.1	6	5.42	5.42	RDA
DC02-868	251.1	257.1	6	4.93	4.93	RDA

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC02-869	107.4	113.4	6	11.27	11.27	RDXB
DC02-869	113.4	119.4	6	4.48	4.48	RDXB
DC02-869	185.4	191.4	6	5.34	5.34	RDA
DC02-869	227.4	233.4	6	5.18	5.18	RDA
DC02-869	233.4	239.4	6	9.14	9.14	RDA
DC02-869	257.4	263.4	6	4.92	4.92	RDX
DC02-869	287.4	293.4	6	4.56	4.56	RDX
DC02-869	293.4	299.4	6	5.94	5.94	RDX
DC02-869	299.4	305.4	6	5.21	5.21	RDX
DC02-869	305.4	311.4	6	4.62	4.62	RDX
DC02-870	73.6	79.6	6	6.33	6.33	GWK
DC02-870	187.6	193.6	6	4.03	4.03	RDA
DC02-870	235.6	241.6	6	4.46	4.46	RDA
DC02-870	247.6	253.6	6	5.55	5.55	RDA
DC02-870	307.6	313.6	6	8.71	8.71	RDA
DC02-870	313.6	319.6	6	19.99	19.99	RDA
DC02-870	319.6	325.6	6	9.64	9.64	RDA
DC02-870	325.6	331.6	6	6.19	6.19	RDA
DC02-870	343.6	349.6	6	7.86	7.86	RDA
DC02-870	349.6	355.6	6	6.05	6.05	RDA
DC02-870	397.6	403.6	6	6.67	6.67	RDX
DC02-870	403.6	409.6	6	4.04	4.04	RDX
DC02-870	469.6	475.6	6	4.29	4.29	RDX
DC02-872	254	260	6	4.85	4.85	RDX
DC02-875	379	385	6	4.75	4.75	RDXL
DC02-875	385	391	6	4.32	4.32	RDXL
DC02-875	391	397	6	6.80	6.80	RDXL
DC02-875	397	403	6	5.56	5.56	RDXL
DC02-878	156	162	6	12.08	12.08	RDX
DC02-878	162	168	6	15.03	15.03	RDX
DC02-878	168	174	6	8.83	8.83	RDX
DC02-879	183.05	189.05	6	4.95	4.95	GWK
DC02-881	101	107	6	5.67	5.67	RDX
DC02-881	107	113	6	8.31	8.31	RDX
DC02-881	161	167	6	4.77	4.77	RDX
DC02-882	103	109	6	5.12	5.12	RDA
DC02-882	109	115	6	11.21	10.49	RDA
DC02-882	115	121	6	5.07	4.53	RDA
DC02-882	169	175	6	4.40	4.40	RDX
DC02-882	175	181	6	11.03	11.03	RDX
DC02-883	20.37	26.37	6	4.53	4.53	RDX
DC02-885	257	263	6	4.22	4.22	RDXL
DC02-885	275	281	6	4.27	4.27	RDXL
DC02-885	281	287	6	4.96	4.96	RDXL
DC02-887	10.28	16.28	6	11.71	11.71	RDX
DC02-887	16.28	22.28	6	12.22	12.22	RDX
DC02-888	15.66	21.66	6	5.27	5.27	RDX
DC02-889	80.2	86.2	6	4.37	4.37	GWK
DC02-889	86.2	92.2	6	4.37	4.37	GWK
DC02-891	110.55	116.55	6	6.40	5.03	GWK
DC02-892	49.35	55.35	6	4.49	4.49	RDX
DC02-894	123.05	129.05	6	8.15	8.14	RDXL
DC02-900	24.15	30.15	6	11.29	10.69	SHL
DC02-908	166.8	172.8	6	5.98	5.98	GWK

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC02-908	178.8	184.8	6	7.92	7.92	GWK
DC02-910	57.1	63.1	6	4.35	4.35	RDXB
DC02-910	93.1	99.1	6	5.74	5.74	RDA
DC02-911	63.4	69.4	6	8.76	8.76	RDXB
DC02-911	69.4	75.4	6	6.30	6.30	RDXB
DC02-911	75.4	81.4	6	5.98	5.98	RDA
DC02-911	249.4	255.4	6	8.96	8.96	RDX
DC02-911	321.4	327.4	6	4.17	4.17	RDXL
DC02-913	20.75	26.75	6	4.54	4.54	RDA
DC02-913	38.75	44.75	6	9.14	9.14	RDA
DC02-913	44.75	50.75	6	6.91	6.91	RDA
DC02-913	116.75	122.75	6	4.73	4.73	RDX
DC02-913	122.75	128.75	6	5.89	5.89	RDX
DC02-914	325	331	6	11.16	6.56	RDXL
DC02-914	331	337	6	8.61	7.83	RDXL
DC02-914	337	343	6	4.03	4.03	RDXL
DC02-915	300.8	306.8	6	5.64	5.64	RDX
DC02-915	360.8	366.8	6	4.62	4.62	RDXL
DC02-916	313	319	6	4.44	4.44	GWK
DC02-917	33.75	39.75	6	4.74	4.74	RDXB
DC02-917	201.75	207.75	6	4.31	4.31	RDX
DC02-918	44.85	50.85	6	5.89	5.89	RDX
DC02-918	50.85	56.85	6	35.49	13.00	RDX
DC02-921	9.97	15.97	6	6.45	6.45	RDX
DC02-921	117.97	123.97	6	7.26	6.59	SHL
DC02-923	165	169.77	4.77	9.56	9.56	GWK
DC02-926	56.5	62.5	6	6.15	6.15	RDX
DC02-926	98.5	104.5	6	4.46	4.46	SHL
DC02-933	15.3	21.3	6	5.57	5.57	RDXL
DC02-934	20.4	26.4	6	6.15	6.15	RDX
DC02-935	14.5	20.5	6	4.57	4.57	RDXL
DC02-936	22.1	28.1	6	6.13	6.13	GWK
DC02-936	34.1	40.1	6	4.23	4.23	RDXL
DC02-937	106.2	112.2	6	7.64	7.64	RDXL
DC02-937	112.2	118.2	6	5.44	5.44	RDXL
DC02-937	130.2	136.2	6	13.32	9.94	GWK
DC02-937	142.2	148.2	6	4.14	4.14	RDXL
DC02-938	15.05	21.05	6	6.71	6.71	RDXL
DC02-939	21.05	27.05	6	7.15	7.15	RDX
DC02-939	27.05	33.05	6	5.02	5.02	RDX
DC02-941	125	131	6	12.57	12.57	GWK
DC02-941	131	137	6	5.58	5.58	MD
DC02-941	143	149	6	6.00	6.00	MD
DC02-941	155	161	6	10.05	10.05	MD
DC02-941	161	167	6	9.87	9.87	GWK
DC02-941	167	173	6	7.58	7.58	GWK
DC02-942	110.3	116.3	6	5.22	5.22	RDX
DC02-942	116.3	122.3	6	6.04	6.04	RDX
DC02-945	161.4	167.4	6	4.55	4.55	RDXL
DC02-947	68.1	74.1	6	8.69	8.69	RDX
DC02-947	74.1	80.1	6	6.57	6.57	RDX
DC02-948	90.34	96.34	6	4.41	4.41	RDX
DC02-948	96.34	102.34	6	10.07	10.07	RDX
DC02-948	108.34	114.34	6	5.28	5.28	GWK

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC02-948	114.34	120.34	6	8.74	8.74	GWK
DC02-949	90.58	96.58	6	4.82	4.82	RDX
DC02-949	96.58	102.58	6	5.58	5.58	RDX
DC02-960	165.42	171.42	6	8.06	8.06	RDX
DC02-960	171.42	177.42	6	4.74	4.74	RDX
DC02-960	231.42	237.42	6	7.30	7.30	RDXL
DC02-961	98	104	6	13.10	13.10	RDX
DC02-961	116	122	6	4.86	4.86	RDX
DC02-961	152	158	6	5.73	5.73	RDXL
DC02-961	182	188	6	4.49	4.49	RDXL
DC02-962	158.45	164.45	6	6.18	4.01	RDXL
DC02-963	121	127	6	7.89	7.89	RDX
DC02-963	139	145	6	4.67	4.67	GWK
DC02-965	66	72	6	4.30	4.30	RDXB
DC02-965	120	126	6	4.54	4.54	RDX
DC02-965	126	132	6	14.11	14.11	RDX
DC02-967	157.93	163.93	6	7.15	6.63	GWK
DC02-967	223.93	229.93	6	4.16	4.16	GWK
DC02-968	126	132	6	8.08	6.75	RDXL
DC02-970	72	78	6	4.89	4.89	RDX
DC02-970	84	90	6	4.31	4.31	RDX
DC02-971	47.6	53.6	6	6.91	6.91	RDX
DC02-971	132	138	6	4.62	4.62	RDXL
DC02-971	138	144	6	4.44	4.44	RDXL
DC02-972	237.35	243.35	6	5.66	5.66	RDXL
DC02-973	65	71	6	13.79	13.79	RDA
DC02-973	71	77	6	6.65	6.65	RDA
DC05-1013	39.2	45.2	6	5.99	5.99	RDA
DC05-1013	63.2	69.2	6	5.19	5.19	RDA
DC05-1013	81.2	87.2	6	6.44	6.44	RDA
DC05-1013	195.2	201.2	6	5.42	5.42	RDA
DC05-1013	201.2	207.2	6	13.22	13.22	RDA
DC05-1013	207.2	213.2	6	10.47	10.47	RDA
DC05-1013	219.2	225.2	6	4.51	4.51	RDA
DC05-1013	273.2	279.2	6	6.05	6.05	SHL
DC05-1013	279.2	285.2	6	35.93	8.64	SHL
DC05-1013	285.2	291.2	6	6.10	6.10	RDXB
DC05-1013	291.2	297.2	6	7.38	7.38	RDXB
DC05-1013	351.2	357.2	6	5.54	5.54	RDA
DC05-1013	357.2	363.2	6	11.69	11.69	RDA
DC05-1013	363.2	369.2	6	6.71	6.71	RDA
DC05-1015	171	177	6	4.21	4.21	RDXB
DC05-1015	177	183	6	9.81	9.81	RDXB
DC05-1015	183	189	6	11.76	11.76	MD
DC05-1016	118	124	6	6.70	6.70	GWK
DC05-1016	130	136	6	4.79	4.79	RDA
DC05-1016	136	142	6	4.67	4.67	RDA
DC05-1018	124.27	130.27	6	6.66	6.66	RDA
DC05-1018	130.27	136.27	6	7.92	7.92	RDA
DC05-1018	202.27	208.27	6	4.81	4.81	RDA
DC05-1018	244.27	250.27	6	4.45	4.45	RDX
DC05-1018	304.27	310.27	6	5.66	5.66	RDX
DC05-1018	310.27	316.27	6	4.43	4.43	RDX
DC05-1018	316.27	322.27	6	4.51	4.51	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC05-1018	322.27	328.27	6	6.17	6.17	RDX
DC05-1018	328.27	334.27	6	4.53	4.53	RDX
DC05-1018	334.27	340.27	6	5.69	5.69	RDX
DC05-1018	340.27	346.27	6	6.20	6.20	RDX
DC05-1018	346.27	352.27	6	7.98	7.98	RDX
DC05-1018	352.27	358.27	6	6.11	6.11	RDX
DC05-1018	358.27	364.27	6	5.13	5.13	RDX
DC05-1018	406.27	412.27	6	10.17	10.17	RDX
DC05-1018	412.27	418.27	6	10.07	10.07	RDX
DC05-1019	3.05	9.05	6	5.91	5.91	RDX
DC05-1019	21.05	27.05	6	4.43	4.43	GWK
DC05-1019	189.05	195.05	6	5.48	5.48	GWK
DC05-1019	261.05	267.05	6	5.47	5.47	RDX
DC05-1019	267.05	273.05	6	8.40	8.40	RDX
DC05-1020	3.66	9.66	6	15.30	15.30	RDX
DC05-1021	56.58	62.58	6	5.02	5.02	RDX
DC05-1021	338.58	344.58	6	7.23	4.37	RDXL
DC05-1022	32.9	38.9	6	4.39	4.39	GWK
DC05-1022	140.9	146.9	6	4.54	4.54	RDX
DC05-1022	152.9	158.9	6	4.69	4.69	RDX
DC05-1022	164.9	170.9	6	5.77	5.77	RDX
DC05-1022	182.9	188.9	6	4.31	4.31	RDX
DC05-1022	212.9	218.9	6	5.33	5.22	GWK
DC05-1022	218.9	224.9	6	5.41	5.29	GWK
DC05-1022	224.9	230.9	6	4.33	4.33	RDX
DC05-1022	290.9	296.9	6	4.82	4.82	RDX
DC05-1025	92.45	98.45	6	7.81	7.81	RDA
DC05-1027	231.96	237.96	6	5.87	5.87	MD
DC05-1028	164.4	170.4	6	7.56	7.56	RDX
DC05-1028	194.4	200.4	6	5.14	5.14	RDX
DC05-1029	195	201	6	4.88	4.88	RDX
DC05-1029	201	207	6	7.86	7.86	RDXB
DC05-1029	255	261	6	7.77	7.77	SHL
DC05-1030	44	50	6	4.72	4.72	RDX
DC05-1031	108.9	114.9	6	4.79	4.79	RDX
DC05-1031	192.9	198.9	6	5.82	5.82	RDX
DC05-1031	222.9	228.9	6	17.06	10.66	RDX
DC05-1031	228.9	234.9	6	23.28	14.03	RDX
DC05-1031	276.9	282.9	6	4.88	4.88	GWK
DC05-1032	291	297	6	5.16	5.16	RDX
DC05-1032	309	315	6	6.20	6.20	RDX
DC05-1032	315	321	6	4.79	4.79	RDX
DC05-1032	333	339	6	5.38	5.38	GWK
DC05-1033	282	288	6	6.86	6.86	RDXB
DC05-1033	294	300	6	10.32	10.32	RDXB
DC05-1033	300	306	6	8.14	8.14	RDXB
DC05-1033	306	312	6	7.42	7.42	RDXB
DC05-1033	312	318	6	21.35	21.35	RDXB
DC05-1033	318	324	6	11.40	11.40	RDXB
DC05-1033	324	330	6	6.82	6.82	RDXB
DC05-1033	336	342	6	4.83	4.83	GWK
DC05-1035	129.66	135.66	6	6.47	6.47	GWK
DC05-1035	195.66	201.66	6	6.42	6.42	RDX
DC05-1035	213.66	219.66	6	4.93	4.93	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC05-1035	225.66	231.66	6	4.69	4.69	RDX
DC05-1035	249.66	255.66	6	6.62	6.59	GWK
DC05-1035	255.66	261.66	6	9.91	9.80	GWK
DC05-1035	261.66	267.66	6	6.60	6.60	GWK
DC05-1036	152	158	6	4.29	4.29	RDA
DC05-1036	176	182	6	11.49	11.49	RDX
DC05-1037	48	54	6	4.94	4.94	RDA
DC05-1037	54	60	6	6.68	6.68	RDA
DC05-1037	210	216	6	4.78	4.78	RDX
DC05-1037	216	222	6	14.37	14.37	RDX
DC05-1037	222	222.8	0.8	12.20	12.20	GWK
DC05-1039	268.08	274.08	6	6.48	6.48	RDXL
DC05-1048	191.61	197.61	6	4.39	4.39	RDX
DC05-1048	209.61	212.14	2.53	4.25	4.25	RDA
DC05-1052	23.18	29.18	6	4.25	4.25	RDXB
DC05-1052	125.18	131.18	6	4.90	4.90	RDA
DC05-1052	173.18	179.18	6	5.16	5.16	RDXB
DC05-1052	179.18	185.18	6	4.36	4.36	RDXB
DC05-1052	353.18	359.18	6	15.17	15.17	RDX
DC05-1052	365.18	371.18	6	6.97	6.97	RDX
DC05-1052	413.18	419.18	6	6.56	6.56	RDX
DC05-1052	443.18	449.18	6	5.46	5.46	RDX
DC05-1053	202	208	6	12.77	11.67	RDXB
DC05-1053	214	220	6	6.93	6.93	RDXB
DC05-1053	220	226	6	8.58	8.58	RDXB
DC05-1053	226	232	6	6.34	6.34	RDXB
DC05-1053	232	238	6	12.46	12.46	RDXB
DC05-1053	238	244	6	8.16	8.16	RDXB
DC05-1053	244	250	6	7.09	7.09	RDXB
DC05-1053	298	304	6	9.18	9.18	RDA
DC05-1053	310	316	6	4.72	4.72	RDA
DC05-1053	340	346	6	6.44	6.44	RDA
DC05-1053	352	358	6	4.69	4.69	RDA
DC05-1054	166.88	172.88	6	4.41	4.41	RDXB
DC05-1054	172.88	178.88	6	4.24	4.24	RDXB
DC05-1054	184.88	190.88	6	5.96	5.96	RDXB
DC05-1054	190.88	196.88	6	5.61	5.61	RDXB
DC05-1054	196.88	202.88	6	4.38	4.38	RDXB
DC05-1054	202.88	208.88	6	4.18	4.18	RDXB
DC05-1054	208.88	214.88	6	5.40	5.40	RDXB
DC05-1054	214.88	220.88	6	4.39	4.39	RDXB
DC05-1054	274.88	280.88	6	6.03	6.03	RDXB
DC05-1054	286.88	292.88	6	6.87	6.87	RDXB
DC05-1054	292.88	298.88	6	6.17	6.17	RDXB
DC05-1054	298.88	304.88	6	5.68	5.68	RDXB
DC05-1054	304.88	310.88	6	5.58	5.58	RDA
DC05-1054	316.88	322.88	6	6.21	6.21	RDA
DC05-1054	334.88	340.88	6	4.45	4.45	RDA
DC05-1054	340.88	346.88	6	9.69	9.69	RDA
DC05-1054	352.88	358.88	6	4.03	4.03	RDA
DC05-1055	283	289	6	4.29	4.29	RDA
DC05-1055	307	313	6	4.25	4.25	RDA
DC05-1055	331	337	6	6.28	6.28	RDA
DC05-1057	222	228	6	7.73	7.73	RDXB

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC05-1057	324	330	6	4.81	4.81	RDA
DC05-1057	330	336	6	4.92	4.92	GWK
DC05-1057	336	342	6	6.05	6.05	GWK
DC05-1057	342	348	6	4.10	4.10	GWK
DC05-1057	354	360	6	4.94	4.94	GWK
DC05-1057	360	366	6	12.15	12.02	GWK
DC05-1057	366	372	6	5.84	5.71	GWK
DC05-1057	372	378	6	5.14	5.14	GWK
DC05-1058	30	36	6	6.95	6.95	RDX
DC05-1058	36	42	6	4.10	4.10	RDX
DC05-1058	42	48	6	6.32	6.32	RDX
DC05-1058	48	54	6	6.41	6.41	RDX
DC05-1058	66	72	6	7.54	7.54	RDX
DC05-1058	131.9	137.9	6	4.03	4.03	RDA
DC05-1058	155.9	161.9	6	5.52	5.52	SHL
DC05-1059	411	417	6	4.32	4.32	GWK
DC05-1060	72	78	6	5.95	5.95	RDX
DC05-1060	150	156	6	5.11	5.11	RDA
DC05-1060	174	180	6	5.39	5.39	RDA
DC05-1060	318	324	6	9.58	9.58	RDA
DC05-1060	324	330	6	4.58	4.58	RDA
DC05-1061	146.32	152.32	6	4.44	4.44	RDA
DC05-1061	218.32	224.32	6	6.51	6.51	RDA
DC05-1061	266.32	272.32	6	12.70	12.70	RDX
DC05-1061	290.32	296.32	6	4.04	4.04	RDX
DC05-1061	326.32	332.32	6	4.03	4.03	SHL
DC05-1061	416.32	422.32	6	14.92	12.94	RDA
DC05-1061	422.32	428.32	6	6.64	6.64	RDA
DC05-1061	428.32	434.32	6	6.28	6.28	RDA
DC05-1061	452.32	458.32	6	6.94	6.94	RDX
DC05-1061	458.32	464.32	6	10.62	10.54	RDX
DC05-1061	494.32	500.32	6	13.18	13.18	RDA
DC05-1061	506.32	512.32	6	6.52	6.52	RDA
DC05-1061	530.32	536.32	6	4.69	4.69	RDA
DC05-1061	536.32	542.32	6	5.48	5.48	RDA
DC05-1062	9.35	15.35	6	4.72	4.72	RDX
DC05-1062	15.35	21.35	6	7.81	7.81	RDX
DC05-1062	21.35	27.35	6	6.52	6.52	GWK
DC05-1062	81.35	87.35	6	16.36	11.52	GWK
DC05-1062	87.35	93.35	6	4.78	4.78	GWK
DC05-1062	261.35	267.35	6	6.92	6.92	RDA
DC05-1062	279.35	285.35	6	12.28	12.28	RDA
DC05-1062	285.35	291.35	6	5.68	5.68	RDA
DC05-1062	291.35	297.35	6	6.23	6.23	RDA
DC05-1062	297.35	303.35	6	5.54	5.54	RDA
DC05-1062	303.35	309.35	6	7.84	7.84	RDA
DC05-1062	309.35	315.35	6	4.64	4.64	RDA
DC05-1062	315.35	319.12	3.77	6.50	6.50	RDA
DC05-1063	6	12	6	5.73	5.73	RDA
DC05-1063	30	36	6	16.57	12.76	RDX
DC05-1063	36	42	6	11.43	10.59	RDX
DC05-1063	42	48	6	4.67	4.67	GWK
DC05-1063	174	180	6	12.60	12.60	RDX
DC05-1063	180	186	6	7.51	7.51	RDA

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC05-1063	186	192	6	6.19	6.19	RDX
DC05-1063	204	210	6	4.15	4.15	RDX
DC05-1063	210	216	6	7.70	7.70	RDA
DC05-1063	216	222	6	8.51	8.51	RDA
DC05-1063	228	234	6	7.66	7.66	GWK
DC05-1063	282	288	6	4.85	4.85	RDA
DC05-1063	294	300	6	4.24	4.24	RDA
DC05-1064	21.05	27.05	6	4.52	4.52	RDX
DC05-1064	27.05	33.05	6	5.87	5.87	RDX
DC05-1065	88.75	94.75	6	12.43	9.93	GWK
DC05-1065	94.75	100.75	6	7.15	7.15	MD
DC05-1065	130.75	136.75	6	12.98	12.98	GWK
DC05-1065	136.75	142.75	6	7.17	7.17	RDX
DC05-1065	142.75	148.75	6	14.97	14.97	RDX
DC05-1065	148.75	154.75	6	5.16	5.16	RDX
DC05-1065	154.75	160.75	6	4.68	4.68	RDX
DC05-1065	160.75	166.75	6	6.73	6.73	RDX
DC05-1065	172.75	178.75	6	8.81	8.81	RDX
DC05-1065	178.75	184.75	6	7.25	7.25	RDX
DC05-1065	238.75	244.75	6	5.00	5.00	RDX
DC05-1065	274.75	280.75	6	5.80	5.80	RDX
DC05-1065	280.75	286.75	6	6.99	6.99	RDX
DC05-1065	286.75	292.75	6	13.88	13.88	RDX
DC05-1065	292.75	298.75	6	14.52	14.52	RDX
DC05-1065	412.75	418.75	6	4.86	4.86	RDX
DC05-1065	430.75	436.75	6	6.97	6.97	RDX
DC05-1065	436.75	437.69	0.94	9.81	9.81	RDX
DC05-1067	306.1	312.1	6	4.89	4.89	RDXL
DC05-1067	312.1	318.1	6	4.83	4.83	RDXL
DC05-1067	318.1	324.1	6	6.73	6.73	RDXL
DC05-1067	324.1	330.1	6	4.52	4.52	RDXL
DC05-1067	336.1	342.1	6	5.22	5.22	RDXL
DC05-1067	342.1	348.1	6	4.78	4.78	RDXL
DC05-1067	348.1	354.1	6	4.40	4.40	RDXL
DC05-1067	354.1	360.1	6	4.15	4.15	RDXL
DC05-1067	366.1	372.1	6	4.85	4.85	RDXL
DC05-1067	372.1	378.1	6	4.14	4.14	RDXL
DC05-1068	132	138	6	7.52	7.52	RDXB
DC05-1068	138	144	6	4.52	4.52	SHL
DC05-1068	210	216	6	10.70	10.70	RDX
DC05-1069	24.1	30.1	6	5.73	5.73	RDA
DC05-1069	30.1	36.1	6	6.87	6.87	RDA
DC05-1069	174.1	180.1	6	4.46	4.46	RDX
DC05-1071	60	66	6	9.34	9.34	RDX
DC05-1071	66	72	6	7.26	7.26	RDX
DC05-1071	72	78	6	10.76	10.76	RDX
DC05-1072	72	78	6	4.45	4.45	RDX
DC05-1072	216	222	6	6.30	6.30	GWK
DC05-1073	8.37	14.37	6	6.19	6.19	RDX
DC05-1074	198	204	6	5.28	5.28	RDX
DC05-1075	92.67	98.67	6	5.22	5.22	RDX
DC05-1078	144.67	150.67	6	7.04	7.04	RDX
DC05-1079	109.4	115.4	6	4.66	4.66	RDF
DC05-1081	137.15	143.15	6	7.04	7.04	MD

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC05-1081	143.15	149.15	6	5.84	5.84	GWK
DC05-1081	149.15	155.15	6	12.48	11.40	MD
DC05-1081	155.15	161.15	6	15.50	14.10	MD
DC05-1081	167.15	173.15	6	8.25	8.25	GWK
DC05-1082	134.85	140.85	6	5.10	5.10	RDX
DC05-1087	78.58	84.58	6	4.77	4.77	RDXL
DC05-1087	84.58	90.58	6	7.68	7.68	RDXL
DC05-1088	56.12	62.12	6	7.23	7.23	RDA
DC05-1088	104.12	110.12	6	4.02	4.02	RDA
DC05-1088	122.12	128.12	6	5.26	5.26	RDA
DC05-1089	48.58	54.58	6	5.93	5.93	GWK
DC05-1089	84.58	90.58	6	4.89	4.89	GWK
DC05-1090	182.25	188.25	6	5.91	5.91	RDXB
DC05-1091	62	68	6	4.48	4.48	GWK
DC05-1094	81.57	87.57	6	6.87	6.87	GWK
DC05-1094	87.57	93.57	6	13.34	13.34	GWK
DC05-1095	33.05	39.05	6	6.28	6.28	GWK
DC05-1096	83.24	89.24	6	6.64	6.64	GWK
DC05-1096	131.24	132.28	1.04	4.08	4.08	GWK
DC05-1097	54.25	60.25	6	6.80	6.80	RDXL
DC05-1097	60.25	66.25	6	8.40	8.40	RDXL
DC05-1097	66.25	72.25	6	7.26	7.26	RDXL
DC05-1098	150.2	156.2	6	10.83	10.83	GWK
DC05-1098	174.2	180.2	6	6.73	6.73	GWK
DC05-1099	204	210	6	5.75	5.75	RDXB
DC05-1099	216	222	6	5.94	5.94	RDX
DC05-1099	270	276	6	4.58	4.58	GWK
DC05-1100	156	162	6	6.91	6.91	GWK
DC05-1101	135.75	141.75	6	4.68	4.68	RDA
DC05-1101	231.75	237.75	6	5.65	5.65	GWK
DC05-1101	273.75	279.75	6	4.33	4.33	GWK
DC05-1102	198	204	6	6.35	6.35	RDX
DC05-1102	222	228	6	5.66	5.66	GWK
DC05-1102	246	252	6	10.30	10.30	GWK
DC05-1102	252	258	6	6.84	6.84	GWK
DC05-1102	258	264	6	4.38	4.38	GWK
DC05-1102	270	276	6	4.18	4.18	SHL
DC05-1102	378	384	6	7.83	7.83	SHL
DC05-1102	396	402	6	4.88	4.88	GWK
DC05-1103	102	108	6	5.57	5.57	RDXB
DC05-1103	264	270	6	4.46	4.46	SHL
DC05-1104	58.37	64.37	6	4.41	4.41	RDXB
DC05-1104	64.37	70.37	6	14.12	14.12	RDXB
DC05-1104	142.37	148.37	6	5.02	5.02	GWK
DC05-1104	166.37	172.37	6	4.52	4.52	GWK
DC05-1104	172.37	178.37	6	7.21	7.21	RDX
DC05-1104	202.37	208.37	6	7.06	7.06	GWK
DC06-1114	199	205	6	4.56	4.56	RDA
DC06-1114	205	211	6	4.79	4.79	RDA
DC06-1114	211	217	6	5.30	5.30	RDA
DC06-1114	241	247	6	5.81	5.81	RDA
DC06-1114	247	253	6	5.53	5.53	RDA
DC06-1114	253	259	6	4.89	4.89	RDA
DC06-1114	265	271	6	4.28	4.28	RDA

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC06-1114	289	295	6	4.71	4.71	RDA
DC06-1115	232.1	238.1	6	6.39	6.39	RDXB
DC06-1115	250.1	256.1	6	15.00	13.61	GWK
DC06-1115	268.1	274.1	6	5.99	5.99	RDA
DC06-1115	274.1	280.1	6	9.10	9.10	RDX
DC06-1115	322.1	328.1	6	4.01	4.01	RDA
DC06-1115	328.1	334.1	6	4.41	4.41	RDA
DC06-1115	346.1	352.1	6	9.26	9.26	RDXB
DC06-1115	358.1	364.1	6	4.42	4.42	SHL
DC06-1115	364.1	370.1	6	5.21	5.21	SHL
DC06-1116	169	175	6	8.25	8.25	RDXB
DC06-1116	193	199	6	6.28	6.28	SHL
DC06-1116	199	205	6	9.78	8.38	RDXB
DC06-1116	205	211	6	5.74	5.74	RDXB
DC06-1116	217	223	6	5.73	5.73	RDXB
DC06-1116	223	229	6	4.05	4.05	RDXB
DC06-1116	259	265	6	4.19	4.19	RDXB
DC06-1118	258	264	6	6.23	6.23	SHL
DC06-1119	155	161	6	4.54	4.54	RDA
DC06-1119	221	227	6	4.69	4.69	RDA
DC06-1120	249.08	255.08	6	4.23	4.23	RDA
DC06-1120	255.08	261.08	6	4.24	4.24	RDA
DC06-1122	171.73	177.73	6	5.43	5.43	RDA
DC06-1122	177.73	183.73	6	5.55	5.55	SHL
DC06-1122	255.73	261.73	6	4.26	4.26	RDA
DC06-1125	249	255	6	5.07	5.07	RDA
DC06-1125	255	261	6	4.49	4.49	RDA
DC06-1126	222.89	228.89	6	4.56	4.56	RDA
DC06-1126	318.89	324.89	6	9.76	9.76	RDA
DC06-1126	348.89	354.89	6	5.27	5.27	RDA
DC06-1127	247.27	253.27	6	4.24	4.24	GWK
DC06-1127	505.27	511.27	6	10.76	10.76	GWK
DC06-1127	511.27	517.27	6	9.75	9.75	GWK
DC06-1127	517.27	523.27	6	11.05	11.05	GWK
DC06-1128	116.45	122.45	6	11.58	11.58	RDXB
DC06-1128	158.45	164.45	6	5.66	5.66	RDXB
DC06-1128	164.45	170.45	6	4.16	4.16	RDXB
DC06-1128	170.45	176.45	6	5.04	5.04	SHL
DC06-1128	248.45	254.45	6	4.43	4.43	SHL
DC06-1129	158.78	164.78	6	5.70	5.70	RDA
DC06-1129	290.78	296.78	6	9.44	9.44	RDX
DC06-1129	296.78	302.78	6	5.41	5.41	RDX
DC06-1129	500.78	506.78	6	8.75	8.75	GWK
DC06-1129	518.78	524.78	6	6.32	6.32	RDX
DC06-1130	186.23	192.23	6	4.39	4.39	RDXB
DC06-1130	270.23	276.23	6	4.75	4.75	RDA
DC06-1130	426.23	432.23	6	4.37	4.37	RDA
DC06-1130	450.23	456.23	6	4.43	4.43	RDA
DC06-1131	27.24	33.24	6	4.23	4.23	RDXB
DC06-1131	153.24	159.24	6	4.52	4.52	RDA
DC06-1131	459.24	465.24	6	13.43	13.43	RDXL
DC06-1131	543.24	549.24	6	34.30	7.86	RDX
DC06-1132	243.96	249.96	6	6.77	6.77	RDX
DC06-1132	249.96	255.96	6	6.32	6.32	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC06-1132	255.96	261.96	6	4.56	4.56	RDX
DC06-1132	333.96	339.96	6	5.61	5.61	GWK
DC06-1132	441.96	447.96	6	4.76	4.76	RDX
DC06-1132	597.96	603.96	6	5.84	5.84	RDX
DC06-1133	66	72	6	7.72	7.72	RDX
DC06-1133	138	144	6	4.20	4.20	RDXL
DC06-1133	150	156	6	5.82	5.82	RDXL
DC06-1133	156	162	6	4.50	4.50	RDXL
DC06-1133	162	168	6	12.32	12.32	RDXL
DC06-1133	168	174	6	11.63	8.63	SHL
DC06-1133	174	180	6	4.97	4.97	RDXL
DC06-1133	180	186	6	15.85	10.02	RDXL
DC06-1133	270	276	6	4.50	4.50	RDXL
DC06-1133	282	288	6	5.95	3.98	RDXL
DC06-1133	318	324	6	10.80	10.24	RDXL
DC06-1134	35.1	41.1	6	4.44	4.44	RDX
DC06-1134	185.1	191.1	6	4.74	4.74	RDXL
DC06-1134	191.1	197.1	6	4.95	4.95	RDXL
DC06-1137	402	408	6	5.04	5.04	RDXL
DC06-1138	284.84	290.84	6	5.47	5.47	RDX
DC06-1138	308.84	314.84	6	5.06	5.06	RDX
DC06-1138	464.84	470.84	6	4.04	4.04	RDX
DC06-1140	119	125	6	5.79	5.79	RDXB
DC06-1140	245	251	6	6.34	6.34	RDA
DC06-1141	149	155	6	4.16	4.16	RDA
DC06-1141	225	231	6	4.25	4.25	RDA
DC06-1141	321	327	6	5.18	5.18	RDA
DC06-1141	423	429	6	6.16	6.16	GWK
DC06-1142	123	129	6	9.37	9.37	GWK
DC06-1142	129	135	6	5.52	5.52	RDXB
DC06-1142	189	195	6	5.80	5.80	RDA
DC06-1142	195	201	6	5.34	5.34	RDA
DC06-1142	225	231	6	6.32	6.32	RDA
DC06-1143	38	44	6	4.15	4.15	RDX
DC06-1143	74	80	6	6.55	6.55	RDA
DC06-1143	80	86	6	5.70	5.70	RDA
DC06-1143	86	92	6	4.56	4.56	RDA
DC06-1144	143	149	6	4.96	4.96	RDXB
DC06-1144	149	155	6	10.44	10.44	RDXB
DC06-1144	239	245	6	7.22	7.22	RDX
DC06-1144	281	287	6	13.40	13.40	RDX
DC06-1144	341	347	6	6.23	6.23	RDX
DC06-1144	377	383	6	6.90	6.90	RDX
DC06-1144	395	401	6	6.84	6.84	RDX
DC06-1144	431	437	6	4.41	4.41	RDX
DC06-1144	455	461	6	10.98	10.98	RDX
DC06-1144	485	487.07	2.07	8.22	8.22	RDX
DC06-1145	108	114	6	4.80	4.80	RDA
DC06-1145	126	132	6	5.83	5.83	RDA
DC06-1145	156	162	6	6.05	6.05	RDX
DC06-1146	107.84	113.84	6	6.35	6.35	RDA
DC06-1146	143.84	149.84	6	6.02	6.02	RDX
DC06-1146	209.84	215.84	6	4.96	4.96	RDX
DC06-1146	215.84	221.84	6	4.57	4.57	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC06-1147	211.84	217.84	6	7.14	7.14	RDX
DC06-1147	253.84	259.84	6	4.75	4.75	RDX
DC06-1147	283.84	289.84	6	4.53	4.53	SHL
DC06-1147	289.84	295.84	6	4.64	4.64	RDX
DC06-1149	141	147	6	5.62	5.62	RDA
DC06-1149	171	177	6	5.04	5.04	SHL
DC06-1182	189	195	6	9.46	9.46	RDF
DC06-1182	195	201	6	8.59	8.59	RDF
DC06-1182	201	207	6	5.75	5.75	RDX
DC06-1182	231	237	6	4.59	4.59	RDX
DC06-1182	261	267	6	19.44	7.54	SHL
DC06-1182	309	315	6	4.59	3.74	RDF
DC06-1182	345	351	6	4.41	4.41	GWK
DC06-1183	96	102	6	6.72	6.72	SHL
DC06-1184	72	78	6	5.20	5.20	GWK
DC06-1184	78	84	6	4.77	4.77	GWK
DC06-1185	54.32	60.32	6	5.65	5.65	SHL
DC06-1185	198.32	204.32	6	7.67	7.67	MD
DC06-1185	228.32	234.32	6	10.39	8.39	GWK
DC06-1185	240.32	246.32	6	4.92	4.92	GWK
DC06-1185	246.32	252.32	6	9.19	9.19	GWK
DC06-1186	240	246	6	6.71	6.71	RDX
DC06-1186	252	258	6	4.95	4.95	RDX
DC06-1186	324	330	6	4.47	4.47	RDX
DC06-1186	342	348	6	4.42	4.42	SHL
DC06-1186	348	354	6	4.10	4.10	SHL
DC06-1187	17	23	6	7.29	7.29	GWK
DC06-1187	23	29	6	11.37	11.37	GWK
DC06-1187	29	35	6	6.89	6.89	GWK
DC06-1187	281	287	6	4.39	4.39	GWK
DC06-1187	287	293	6	7.24	7.24	GWK
DC06-1187	299	305	6	8.05	8.05	GWK
DC06-1187	317	323	6	6.33	6.33	RDX
DC06-1187	323	329	6	5.40	5.40	RDX
DC06-1187	329	335	6	5.76	5.76	RDX
DC06-1188	1.6	7.6	6	4.85	4.85	GWK
DC06-1188	7.6	13.6	6	11.03	11.03	GWK
DC06-1188	13.6	19.6	6	9.76	9.76	GWK
DC06-1188	43.6	49.6	6	13.24	12.54	GWK
DC06-1188	313.6	319.6	6	6.63	6.63	RDX
DC06-1188	325.6	331.6	6	4.17	4.17	RDX
DC06-1188	331.6	337.6	6	4.80	4.80	RDX
DC06-1188	337.6	343.6	6	6.23	6.23	RDX
DC06-1188	409.6	415.6	6	4.89	4.89	RDX
DC06-1189	60	66	6	4.89	4.89	RDX
DC06-1189	66	72	6	5.94	5.94	GWK
DC06-1189	72	78	6	5.80	5.80	GWK
DC06-1189	78	84	6	9.04	9.04	GWK
DC06-1190	30.09	36.09	6	8.60	8.60	GWK
DC06-1192	46	52	6	5.87	5.87	RDA
DC06-1192	538	544	6	4.09	4.09	RDXL
DC06-1194	6.71	12.71	6	4.59	4.59	RDA
DC06-1194	12.71	18.71	6	4.86	4.86	RDA
DC06-1194	156.71	162.71	6	5.64	5.64	RDA

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC06-1194	228.71	234.71	6	6.53	6.53	RDX
DC06-1194	474.71	480.71	6	6.75	6.75	RDXL
DC06-1194	486.71	492.71	6	4.57	4.57	RDXL
DC06-1196	285	291	6	8.16	8.16	RDA
DC06-1197	384	390	6	4.21	4.21	GWK
DC06-1198	24.97	30.97	6	4.86	4.86	SHL
DC06-1198	120.97	126.97	6	5.08	5.08	RDA
DC06-1199	116	122	6	4.66	4.66	RDXB
DC06-1199	134	140	6	4.11	4.11	RDA
DC06-1199	140	146	6	6.62	6.62	RDA
DC06-1222	84	90	6	4.78	4.78	MD
DC06-1222	96	102	6	10.38	10.38	RDX
DC06-1222	102	108	6	6.92	6.92	RDX
DC06-1222	210	216	6	4.84	4.84	RDX
DC06-1222	216	222	6	6.19	6.19	RDA
DC06-1222	240	246	6	7.37	7.37	RDA
DC06-1222	258	264	6	4.11	4.11	RDA
DC06-1224	139	145	6	4.38	4.38	RDXB
DC06-1224	145	151	6	7.67	7.67	RDXB
DC06-1224	169	175	6	5.84	5.84	RDA
DC06-1224	286	292	6	6.42	6.42	RDXB
DC06-1224	298	304	6	4.38	4.38	RDXB
DC06-1224	304	310	6	4.33	4.33	RDXB
DC06-1224	310	316	6	5.62	5.62	RDXB
DC06-1224	352	358	6	4.01	4.01	RDXB
DC06-1224	400	406	6	4.04	4.04	RDXB
DC06-1224	424	430	6	4.24	4.24	RDA
DC06-1224	430	436	6	6.23	6.23	RDA
DC06-1224	448	454	6	7.02	7.02	RDA
DC06-1225	30.8	36.8	6	4.01	4.01	RDA
DC06-1225	36.8	42.8	6	4.24	4.24	RDA
DC06-1225	66.8	72.8	6	7.74	7.74	RDA
DC06-1226	126.8	132.8	6	4.06	4.06	RDA
DC06-1226	132.8	138.8	6	4.34	4.34	RDA
DC06-1226	300.8	306.8	6	4.35	4.35	RDA
DC06-1227	74	80	6	4.87	4.87	MD
DC06-1228	38.02	44.02	6	4.22	4.22	RDX
DC06-1228	74.02	80.02	6	5.12	5.12	GWK
DC06-1228	128.02	134.02	6	8.36	8.36	GWK
DC06-1230	40.32	46.32	6	4.07	4.07	RDXL
DC06-1233	196	202	6	5.06	5.06	GWK
DC06-1233	202	208	6	6.66	6.66	GWK
DC06-1235	125.72	131.72	6	6.52	6.52	RDXL
DC06-1235	137.72	143.72	6	16.95	16.38	GWK
DC06-1235	143.72	149.72	6	5.62	5.57	GWK
DC06-1235	149.72	155.72	6	5.51	5.51	GWK
DC06-1236	220.84	226.84	6	11.51	11.51	GWK
DC06-1236	226.84	232.84	6	8.93	8.93	GWK
DC06-1237	71.97	77.97	6	7.28	7.28	RDA
DC06-1237	77.97	83.97	6	10.94	8.12	GWK
DC06-1237	113.97	119.97	6	4.64	4.64	GWK
DC06-1238	16.52	22.52	6	8.00	8.00	RDA
DC06-1239	28.48	34.48	6	4.68	4.68	RDX
DC06-1239	58.48	64.48	6	6.25	6.25	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC06-1239	64.48	70.48	6	5.60	5.60	RDX
DC06-1239	238.48	244.48	6	4.35	4.35	RDXL
DC06-1241	162	168	6	4.14	4.14	GWK
DC06-1241	324	330	6	4.79	4.79	GWK
DC06-1241	330	336	6	6.72	6.72	GWK
DC06-1242	288.24	294.24	6	10.95	10.95	GWK
DC06-1242	294.24	300.24	6	14.99	14.99	GWK
DC06-1242	300.24	306.24	6	9.85	9.85	GWK
DC06-1242	312.24	318.24	6	6.02	6.02	GWK
DC06-1243	97.7	103.7	6	5.40	5.40	RDX
DC06-1243	103.7	109.7	6	10.92	10.92	RDX
DC06-1243	109.7	115.7	6	7.75	7.75	RDX
DC06-1243	115.7	121.7	6	8.14	8.14	RDX
DC06-1243	121.7	127.7	6	8.78	8.78	RDX
DC06-1243	127.7	133.7	6	5.18	5.18	RDX
DC06-1243	157.7	163.7	6	4.20	4.20	RDX
DC06-1243	403.7	409.7	6	10.17	10.17	SHL
DC06-1244	181.32	187.32	6	6.59	6.59	MD
DC06-1245	101.33	107.33	6	7.95	7.95	RDX
DC06-1245	209.33	215.33	6	7.92	7.92	RDX
DC06-1245	215.33	221.33	6	14.53	14.53	RDX
DC06-1245	227.33	233.33	6	18.44	18.44	RDX
DC06-1246	24	30	6	5.12	5.12	RDX
DC06-1246	186	192	6	6.85	6.85	RDX
DC06-1250	296	302	6	6.24	6.24	RDX
DC06-1250	380	386	6	4.42	4.42	RDXB
DC06-1250	386	392	6	11.95	11.95	RDXB
DC06-1252	198	204	6	5.42	5.42	GWK
DC06-1252	269.1	275.1	6	8.46	8.46	RDXB
DC06-1253	326.54	332.54	6	4.28	4.28	RDXB
DC06-1255	140.46	146.46	6	8.55	8.55	GWK
DC06-1255	146.46	152.46	6	22.20	18.71	GWK
DC06-1255	152.46	158.46	6	26.78	21.59	GWK
DC06-1255	158.46	164.46	6	5.20	5.20	GWK
DC06-1255	212.46	218.46	6	21.10	21.10	GWK
DC06-1255	218.46	224.46	6	21.06	18.19	GWK
DC06-1255	224.46	230.46	6	5.31	4.68	GWK
DC06-1255	242.46	248.46	6	4.04	4.04	RDXB
DC06-1255	248.46	254.46	6	5.74	5.74	RDXB
DC06-1257	249.69	255.69	6	6.48	6.48	GWK
DC06-1259	223.82	229.82	6	5.26	5.26	RDX
DC06-1259	229.82	235.82	6	5.96	5.96	RDX
DC06-1259	337.82	343.82	6	4.05	4.05	RDX
DC06-1259	343.82	349.82	6	9.05	9.05	GWK
DC06-1259	349.82	355.82	6	9.25	8.78	GWK
DC06-1259	355.82	361.82	6	13.89	10.73	GWK
DC06-1260	201.07	205.44	4.37	4.05	4.05	GWK
DC06-1260	205.5	211.5	6	11.49	11.49	GWK
DC06-1260	283.5	289.5	6	4.89	4.89	RDX
DC06-1260	289.5	295.5	6	4.25	4.25	GWK
DC06-1263	258	264	6	9.07	9.07	GWK
DC06-1263	264	270	6	5.39	5.39	GWK
DC06-1263	534	540	6	4.06	4.06	GWK
DC06-1266	245.18	251.18	6	4.34	4.34	RDF

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC06-1266	281.18	287.18	6	5.39	5.39	GWK
DC06-1268	236.45	242.45	6	11.10	11.10	RDX
DC06-1269	121.03	127.03	6	4.29	4.29	RDX
DC06-1269	199.03	205.03	6	6.21	6.21	SHL
DC06-1270	6	12	6	8.34	4.73	GWK
DC06-1270	12	18	6	9.72	6.52	GWK
DC06-1270	48	54	6	6.33	6.33	RDX
DC06-1271	156	162	6	4.65	4.65	GWK
DC06-1273	163.53	169.53	6	9.15	9.15	GWK
DC06-1273	169.53	175.53	6	25.26	22.06	GWK
DC06-1274	306	312	6	4.17	4.17	GWK
DC06-1274	366	372	6	5.82	5.82	GWK
DC06-1274	372	378	6	6.55	6.55	GWK
DC06-1275	82.02	88.02	6	6.17	6.17	RDA
DC06-1275	94.02	100.02	6	5.68	5.68	RDA
DC06-1275	100.02	106.02	6	7.59	7.59	RDA
DC06-1275	232.02	238.02	6	4.68	4.68	GWK
DC06-1278	262.42	268.42	6	5.95	5.95	GWK
DC06-1278	268.42	274.42	6	4.76	4.76	IBX
DC06-1279	197.49	203.49	6	4.93	4.93	RDA
DC06-1280	189.94	195.94	6	4.15	4.15	RDA
DC06-1280	195.94	201.94	6	4.14	4.14	RDA
DC06-1280	231.94	237.94	6	8.29	8.29	GWK
DC06-1280	261.94	267.94	6	8.03	8.03	GWK
DC06-1280	267.94	273.94	6	4.30	4.30	GWK
DC06-1281	117.96	123.96	6	7.30	7.30	RDXB
DC06-1281	147.96	153.96	6	4.86	4.86	SHL
DC06-1281	159.96	165.96	6	5.04	5.04	RDXB
DC06-1281	309.96	315.96	6	5.41	5.41	RDX
DC06-1281	315.96	321.96	6	6.15	6.15	RDX
DC06-1281	369.96	375.96	6	4.43	4.43	MD
DC06-1282	93.63	99.63	6	5.04	5.04	RDX
DC06-1282	111.63	117.63	6	4.70	4.70	RDX
DC06-1282	387.63	393.63	6	5.49	5.49	RDX
DC06-1282	393.63	399.63	6	8.93	8.93	RDX
DC06-1282	405.63	411.63	6	4.31	4.31	RDX
DC06-1282	411.63	417.63	6	7.55	7.55	RDX
DC06-1282	417.63	423.63	6	4.89	4.89	RDX
DC06-1283	246	252	6	7.91	6.88	GWK
DC06-1283	252	258	6	10.47	8.40	GWK
DC06-1283	264	270	6	4.69	4.69	GWK
DC06-1283	312	318	6	5.36	5.36	RDA
DC06-1283	384	390	6	4.76	4.76	RDF
DC06-1284	116.08	122.08	6	8.36	8.36	GWK
DC06-1284	122.08	128.08	6	4.71	4.71	GWK
DC06-1284	302.08	308.08	6	15.00	14.65	GWK
DC06-1284	326.08	332.08	6	6.41	6.41	RDA
DC06-1284	338.08	344.08	6	4.05	4.05	GWK
DC06-1284	404.08	410.08	6	5.06	5.06	RDXL
DC06-1284	530.08	536.08	6	5.90	5.90	GWK
DC06-1285	252	258	6	4.59	4.59	GWK
DC06-1285	324	330	6	6.85	6.85	RDF
DC06-1285	342	348	6	4.12	4.12	RDF
DC06-1285	444	450	6	11.12	11.12	MD

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC06-1286	249.25	255.25	6	6.12	5.40	RDF
DC06-1286	363.25	369.25	6	5.03	5.03	RDF
DC06-1286	387.25	393.25	6	4.16	4.16	RDF
DC06-1286	459.25	465.25	6	5.39	5.39	GWK
DC06-1286	465.25	471.25	6	4.66	4.66	GWK
DC06-1287	68.9	74.9	6	7.59	7.59	GWK
DC06-1287	110.9	116.9	6	5.44	5.44	GWK
DC06-1287	122.9	128.9	6	5.04	5.04	RDX
DC06-1287	320.9	326.9	6	5.73	5.73	RDX
DC06-1287	428.9	434.9	6	9.16	9.16	GWK
DC06-1287	458.9	464.9	6	4.46	4.46	GWK
DC06-1288	276	282	6	17.54	14.69	GWK
DC06-1288	282	288	6	11.00	9.72	SHL
DC06-1288	342	348	6	5.52	5.52	RDF
DC06-1288	546	552	6	9.91	9.91	GWK
DC06-1289	65.93	71.93	6	4.25	4.25	RDXL
DC06-1290	27.24	33.24	6	6.16	6.16	RDXL
DC06-1324	348	354	6	8.46	8.46	SHL
DC06-1324	372	378	6	5.55	5.55	GWK
DC06-1324	516	522	6	6.98	6.98	RDX
DC06-1326	219	225	6	4.87	4.87	SHL
DC06-1326	441	447	6	13.10	11.62	GWK
DC06-1326	447	453	6	5.79	4.31	GWK
DC06-1328	78	84	6	5.28	5.28	RDF
DC06-1328	126	132	6	4.21	4.21	RDX
DC06-1330	187.2	193.2	6	4.73	4.73	RDXL
DC06-1330	233	237	4	8.63	8.63	SHL
DC06-1330	305	311	6	9.82	9.82	RDX
DC06-1330	329	335	6	6.35	6.35	GWK
DC06-1330	359	365	6	14.20	8.54	GWK
DC06-1332	208.82	214.82	6	4.37	4.37	SHL
DC06-1333	6	12	6	4.42	4.42	RDXB
DC06-1334	213.96	219.96	6	5.53	5.53	RDX
DC06-1335	105.96	111.96	6	4.13	4.13	GWK
DC06-1335	159.96	165.96	6	6.36	6.36	RDX
DC06-1336	81.96	87.96	6	4.01	4.01	SHL
DC06-1336	357.96	363.96	6	4.69	4.69	RDX
DC06-1337	164	170	6	5.65	5.65	RDX
DC06-1337	170	176	6	4.74	4.74	RDX
DC06-1337	200	206	6	4.06	4.06	RDX
DC06-1338	122.13	128.13	6	7.32	7.32	GWK
DC06-1338	230.13	236.13	6	4.55	4.55	SHL
DC06-1338	236.13	242.13	6	7.36	7.36	GWK
DC06-1338	254.13	260.13	6	5.10	5.10	GWK
DC06-1339	276.71	282.71	6	5.00	5.00	RDX
DC06-1340	129.66	135.66	6	4.42	4.42	RDXB
DC06-1340	177.66	183.66	6	6.78	6.78	GWK
DC06-1340	285.66	291.66	6	4.06	4.06	GWK
DC06-1341	114	120	6	4.23	4.23	GWK
DC06-1341	144	150	6	7.13	7.13	RDA
DC06-1341	156	162	6	4.79	4.79	RDA
DC06-1341	216	222	6	19.48	17.38	RDX
DC06-1341	234	240	6	9.32	9.32	GWK
DC06-1342	20.85	26.85	6	4.97	4.97	RDXB

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC06-1342	38.85	44.85	6	4.32	4.32	RDXB
DC06-1342	170.85	176.85	6	4.34	4.34	GWK
DC06-1342	212.85	218.85	6	4.40	4.40	RDA
DC06-1342	218.85	224.85	6	6.60	6.60	GWK
DC06-1342	260.85	266.85	6	4.26	4.26	RDX
DC06-1342	284.85	290.85	6	5.82	5.82	RDX
DC06-1342	290.85	296.85	6	17.06	17.06	RDX
DC06-1342	314.85	320.85	6	6.70	6.70	GWK
DC06-1342	368.85	374.85	6	4.71	4.71	SHL
DC06-1342	380.85	386.85	6	7.31	7.31	GWK
DC06-1342	398.85	404.85	6	4.20	4.20	GWK
DC06-1343	171.97	177.97	6	16.11	14.11	RDX
DC06-1343	177.97	183.97	6	12.98	12.95	RDX
DC06-1343	219.97	225.97	6	4.20	4.20	RDXB
DC06-1343	249.97	255.97	6	9.27	9.27	RDXB
DC06-1343	255.97	261.97	6	4.41	4.41	GWK
DC06-1344	348	354	6	11.73	11.73	GWK
DC06-1346	54	60	6	6.89	6.89	GWK
DC06-1346	60	66	6	7.85	7.85	CGL
DC06-1346	318	324	6	8.13	8.13	RDF
DC06-1347	123.84	129.84	6	10.23	10.23	RDX
DC06-1347	261.84	267.84	6	10.48	10.48	RDX
DC06-1347	291.84	297.84	6	5.36	5.36	GWK
DC06-1348	229.59	235.59	6	4.31	4.31	SHL
DC06-1348	283.59	289.59	6	6.23	6.23	SHL
DC06-1348	289.59	295.59	6	4.76	4.76	SHL
DC06-1348	379.59	385.59	6	4.24	4.24	SHL
DC06-1348	397.59	403.59	6	4.32	4.32	MD
DC06-1349	104.3	110.3	6	5.66	5.66	RDX
DC06-1349	146.3	152.3	6	4.64	3.76	RDXL
DC06-1349	308.3	314.3	6	5.19	5.19	SHL
DC06-1349	428.3	434.3	6	11.74	10.89	RDF
DC06-1349	434.3	440.3	6	6.64	6.64	RDF
DC06-1350	130.93	136.93	6	4.45	3.37	RDXL
DC06-1350	136.93	142.93	6	6.52	5.01	RDXL
DC06-1350	148.93	154.93	6	5.51	5.51	RDXL
DC06-1353	89.16	95.16	6	8.15	8.15	RDX
DC06-1354	76.23	82.23	6	4.44	4.44	GWK
DC06-1356	34.88	40.88	6	4.73	4.73	GWK
DC06-1356	82.88	88.88	6	4.39	4.39	RDXL
DC06-1356	136.88	142.88	6	4.30	4.30	MD
DC06-1357	70.36	76.36	6	11.19	11.19	RDXB
DC06-1357	124.36	130.36	6	4.19	4.19	GWK
DC06-1357	136.36	142.36	6	7.36	7.36	GWK
DC06-1358	160.88	166.88	6	8.66	8.66	GWK
DC06-1358	196.88	202.88	6	5.96	5.96	GWK
DC06-1358	202.88	208.88	6	7.27	7.27	GWK
DC06-1358	268.88	274.88	6	4.59	4.59	MD
DC06-1359	271.23	277.23	6	5.09	5.09	RDX
DC06-1360	157.62	163.62	6	7.39	7.39	GWK
DC06-1361	102	108	6	4.19	4.19	GWK
DC06-1363	192.7	198.7	6	8.80	8.80	SHL
DC06-1363	198.7	204.7	6	5.72	5.72	SHL
DC06-1363	204.7	210.7	6	13.32	13.32	SHL

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC06-1363	210.7	216.7	6	5.63	5.63	SHL
DC06-1363	234.7	240.7	6	8.13	8.13	GWK
DC06-1363	330.7	336.7	6	5.15	5.15	RDXB
DC06-1364	104.05	110.05	6	4.16	4.16	RDXB
DC06-1364	236.05	242.05	6	4.31	4.31	RDX
DC06-1364	296.05	302.05	6	4.35	4.35	RDX
DC06-1364	314.05	320.05	6	4.66	4.66	RDX
DC06-1366	150	156	6	21.52	20.52	RDX
DC06-1366	156	162	6	24.02	21.65	GWK
DC06-1367	234	240	6	4.01	4.01	GWK
DC06-1367	366	372	6	5.81	5.81	RDXL
DC06-1367	420	426	6	4.75	4.75	MD
DC06-1367	450	456	6	9.44	9.44	GWK
DC06-1367	456	462	6	17.96	11.89	RDF
DC06-1368	122.35	128.35	6	7.94	7.94	MD
DC06-1368	146.35	152.35	6	7.68	7.68	RDXB
DC06-1368	158.35	164.35	6	16.53	16.32	RDXB
DC06-1368	164.35	170.35	6	21.24	20.25	RDXB
DC06-1368	170.35	176.35	6	12.86	12.86	RDXB
DC06-1368	182.35	188.35	6	5.14	5.14	RDXB
DC06-1368	188.35	194.35	6	6.03	6.03	RDXB
DC06-1368	200.35	206.35	6	15.67	14.54	RDXB
DC06-1368	206.35	212.35	6	8.94	8.94	RDXB
DC06-1368	296.35	302.35	6	4.22	4.13	GWK
DC06-1368	302.35	308.35	6	17.03	16.37	GWK
DC06-1368	308.35	314.35	6	11.22	11.22	GWK
DC06-1369	136.42	142.42	6	7.67	7.67	GWK
DC06-1369	142.42	148.42	6	5.70	5.70	GWK
DC06-1370	63.35	69.35	6	6.39	6.39	RDX
DC06-1370	99.35	105.35	6	4.96	4.96	GWK
DC06-1371	61.49	67.49	6	13.20	13.20	RDX
DC06-1371	67.49	73.49	6	6.98	6.98	RDX
DC06-1371	115.49	121.49	6	4.36	4.36	RDA
DC06-1371	121.49	127.49	6	9.94	9.94	RDX
DC06-1371	163.49	169.49	6	5.73	5.73	RDX
DC06-1371	175.49	181.49	6	5.45	5.45	RDX
DC06-1371	253.49	259.49	6	4.79	4.79	RDX
DC06-1371	259.49	265.49	6	6.55	6.55	RDX
DC06-1371	271.49	277.49	6	45.62	13.82	SHL
DC06-1371	355.49	361.49	6	5.69	5.69	RDX
DC06-1371	361.49	367.49	6	8.93	8.93	RDX
DC06-1371	403.49	409.49	6	4.17	4.17	SHL
DC06-1372	45.15	51.15	6	5.22	5.22	GWK
DC06-1372	57.15	63.15	6	6.57	6.57	GWK
DC06-1372	237.15	243.15	6	5.96	5.96	GWK
DC06-1372	267.15	273.15	6	9.73	9.73	GWK
DC06-1373	384	390	6	4.63	4.63	RDA
DC06-1374	282	288	6	17.70	17.70	RDXB
DC06-1376	120	126	6	8.60	8.60	RDX
DC06-1376	150	156	6	4.92	4.92	GWK
DC06-1376	168	174	6	8.43	8.43	RDF
DC06-1376	192	198	6	5.81	5.81	GWK
DC06-1378	207.62	213.62	6	4.02	4.02	SHL
DC06-1378	255.62	261.62	6	6.11	6.11	RDXB

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC06-1379	351.96	357.96	6	4.57	4.57	RDXB
DC06-1379	357.96	363.96	6	15.20	11.53	GWK
DC06-1380	130.6	136.6	6	9.70	9.70	RDX
DC06-1380	262.6	268.6	6	4.45	4.45	RDA
DC06-1380	268.6	274.6	6	4.48	4.48	RDA
DC06-1381	351.1	357.1	6	6.22	6.22	RDX
DC06-1388	58	64	6	7.81	7.81	GWK
DC06-1388	64	70	6	14.25	14.25	SHL
DC06-1388	70	76	6	6.31	6.31	SHL
DC06-1388	76	82	6	4.08	4.08	SHL
DC06-1388	172	178	6	6.32	6.32	SHL
DC06-1388	214	220	6	7.43	7.43	RDX
DC06-1388	226	232	6	4.60	4.60	RDA
DC06-1388	262	268	6	11.65	10.22	SHL
DC06-1390	297.66	303.66	6	6.43	6.43	GWK
DC06-1390	303.66	309.66	6	5.57	5.57	GWK
DC06-1393	27.35	33.35	6	5.33	5.33	RDX
DC06-1393	135.35	141.35	6	8.30	8.30	RDX
DC06-1394	29	35	6	18.33	15.23	GWK
DC06-1394	245	251	6	5.01	5.01	SHL
DC06-1395	78.4	84.4	6	5.83	5.83	GWK
DC06-1395	216.4	222.4	6	4.56	4.56	GWK
DC06-1395	330.4	336.4	6	8.91	8.91	GWK
DC06-1395	384.4	390.4	6	7.15	4.75	RDF
DC06-1396	70	76	6	6.97	6.97	GWK
DC06-1396	112	114.5	2.5	12.05	12.05	GWK
DC06-1396	162.6	168.6	6	6.85	6.85	RDA
DC06-1396	198.6	204.6	6	5.14	5.14	RDX
DC06-1396	204.6	210.6	6	4.00	4.00	RDX
DC06-1396	216.6	222.6	6	5.18	5.18	RDX
DC06-1396	228.6	234.6	6	4.40	4.40	RDX
DC06-1396	234.6	240.6	6	6.07	6.07	RDX
DC06-1396	240.6	246.6	6	5.96	5.96	RDX
DC06-1399	249.6	255.6	6	4.98	4.98	RDX
DC06-1399	255.6	261.6	6	4.21	4.21	RDX
DC06-1399	369.6	375.6	6	4.67	4.67	RDF
DC06-1402	576.5	582.5	6	4.81	4.81	RDX
DC06-1402	600.5	606.5	6	4.94	4.94	RDX
DC06-1402	606.5	612.5	6	4.04	4.04	RDX
DC06-1405	308.63	314.63	6	40.11	26.62	SHL
DC06-1405	314.63	320.63	6	26.41	24.92	SHL
DC06-1406	152.5	158.5	6	4.00	4.00	RDA
DC06-1406	200.5	206.5	6	11.17	11.17	GWK
DC06-1406	350.5	356.5	6	9.18	9.18	GWK
DC06-1406	386.5	392.5	6	8.52	8.52	GWK
DC06-1406	392.5	398.5	6	28.42	16.18	GWK
DC06-1406	440.5	446.5	6	4.91	4.91	RDXB
DC06-1407	148.01	154.01	6	8.16	8.16	GWK
DC06-1407	154.01	160.01	6	11.94	11.94	GWK
DC06-1407	160.01	166.01	6	4.94	4.94	GWK
DC06-1407	208.01	214.01	6	4.15	4.15	GWK
DC06-1407	214.01	220.01	6	4.59	4.59	GWK
DC06-1407	262.01	268.01	6	5.81	5.81	GWK
DC06-1408	38.13	44.13	6	7.59	7.59	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC06-1408	44.13	50.13	6	4.63	4.63	RDX
DC06-1410	164	170	6	5.10	5.10	RDA
DC06-1411	60	66	6	4.04	4.04	RDXB
DC06-1411	66	72	6	5.49	5.49	RDX
DC06-1411	108	114	6	4.66	4.66	RDXB
DC06-1411	204	210	6	5.77	5.77	RDX
DC06-1411	354	360	6	4.00	4.00	RDX
DC06-1412	198.2	204.2	6	8.60	8.60	GWK
DC06-1412	246.2	252.2	6	4.77	4.77	GWK
DC06-1412	336.2	342.2	6	5.28	5.28	RDXB
DC06-1412	366.2	372.2	6	4.77	4.77	RDXB
DC06-1412	372.2	378.2	6	5.91	5.91	GWK
DC06-1412	390.2	396.2	6	6.30	6.30	GWK
DC06-1412	402.2	408.2	6	13.35	13.35	GWK
DC06-1412	408.2	414.2	6	6.96	6.96	GWK
DC06-1413	186	192	6	8.32	8.32	GWK
DC06-1413	276	282	6	4.20	4.20	RDX
DC06-1413	294	300	6	10.25	10.25	SLT
DC06-1414	78.1	84.1	6	5.45	5.45	RDA
DC06-1414	84.1	90.1	6	5.28	5.28	RDA
DC06-1415	292	298	6	5.31	5.31	RDXL
DC06-1417	298.32	304.32	6	5.00	5.00	RDX
DC06-1417	304.32	310.32	6	4.30	4.30	RDX
DC06-1418	130.57	136.57	6	6.60	6.60	RDX
DC06-1418	154.57	160.57	6	5.03	5.03	GWK
DC06-1418	196.57	202.57	6	6.97	6.97	RDX
DC06-1418	214.57	220.57	6	4.09	4.09	RDX
DC06-1418	238.57	244.57	6	5.01	5.01	RDX
DC06-1418	262.57	268.57	6	5.54	5.54	RDX
DC06-1418	388.57	394.57	6	4.41	4.41	RDX
DC06-1418	478.57	484.57	6	5.97	5.97	MD
DC06-1420	162.92	168.92	6	4.80	4.80	SHL
DC06-1422	194.44	200.44	6	4.18	4.18	GWK
DC06-1422	200.44	206.44	6	6.72	6.72	GWK
DC06-1424	48.1	54.1	6	4.41	4.41	RDXB
DC06-1424	198.1	204.1	6	5.06	4.37	MD
DC06-1424	204.1	210.1	6	15.99	15.21	RDX
DC06-1424	210.1	216.1	6	5.38	5.38	RDX
DC06-1424	222.1	228.1	6	4.43	4.43	RDX
DC06-1424	228.1	234.1	6	5.13	5.13	RDX
DC06-1424	234.1	240.1	6	8.06	8.06	RDX
DC06-1424	288.1	294.1	6	5.97	5.97	RDX
DC06-1426	142.54	148.54	6	5.18	5.18	MD
DC06-1426	160.54	166.54	6	4.72	4.72	GWK
DC06-1426	178.54	184.54	6	13.27	7.56	GWK
DC06-1426	184.54	190.54	6	6.16	6.16	MD
DC06-1426	202.54	208.54	6	9.70	9.70	GWK
DC06-1428	260.6	266.6	6	5.82	5.82	RDX
DC06-1429	195.56	201.56	6	4.95	4.95	RDXB
DC06-1429	249.56	255.56	6	6.04	6.04	RDX
DC06-1429	255.56	261.56	6	8.72	8.72	RDX
DC06-1429	261.56	267.56	6	9.16	9.16	RDX
DC06-1429	267.56	273.56	6	5.27	5.27	RDX
DC06-1429	291.56	297.56	6	7.73	7.73	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC06-1429	297.56	303.56	6	7.19	7.19	RDX
DC06-1430	79.99	85.99	6	6.59	6.59	GWK
DC06-1430	304.17	310.17	6	5.20	5.20	GWK
DC06-1430	481	487	6	5.62	5.62	RDX
DC06-1430	487	493	6	6.34	6.34	RDX
DC06-1431	200.72	206.72	6	4.31	4.31	RDX
DC06-1432	104.3	110.3	6	6.62	6.62	RDA
DC06-1432	266.3	272.3	6	5.72	5.72	MD
DC06-1433	24.32	30.32	6	4.37	4.37	RDX
DC06-1433	36.32	42.32	6	4.97	4.97	GWK
DC06-1433	216	222	6	6.13	6.13	RDX
DC06-1433	222	228	6	5.33	5.33	RDX
DC06-1435	78.71	84.71	6	4.00	4.00	RDXB
DC06-1435	360.71	366.71	6	9.97	9.97	RDX
DC06-1435	366.71	372.71	6	5.18	5.18	RDX
DC06-1438	243.08	249.08	6	6.70	6.70	RDX
DC06-1438	249.08	255.08	6	5.38	5.38	RDX
DC06-1439	64.17	70.17	6	5.74	5.74	RDA
DC07-1496	203.49	209.49	6	4.36	4.36	RDA
DC07-1496	299.49	305.49	6	4.08	4.08	RDXB
DC07-1496	365.49	371.49	6	9.79	9.79	RDA
DC07-1496	371.49	377.49	6	9.33	9.33	RDA
DC07-1498	447.75	453.75	6	6.48	6.48	RDXB
DC07-1498	453.75	459.75	6	11.42	11.42	RDXB
DC07-1498	477.75	483.75	6	5.73	5.73	RDXB
DC07-1499	285.53	291.53	6	7.07	7.07	RDXB
DC07-1499	291.53	297.53	6	6.79	6.79	RDXB
DC07-1499	375.53	381.53	6	4.72	4.72	RDA
DC07-1499	399.53	405.53	6	4.94	4.94	RDA
DC07-1499	405.53	411.53	6	6.40	6.40	RDA
DC07-1499	411.53	417.53	6	6.03	6.03	RDA
DC07-1499	447.53	453.53	6	4.22	4.22	RDXB
DC07-1499	483.53	489.53	6	5.80	5.80	RDXB
DC07-1499	489.53	495.53	6	6.52	6.52	SHL
DC07-1499	555.53	561.53	6	4.49	4.49	RDA
DC07-1510	61.26	67.26	6	6.07	6.07	RDX
DC07-1510	73.26	79.26	6	13.69	13.69	RDX
DC07-1510	79.26	85.26	6	6.12	6.12	RDX
DC07-1512	504	510	6	4.21	4.21	RDF
DC07-1512	594	600	6	5.01	5.01	GWK
DC07-1513	480	486	6	4.38	4.38	RDX
DC07-1513	534	540	6	9.80	9.80	GWK
DC07-1515	171.11	177.11	6	5.50	5.50	RDA
DC07-1515	237.11	243.11	6	4.77	4.77	SLT
DC07-1515	243.11	249.11	6	9.71	9.71	RDX
DC07-1515	255.11	261.11	6	9.59	9.59	RDX
DC07-1515	261.11	267.11	6	13.20	13.20	RDX
DC07-1515	267.11	273.11	6	7.57	7.57	RDX
DC07-1516	9	15	6	18.00	15.00	RDX
DC07-1516	45	51	6	4.17	4.17	GWK
DC07-1517	299.25	305.25	6	5.17	5.17	GWK
DC07-1518	25.85	31.85	6	5.06	5.06	RDXB
DC07-1518	271.85	277.85	6	4.29	4.29	RDX
DC07-1525	272.36	278.36	6	5.58	5.58	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC07-1525	344.36	350.36	6	6.84	6.84	GWK
DC07-1525	440.36	446.36	6	4.02	4.02	RDX
DC07-1525	470.36	476.36	6	9.70	9.70	RDX
DC07-1525	476.36	482.36	6	6.03	6.03	RDX
DC07-1526	186.65	192.65	6	5.81	5.81	RDA
DC07-1526	192.65	198.65	6	5.73	5.73	RDA
DC07-1526	222.65	228.65	6	5.67	5.67	RDXB
DC07-1526	228.65	234.65	6	6.61	6.61	RDXB
DC07-1526	291.25	297.25	6	5.78	5.78	RDXB
DC07-1526	309.25	315.25	6	4.33	4.33	RDXB
DC07-1526	399.25	405.25	6	7.57	7.57	RDA
DC07-1527	132.4	138.4	6	4.43	4.43	GWK
DC07-1527	426.4	432.4	6	5.05	5.05	RDA
DC07-1528	156	162	6	4.22	4.22	RDA
DC07-1528	288	294	6	4.31	4.31	GWK
DC07-1528	294	300	6	7.01	7.01	GWK
DC07-1529	285.9	291.9	6	5.42	5.42	RDX
DC07-1532	297.35	303.35	6	5.52	5.52	RDA
DC07-1532	309.35	315.35	6	4.48	4.48	RDA
DC07-1532	327.35	333.35	6	5.71	5.71	RDA
DC07-1532	339.35	345.35	6	4.25	4.25	RDA
DC07-1533	267.83	273.83	6	8.87	8.87	RDX
DC07-1535	270	276	6	4.85	4.85	RDA
DC07-1535	276	282	6	5.12	5.12	GWK
DC07-1535	282	288	6	4.27	4.27	RDA
DC07-1535	288	294	6	7.74	7.74	GWK
DC07-1536	260.53	266.53	6	4.07	4.07	GWK
DC07-1536	266.53	272.53	6	4.11	4.11	GWK
DC07-1537	291.22	297.22	6	5.21	5.21	SLT
DC07-1538	73	79	6	6.00	6.00	RDX
DC07-1540	210	216	6	6.41	6.41	RDX
DC07-1540	216	222	6	6.51	6.51	RDX
DC07-1540	228	234	6	4.23	4.23	RDX
DC07-1541	345	351	6	4.93	4.93	RDA
DC07-1541	351	357	6	8.72	8.72	RDA
DC07-1541	357	363	6	5.62	5.62	RDA
DC07-1541	489	495	6	7.97	7.97	RDX
DC07-1541	495	501	6	8.34	8.34	RDA
DC07-1541	501	507	6	8.23	8.23	RDA
DC07-1541	561	567	6	18.16	16.62	RDX
DC07-1541	567	573	6	20.10	19.20	RDX
DC07-1541	573	579	6	10.25	10.25	RDX
DC07-1541	579	585	6	7.76	7.76	RDX
DC07-1542	112.86	118.86	6	12.93	10.53	MD
DC07-1542	226.86	232.86	6	8.75	8.75	GWK
DC07-1542	232.86	238.86	6	15.16	13.70	GWK
DC07-1544	126	132	6	5.67	5.67	GWK
DC07-1544	312	318	6	13.42	13.42	RDXB
DC07-1544	318	324	6	8.27	8.27	RDXB
DC07-1544	390	396	6	6.27	6.27	GWK
DC07-1545	152.08	158.08	6	11.58	8.96	GWK
DC07-1547	113	119	6	5.33	5.33	GWK
DC07-1547	119	125	6	6.24	6.24	GWK
DC07-1548	125.49	131.49	6	4.96	4.96	RDA

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC07-1548	137.49	143.49	6	4.35	4.35	RDA
DC07-1548	143.49	149.49	6	4.82	4.82	RDA
DC07-1548	149.49	155.49	6	11.52	11.52	RDA
DC07-1548	221.49	227.49	6	17.66	17.66	RDA
DC07-1548	233.49	239.49	6	4.82	4.82	RDA
DC07-1548	245.49	251.49	6	7.13	7.13	RDA
DC07-1548	263.49	269.49	6	4.17	4.17	RDA
DC07-1549	191.8	197.8	6	6.60	6.60	RDA
DC07-1549	311.8	317.8	6	4.42	4.42	GWK
DC07-1549	479.8	485.8	6	4.02	4.02	RDXB
DC07-1549	485.8	491.8	6	4.38	4.38	RDXB
DC07-1550	109.62	115.62	6	4.48	4.48	GWK
DC07-1550	139.62	145.62	6	4.22	4.22	GWK
DC07-1550	145.62	151.62	6	22.03	15.21	MD
DC07-1550	163.62	169.62	6	8.79	8.79	MD
DC07-1550	169.62	175.62	6	8.11	8.11	GWK
DC07-1550	181.62	187.62	6	28.26	10.53	GWK
DC07-1550	187.62	193.62	6	16.79	10.53	GWK
DC07-1550	229.62	235.62	6	5.01	5.01	GWK
DC07-1550	295.62	301.62	6	8.04	6.46	GWK
DC07-1556	269.12	275.12	6	5.83	5.83	RDA
DC07-1556	305.12	311.12	6	4.26	4.26	RDA
DC07-1556	347.12	353.12	6	4.66	4.66	RDA
DC07-1556	353.12	359.12	6	6.16	6.16	RDA
DC07-1556	425.12	431.12	6	9.17	9.17	RDA
DC07-1556	431.12	437.12	6	4.82	4.82	RDA
DC07-1556	515.12	521.12	6	10.12	10.12	RDA
DC07-1556	521.12	527.12	6	10.50	10.50	RDA
DC07-1556	527.12	533.12	6	19.65	19.65	RDA
DC07-1556	533.12	539.12	6	15.92	15.92	RDA
DC07-1556	539.12	545.12	6	5.66	5.66	RDA
DC07-1556	551.12	557.12	6	4.42	4.42	RDA
DC07-1556	557.12	563.12	6	7.54	7.54	RDA
DC07-1556	563.12	569.12	6	19.46	19.09	RDA
DC07-1556	569.12	575.12	6	11.48	11.48	RDA
DC07-1556	593.12	599.12	6	8.64	8.64	RDA
DC07-1556	599.12	605.12	6	17.33	15.86	RDA
DC07-1556	605.12	611.12	6	4.33	4.33	RDA
DC07-1556	611.12	617.12	6	5.08	5.08	RDA
DC07-1557	81.97	87.97	6	4.91	4.91	GWK
DC07-1557	111.97	117.97	6	4.55	4.55	GWK
DC07-1557	117.97	123.97	6	6.05	6.05	GWK
DC07-1557	231.97	237.97	6	4.27	4.27	RDXL
DC07-1558	160.89	166.89	6	7.56	7.56	GWK
DC07-1558	166.89	172.89	6	12.54	9.34	GWK
DC07-1559	48	54	6	6.68	6.68	RDA
DC07-1559	54	60	6	5.49	5.49	RDA
DC07-1559	216	222	6	4.46	4.46	RDX
DC07-1559	360	366	6	4.81	4.81	RDX
DC07-1560	390	396	6	5.26	5.26	GWK
DC07-1562	139.54	145.54	6	7.56	7.56	GWK
DC07-1562	193.54	199.54	6	4.17	4.17	GWK
DC07-1564	156	162	6	4.89	4.89	RDXB
DC07-1564	168	174	6	4.72	4.72	RDXB

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC07-1564	354	360	6	4.49	4.49	RDX
DC07-1564	408	414	6	6.78	6.78	RDX
DC07-1564	414	420	6	6.70	6.70	RDX
DC07-1564	468	474	6	5.19	5.19	RDA
DC07-1564	474	480	6	10.00	10.00	RDA
DC07-1564	480	486	6	6.82	6.82	RDA
DC07-1564	486	492	6	10.07	10.07	RDA
DC07-1564	492	498	6	5.93	5.93	RDX
DC07-1564	564	570	6	22.52	16.82	GWK
DC07-1564	570	576	6	8.08	3.48	SLT
DC07-1564	630	636	6	17.55	12.19	SLT
DC07-1564	636	642	6	24.67	21.93	RDX
DC07-1564	642	648	6	6.18	6.18	RDX
DC07-1564	696	702	6	7.22	7.22	RDX
DC07-1566	6	12	6	7.24	7.24	RDXB
DC07-1566	12	18	6	6.37	6.37	GWK
DC07-1566	36	42	6	8.39	7.74	GWK
DC07-1566	42	48	6	5.99	5.82	RDA
DC07-1566	48	54	6	6.03	6.03	RDA
DC07-1566	54	60	6	6.57	6.57	RDA
DC07-1566	222	228	6	4.23	4.23	RDX
DC07-1567	384.99	390.99	6	11.77	11.77	GWK
DC07-1567	492.99	498.99	6	8.41	8.41	RDXB
DC07-1567	564.99	570.99	6	7.74	7.74	RDXB
DC07-1568	420	426	6	4.86	4.86	RDXB
DC07-1568	426	432	6	4.39	4.39	RDXB
DC07-1570	12	18	6	5.22	5.22	RDX
DC07-1573	12	18	6	7.03	7.03	RDX
DC07-1573	54	60	6	4.32	4.32	RDF
DC07-1573	138	144	6	5.70	5.70	RDF
DC07-1573	378	384	6	4.18	4.18	GWK
DC07-1574	229.62	235.62	6	10.33	8.72	MD
DC07-1575	40	46	6	8.05	8.05	RDX
DC07-1575	46	52	6	7.53	7.53	RDX
DC07-1575	76	82	6	5.48	5.48	RDA
DC07-1575	232	238	6	5.55	5.55	RDA
DC07-1575	274	280	6	5.58	5.58	RDA
DC07-1575	280	286	6	8.09	8.09	RDA
DC07-1575	286	292	6	8.18	8.18	RDA
DC07-1575	292	298	6	4.30	4.30	RDA
DC07-1575	310	316	6	4.45	4.45	RDA
DC07-1575	496	502	6	4.03	4.03	RDX
DC07-1576	423	429	6	5.41	5.41	GWK
DC07-1577	150	156	6	4.66	4.66	SLT
DC07-1577	162	168	6	6.34	6.34	GWK
DC07-1577	258	264	6	4.36	4.36	RDX
DC07-1577	270	276	6	4.90	4.90	RDX
DC07-1580	122.73	128.73	6	6.63	6.63	RDX
DC07-1580	128.73	134.73	6	5.97	5.97	RDX
DC07-1580	140.73	146.73	6	10.17	9.15	MD
DC07-1580	158.73	164.73	6	6.46	6.46	RDF
DC07-1580	200.73	206.73	6	10.26	10.26	RDF
DC07-1580	224.73	230.73	6	5.99	5.42	MD
DC07-1580	230.73	236.73	6	5.24	5.24	GWK

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC07-1582	168	174	6	4.17	4.17	GWK
DC07-1583	83	89	6	8.32	8.32	RDX
DC07-1583	293	299	6	4.29	4.29	GWK
DC07-1583	365	371	6	7.27	7.27	GWK
DC07-1584	222	228	6	5.31	5.31	GWK
DC07-1584	228	234	6	8.35	8.35	CGL
DC07-1584	432	438	6	4.42	4.42	GWK
DC07-1584	438	444	6	4.48	4.48	GWK
DC07-1585	179	185	6	4.61	4.61	GWK
DC07-1585	185	191	6	8.62	8.62	GWK
DC07-1585	245	251	6	11.10	11.10	MD
DC07-1585	293	299	6	4.34	4.34	GWK
DC07-1586	243.6	249.6	6	4.41	4.07	MD
DC07-1587	56.17	62.17	6	5.68	5.68	RDA
DC07-1588	81.5	87.5	6	4.44	4.44	MD
DC07-1589	158.25	164.25	6	4.73	4.73	GWK
DC07-1589	194.25	200.25	6	4.42	4.42	GWK
DC07-1589	284.25	290.25	6	5.71	5.71	GWK
DC07-1589	296.25	302.25	6	13.51	9.11	GWK
DC07-1589	302.25	308.25	6	6.30	6.30	RDXL
DC07-1589	332.25	338.25	6	4.15	4.15	RDXL
DC07-1589	338.25	344.25	6	4.72	4.72	RDXL
DC07-1589	344.25	350.25	6	5.53	5.53	RDXL
DC07-1589	350.25	356.25	6	7.59	7.59	RDXL
DC07-1589	428.25	434.25	6	5.33	5.33	GWK
DC07-1590	9.05	15.05	6	4.16	4.16	RDXB
DC07-1590	15.05	21.05	6	7.32	7.32	RDXB
DC07-1590	405.05	411.05	6	5.96	5.96	RDA
DC07-1591	162	168	6	4.97	4.97	RDX
DC07-1592	276	282	6	29.27	18.67	SHL
DC07-1592	312	318	6	4.55	4.55	SHL
DC07-1593	177	183	6	5.38	5.38	GWK
DC07-1593	207	213	6	15.27	11.77	GWK
DC07-1593	213	219	6	4.10	4.10	GWK
DC07-1593	303	309	6	21.39	15.25	RDX
DC07-1593	357	363	6	9.23	9.23	MD
DC07-1593	435	441	6	4.61	4.61	RDX
DC07-1593	477	483	6	9.58	8.35	GWK
DC07-1594	33.55	39.55	6	4.21	4.21	GWK
DC07-1595	18	24	6	11.52	9.06	GWK
DC07-1595	258	264	6	5.08	5.08	GWK
DC07-1595	264	270	6	16.02	16.02	GWK
DC07-1596	66	72	6	5.17	5.17	RDA
DC07-1596	84	90	6	4.02	4.02	RDA
DC07-1596	264	270	6	13.04	7.15	GWK
DC07-1597	78.1	84.1	6	6.25	6.25	RDXL
DC07-1598	40	46	6	4.71	4.71	RDX
DC07-1598	46	52	6	6.86	6.86	RDX
DC07-1598	52	58	6	8.51	8.51	RDX
DC07-1598	58	64	6	13.89	13.89	RDX
DC07-1598	64	70	6	10.37	10.37	RDX
DC07-1598	82	88	6	10.25	10.25	RDX
DC07-1598	184	190	6	5.71	5.71	RDA
DC07-1598	190	196	6	11.40	11.40	RDA

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC07-1598	196	202	6	11.85	11.85	RDA
DC07-1598	226	232	6	5.80	5.80	GWK
DC07-1599	81.5	87.5	6	6.96	6.96	RDA
DC07-1601	12	18	6	4.68	4.68	RDX
DC07-1601	90	96	6	7.92	7.92	RDX
DC07-1601	96	102	6	4.96	4.96	RDX
DC07-1601	102	108	6	6.10	6.10	RDX
DC07-1602	18	24	6	4.91	4.91	GWK
DC07-1602	156	162	6	4.24	4.24	GWK
DC07-1602	204	210	6	12.84	10.77	RDX
DC07-1602	234	240	6	4.03	4.03	RDX
DC07-1602	264	270	6	14.89	14.25	GWK
DC07-1603	216.04	222.04	6	13.59	11.61	GWK
DC07-1604	136.56	142.56	6	4.28	4.28	RDF
DC07-1604	154.56	160.56	6	5.35	5.35	GWK
DC07-1604	256.56	262.56	6	8.28	8.28	GWK
DC07-1606	72	78	6	4.01	4.01	GWK
DC07-1606	114	120	6	5.05	3.61	RDX
DC07-1606	138	144	6	12.83	12.83	RDX
DC07-1606	150	156	6	9.37	9.37	RDX
DC07-1606	162	168	6	4.83	4.83	RDX
DC07-1608	312	318	6	13.15	12.99	GWK
DC07-1608	318	324	6	10.64	10.57	GWK
DC07-1611	48	54	6	9.56	9.56	RDX
DC07-1611	54	60	6	4.72	4.72	RDX
DC07-1611	126	132	6	6.50	6.50	GWK
DC07-1611	228	234	6	8.51	8.51	RDX
DC07-1612	4	10	6	4.01	4.01	RDXB
DC07-1612	16	22	6	4.41	4.41	RDA
DC07-1612	94	100	6	4.30	4.30	RDA
DC07-1612	256	262	6	7.42	7.42	GWK
DC07-1612	304	310	6	4.55	4.55	RDX
DC07-1613	152.91	158.91	6	4.07	3.80	SLT
DC07-1613	158.91	164.91	6	10.33	10.00	GWK
DC07-1613	230.91	236.91	6	8.93	8.93	GWK
DC07-1613	272.91	278.91	6	4.56	4.56	GWK
DC07-1614	176.5	182.5	6	6.48	6.48	GWK
DC07-1615	447	453	6	7.66	7.66	GWK
DC07-1616	73.1	79.1	6	8.23	8.23	GWK
DC07-1617	254.74	260.74	6	5.47	5.47	GWK
DC07-1617	260.74	266.74	6	5.14	5.14	GWK
DC07-1617	284.74	290.74	6	4.24	4.24	GWK
DC07-1618	124	130	6	8.19	8.19	RDX
DC07-1618	136	142	6	4.21	4.21	RDF
DC07-1618	196	202	6	13.14	10.48	RDX
DC07-1618	304	310	6	10.58	10.21	GWK
DC07-1618	322	328	6	4.37	4.37	GWK
DC07-1619	7.83	13.83	6	4.34	4.34	RDX
DC07-1619	103.83	109.83	6	14.24	14.24	RDX
DC07-1619	109.83	115.83	6	14.39	13.59	RDX
DC07-1619	115.83	121.83	6	4.24	4.24	RDX
DC07-1619	187.83	193.83	6	4.41	4.41	RDXL
DC07-1621	78	84	6	7.43	7.43	GWK
DC07-1621	84	90	6	8.50	8.50	GWK

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC07-1621	150	156	6	5.57	5.57	GWK
DC07-1621	156	162	6	5.40	5.40	GWK
DC07-1621	198	204	6	6.22	6.22	GWK
DC07-1621	204	210	6	11.45	11.45	GWK
DC07-1621	210	216	6	9.09	9.09	GWK
DC07-1621	216	222	6	6.23	6.23	GWK
DC07-1621	324	330	6	4.64	4.64	SLT
DC07-1621	360	366	6	9.80	9.80	RDXB
DC07-1621	366	372	6	4.90	4.90	RDXB
DC07-1622	340.81	346.81	6	4.43	4.43	RDXL
DC07-1622	400.81	406.81	6	7.71	7.71	GWK
DC07-1624	354	360	6	14.07	11.81	GWK
DC07-1625	160	166	6	4.25	4.25	GWK
DC07-1626	420	426	6	7.37	7.37	GWK
DC07-1626	426	432	6	5.15	5.15	GWK
DC07-1626	468	471	3	7.37	7.37	RDF
DC07-1628	66	72	6	4.38	4.38	GWK
DC07-1628	72	78	6	4.86	4.86	GWK
DC07-1628	84	90	6	4.97	4.97	RDXL
DC07-1635	338.13	344.13	6	4.33	4.33	RDF
DC07-1635	362.13	368.13	6	4.71	4.71	RDXL
DC07-1635	392.13	398.13	6	6.01	6.01	RDA
DC07-1637	15.5	21.5	6	5.39	5.39	GWK
DC07-1637	219.5	225.5	6	5.88	5.88	RDXL
DC07-1637	231.5	237.5	6	9.88	9.83	GWK
DC07-1637	291.5	297.5	6	13.38	13.38	GWK
DC07-1637	303.5	309.5	6	4.35	4.35	GWK
DC95-162	87.4	93.4	6	5.92	5.92	GWK
DC95-162	93.4	99.4	6	6.08	6.08	GWK
DC95-162	213.4	219.4	6	4.36	4.36	RDA
DC95-163	43.2	49.2	6	4.06	4.06	RDX
DC95-163	73.2	79.2	6	5.45	5.45	RDA
DC95-164	60	66	6	4.33	4.33	RDA
DC95-165	134.7	140.7	6	4.87	4.87	SLT
DC95-165	140.7	141.8	1.1	10.60	10.60	RDX
DC95-166	119.8	125.8	6	6.11	6.11	RDA
DC95-166	149.8	155.8	6	5.28	5.28	RDXB
DC95-166	167.8	173.8	6	6.55	6.55	RDXB
DC95-166	203.8	209.8	6	4.50	4.50	RDXB
DC95-167	121.5	127.5	6	7.35	6.79	RDXB
DC95-167	145.5	151.5	6	16.78	16.78	RDXB
DC95-167	151.5	157.5	6	7.52	7.52	RDXB
DC95-168	108	114	6	4.92	4.92	RDX
DC95-168	258	264	6	4.01	4.01	SLT
DC95-169	118.2	124.2	6	4.65	4.65	GWK
DC95-169	124.2	130.2	6	5.83	5.83	GWK
DC95-169	136.2	142.2	6	6.63	6.63	GWK
DC95-169	148.2	154.2	6	5.00	5.00	GWK
DC95-169	154.2	160.2	6	5.94	5.94	GWK
DC95-169	238.2	244.2	6	4.62	4.62	RDA
DC95-169	280.2	286.2	6	4.85	4.85	MD
DC95-169	286.2	292.2	6	6.78	6.78	RDF
DC95-174	84	90	6	4.76	4.76	MD
DC95-175	171.4	177.4	6	5.34	5.34	MD

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC95-178	361.2	367.2	6	4.10	4.10	GWK
DC95-178	367.2	373.2	6	4.57	4.57	GWK
DC95-180	189	195	6	11.90	10.45	GWK
DC95-188	103.2	109.2	6	5.89	5.89	RDXB
DC95-188	121.2	127.2	6	11.49	8.93	GWK
DC95-190	45	51	6	4.13	4.13	RDX
DC95-191	15	21	6	4.47	4.47	RDXB
DC95-191	21	27	6	5.64	5.64	RDXB
DC95-191	99	105	6	5.57	5.57	RDXB
DC95-191	159	165	6	8.08	5.78	MD
DC95-193	204	210	6	7.04	7.04	GWK
DC95-193	228	234	6	4.66	4.66	RDA
DC96-194	39.31	45.31	6	8.21	8.21	RDX
DC96-194	207.31	213.31	6	4.23	4.23	SLT
DC96-194	231.31	237.31	6	8.05	8.05	SLT
DC96-194	237.31	243.31	6	4.52	4.52	SLT
DC96-194	243.31	249.31	6	7.99	7.99	SLT
DC96-195	150	156	6	4.18	4.18	RDX
DC96-195	192	198	6	6.62	6.62	RDX
DC96-195	204	210	6	7.56	7.56	RDX
DC96-195	246	252	6	5.85	5.85	MD
DC96-195	270	276	6	8.08	8.08	RDF
DC96-197	113	119	6	6.26	6.26	RDXB
DC96-197	180	186	6	5.67	5.67	ARG
DC96-197	204	210	6	4.98	4.98	GWK
DC96-197	246	252	6	4.15	4.15	RDX
DC96-197	264	266.2	2.2	6.10	6.10	GWK
DC96-198	27.55	33.55	6	7.20	7.20	GWK
DC96-198	33.55	39.55	6	7.59	7.59	GWK
DC96-198	39.55	45.55	6	5.68	5.68	GWK
DC96-198	69.55	75.55	6	7.80	7.80	RDX
DC96-198	93.55	99.55	6	4.23	4.23	GWK
DC96-198	243.55	248.8	5.25	6.64	6.64	GWK
DC96-199	54	60	6	8.17	8.17	RDF
DC96-203	150	156	6	4.31	4.31	RDF
DC96-206	43.52	49.52	6	4.67	4.67	RDX
DC96-206	97.52	103.52	6	5.22	5.22	RDX
DC96-207	141.2	147.2	6	4.53	4.53	RDF
DC96-209	192.72	198.72	6	4.54	4.54	RDA
DC96-209	228.72	234.72	6	7.61	7.61	SLT
DC96-211	94.5	100.5	6	8.60	8.60	SLT
DC96-212	120.71	126.71	6	4.10	4.10	GWK
DC96-213	229.05	235.05	6	4.60	4.60	ARG
DC96-213	265.05	271.05	6	4.07	4.07	GWK
DC96-214	228	234	6	6.79	5.55	RDF
DC96-215	32	38	6	4.24	4.24	SLT
DC96-215	38	44	6	5.91	5.91	RDX
DC96-215	50	56	6	8.40	8.40	RDX
DC96-216	216	222	6	4.27	4.27	RDF
DC96-216	258	264	6	4.72	4.72	GWK
DC96-217	38	44	6	6.84	6.84	RDA
DC96-217	50	56	6	6.87	6.87	RDA
DC96-217	56	62	6	4.50	4.50	RDA
DC96-218	126	132	6	4.05	4.05	GWK

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC96-218	210	216	6	4.73	4.73	RDX
DC96-219	54.4	60.4	6	8.98	8.98	GWK
DC96-219	60.4	66.4	6	6.17	6.17	RDA
DC96-219	66.4	72.4	6	5.04	5.04	RDX
DC96-219	144.4	150.4	6	4.46	4.46	GWK
DC96-219	162.4	168.4	6	5.15	5.15	GWK
DC96-219	174.4	180.4	6	8.43	8.43	GWK
DC96-220	253	259	6	14.20	4.12	RDF
DC96-220	259	264.6	5.6	5.57	5.57	GWK
DC96-223	189.5	195.5	6	7.87	6.55	RDF
DC96-223	207.5	213.5	6	4.39	4.39	GWK
DC96-224	0	6	6	5.61	5.61	RDX
DC96-224	6	12	6	5.61	5.61	RDX
DC96-224	234	240	6	5.45	5.45	GWK
DC96-226	140	146	6	4.47	4.47	RDX
DC96-226	254	260	6	4.50	4.50	MD
DC96-227	114.8	120.8	6	6.69	6.69	RDX
DC96-227	120.8	126.8	6	8.80	8.80	RDX
DC96-227	126.8	132.8	6	4.38	4.38	SLT
DC96-228	48	54	6	6.89	6.64	GWK
DC96-228	54	60	6	10.26	10.01	GWK
DC96-228	60	66	6	4.41	4.41	GWK
DC96-230	198	204	6	4.11	4.11	RDXB
DC96-230	210	216	6	4.52	4.52	RDXB
DC96-232	35.2	41.2	6	4.51	4.51	GWK
DC96-232	41.2	47.2	6	7.98	7.98	GWK
DC96-232	47.2	53.2	6	4.63	4.63	RDX
DC96-232	53.2	59.2	6	7.48	7.48	RDX
DC96-232	113.2	119.2	6	4.17	4.17	RDX
DC96-232	125.2	131.2	6	4.24	4.24	SLT
DC96-232	155.2	161.2	6	5.29	5.29	RDX
DC96-232	161.2	167.2	6	14.38	9.61	SLT
DC96-232	215.2	221.2	6	5.37	5.37	RDX
DC96-233	280	286	6	5.60	5.60	RDX
DC96-233	298	304	6	5.97	5.97	RDX
DC96-234	118.6	124.6	6	6.56	5.49	RDXL
DC96-234	178.6	184.6	6	6.60	6.60	GWK
DC96-234	220.6	226.6	6	4.00	4.00	GWK
DC96-235	15.1	21.1	6	4.29	4.29	RDX
DC96-235	27.1	33.1	6	4.18	4.18	RDX
DC96-235	231.1	237.1	6	4.60	4.60	RDA
DC96-237	86.45	92.45	6	6.20	6.20	RDX
DC96-237	92.45	98.45	6	5.11	5.11	RDX
DC96-237	98.45	104.45	6	5.32	5.32	RDX
DC96-237	200.45	206.45	6	8.61	8.61	RDX
DC96-237	272.45	278.45	6	6.23	6.23	GWK
DC96-238	162	168	6	4.29	4.29	RDA
DC96-240	232.2	238.2	6	7.34	7.34	GWK
DC96-241	103.5	109.5	6	4.00	4.00	RDX
DC96-241	145.5	151.5	6	5.31	5.31	RDA
DC96-241	163.5	169.5	6	4.40	4.40	GWK
DC96-241	181.5	187.5	6	6.86	6.86	RDX
DC96-241	187.5	193.5	6	8.69	8.69	RDX
DC96-241	193.5	199.5	6	4.65	4.65	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC96-241	229.5	235.5	6	4.57	4.57	GWK
DC96-241	241.5	247.5	6	7.68	7.68	RDXB
DC96-241	247.5	253.5	6	7.10	7.10	RDXB
DC96-241	259.5	265.5	6	4.38	4.38	RDXB
DC96-241	265.5	271.5	6	4.81	4.81	RDXB
DC96-243	78.1	84.1	6	5.36	5.36	RDX
DC96-243	198.1	204.1	6	16.68	16.68	RDX
DC96-243	270.1	276.1	6	6.92	6.92	RDX
DC96-243	282.1	288.1	6	4.59	4.59	ARG
DC96-244	51.55	57.55	6	9.88	7.38	RDX
DC96-244	57.55	63.55	6	4.16	3.43	RDX
DC96-244	75.55	81.55	6	31.38	20.99	RDXB
DC96-244	81.55	87.55	6	30.19	28.46	RDXB
DC96-244	87.55	93.55	6	22.73	22.23	RDXB
DC96-244	93.55	99.55	6	26.26	24.02	RDXB
DC96-244	111.55	117.55	6	8.51	8.51	RDX
DC96-244	159.55	165.55	6	6.69	6.69	RDX
DC96-244	165.55	171.55	6	35.00	26.10	RDA
DC96-244	171.55	177.55	6	17.39	14.22	RDA
DC96-244	177.55	183.55	6	15.51	11.14	RDA
DC96-245	80.2	86.2	6	10.76	10.76	RDF
DC96-247	222	228	6	7.08	7.08	RDF
DC96-247	234	240	6	4.30	4.30	SLT
DC96-247	240	246	6	6.31	6.31	RDF
DC96-248	138	144	6	4.58	4.58	RDX
DC96-248	204	210	6	4.98	4.98	GWK
DC96-249	81.4	87.4	6	6.07	6.07	GWK
DC96-249	165.4	171.4	6	16.53	14.44	RDXB
DC96-258	0.5	6.5	6	8.31	8.31	RDX
DC96-258	6.5	12.5	6	7.64	7.64	RDX
DC96-258	12.5	18.5	6	6.04	6.04	RDX
DC96-258	24.5	30.5	6	5.16	5.16	RDX
DC96-258	180.5	186.5	6	10.48	6.28	GWK
DC96-258	258.5	264.5	6	5.64	5.64	RDX
DC96-260	90	96	6	17.77	17.77	RDA
DC96-260	102	108	6	6.68	6.68	RDA
DC96-262	249	255	6	30.03	16.03	GWK
DC96-265	44.4	50.4	6	5.19	5.19	RDXB
DC96-265	50.4	56.4	6	4.52	4.52	RDXB
DC96-265	152.4	158.4	6	4.52	4.52	RDA
DC96-265	188.4	194.4	6	9.09	9.09	RDX
DC96-265	218.4	224.4	6	13.70	12.71	GWK
DC96-266	55.3	61.3	6	5.77	5.77	SLT
DC96-266	277.3	283.3	6	6.11	6.11	GWK
DC96-266	301.3	307.3	6	8.53	6.74	GWK
DC96-267	86	92	6	5.06	5.06	RDX
DC96-267	98	104	6	4.04	4.04	RDX
DC96-267	110	116	6	7.19	5.08	GWK
DC96-268	234	240	6	5.20	5.20	RDXB
DC96-269	182	188	6	7.21	7.21	RDX
DC96-269	188	194	6	9.18	9.18	RDXB
DC96-270B	150	156	6	7.62	7.62	RDX
DC96-274	43.3	49.3	6	4.53	4.53	RDXB
DC96-274	163.3	169.3	6	7.88	7.88	RDA

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC96-274	301.3	307.3	6	35.38	3.30	RDA
DC96-275	43.1	49.1	6	8.11	8.11	RDX
DC96-275	217.1	223.1	6	6.99	6.99	RDX
DC96-275	271.1	277.1	6	5.32	5.32	GWK
DC96-275	301.1	307.1	6	6.22	6.22	ARG
DC96-276	60	66	6	5.61	5.61	RDX
DC96-278	21.4	27.4	6	4.39	4.39	GWK
DC96-278	75.4	81.4	6	10.64	10.64	SLT
DC96-278	195.4	201.4	6	5.75	5.75	GWK
DC96-278	201.4	207.4	6	4.55	4.55	GWK
DC96-282	217	223	6	8.58	6.78	RDA
DC96-282	223	229	6	21.81	12.61	GWK
DC96-282	241	247	6	5.15	5.15	RDX
DC96-282	265	271	6	10.03	9.55	GWK
DC96-282	271	277	6	22.87	22.38	GWK
DC96-282	277	281.9	4.9	16.06	15.32	GWK
DC96-283	217	223	6	7.55	7.55	SLT
DC96-284	160.8	166.8	6	4.59	4.59	RDXB
DC96-284	208.8	214.8	6	4.15	4.15	RDXB
DC96-287	187.8	193.8	6	5.46	5.46	RDX
DC96-288	174.6	180.6	6	4.21	4.21	SLT
DC96-289	108.8	114.8	6	6.21	6.21	RDA
DC96-289	114.8	120.8	6	4.93	4.93	RDA
DC96-289	150.8	156.8	6	4.86	4.86	RDX
DC96-289	162.8	168.8	6	4.81	4.81	GWK
DC96-289	324.8	330.8	6	6.14	5.74	GWK
DC96-289	330.8	336.8	6	6.75	6.15	GWK
DC96-290	68.8	74.8	6	4.61	4.61	RDX
DC96-290	218.8	224.8	6	4.43	4.43	RDX
DC96-291	29	35	6	5.57	5.57	RDA
DC96-292	258	264	6	4.20	4.20	RDF
DC96-292	318	324	6	5.12	5.12	GWK
DC96-293	67.6	73.6	6	5.17	5.17	SLT
DC96-293	73.6	79.6	6	9.64	9.32	SLT
DC96-293	139.6	145.6	6	9.45	8.17	GWK
DC96-293	145.6	151.6	6	6.89	6.57	GWK
DC96-293	223.6	229.6	6	4.27	4.27	ARG
DC96-296	288	294	6	7.19	7.19	RDX
DC96-297	130	136	6	6.43	6.43	RDXB
DC96-299	96.4	102.4	6	11.89	11.89	RDA
DC96-299	102.4	108.4	6	5.58	5.58	RDA
DC96-299	144.4	150.4	6	10.29	10.29	RDX
DC96-299	264.4	270.4	6	5.83	5.83	RDX
DC96-299	282.4	288.4	6	4.21	4.21	RDX
DC96-300	127	133	6	4.55	4.55	RDA
DC96-300	199	205	6	5.62	5.62	RDX
DC96-303	156	162	6	6.30	6.30	RDXB
DC96-304A	48	54	6	4.22	4.22	RDA
DC96-304B	66	72	6	8.52	8.52	RDX
DC97-383	36	42	6	4.80	4.80	SHL
DC97-384	54	60	6	4.77	4.77	GWK
DC97-395	200.8	206.8	6	9.70	9.70	MD
DC97-397	12	18	6	12.05	12.05	RDA
DC97-397	18	24	6	14.63	14.63	RDA

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC97-397	24	30	6	14.78	14.78	RDA
DC97-397	60	66	6	4.46	4.46	RDA
DC97-398	96	102	6	5.38	5.38	RDX
DC97-398	108	114	6	5.80	5.80	RDA
DC97-400	84.3	90.3	6	14.10	8.61	MD
DC97-401	10.2	16.2	6	6.19	6.19	GWK
DC97-401	16.2	22.2	6	6.46	6.46	RDX
DC97-401	22.2	28.2	6	5.04	5.04	RDX
DC97-401	28.2	34.2	6	14.90	14.90	RDX
DC97-401	34.2	40.2	6	10.20	10.20	RDX
DC97-403	62.5	68.5	6	7.97	7.97	RDX
DC97-403	74.5	80.5	6	6.52	6.52	RDX
DC97-404	128.7	134.7	6	6.44	6.44	RDX
DC97-409	90	96	6	4.42	4.42	RDX
DC97-409	120	126	6	4.18	4.18	RDA
DC97-409	144	150	6	6.32	6.32	RDA
DC97-410	30	36	6	4.12	4.12	RDX
DC97-411	72	78	6	4.60	4.60	RDXB
DC97-411	120	126	6	4.83	4.83	GWK
DC97-411	126	132	6	9.28	9.28	RDX
DC97-411	132	138	6	4.71	4.71	RDX
DC97-411	138	144	6	8.66	8.66	RDX
DC97-411	144	150	6	5.38	5.38	RDX
DC97-411	234	240	6	4.20	4.20	RDA
DC97-411	246	250.24	4.24	13.93	13.93	RDA
DC97-412	54	60	6	5.38	5.38	RDX
DC97-412	198	204	6	7.53	7.53	GWK
DC97-412	264	270	6	8.42	8.42	RDX
DC97-414	108	114	6	6.59	6.59	GWK
DC97-415	235.4	241.4	6	4.10	4.10	GWK
DC97-417	114	120	6	5.33	5.33	GWK
DC97-417	120	126	6	10.19	8.39	GWK
DC97-417	162	168	6	5.47	5.47	GWK
DC97-418	60	66	6	6.38	6.38	GWK
DC97-418	240	246	6	5.45	5.45	RDX
DC97-419	114	120	6	12.59	12.25	GWK
DC97-419	198	204	6	10.11	9.81	GWK
DC97-419	246	252	6	4.23	4.23	RDX
DC97-420	60	66	6	8.93	8.40	SHL
DC97-420	66	72	6	12.35	12.35	GWK
DC97-420	150	156	6	6.56	6.56	GWK
DC97-421	72	78	6	4.57	4.57	RDX
DC97-421	78	84	6	6.54	6.54	RDX
DC97-421	114	120	6	7.47	7.47	RDF
DC97-421	186	192	6	5.02	5.02	RDF
DC97-421	192	198	6	16.17	10.45	RDF
DC97-421	198	204	6	11.84	6.12	GWK
DC97-421	240	246	6	6.92	6.92	RDXB
DC97-422	282	288	6	4.43	4.43	GWK
DC97-424	54	60	6	7.50	7.50	RDX
DC97-424	114	120	6	4.09	4.09	GWK
DC97-424	198	204	6	4.49	4.49	RDX
DC97-425	66	72	6	5.07	5.07	RDXB
DC97-425	108	114	6	5.78	5.78	RDXB

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC97-425	114	120	6	4.58	4.58	RDXB
DC97-425	132	138	6	5.64	5.64	RDXB
DC97-425	138	144	6	6.95	6.95	RDXB
DC97-425	198	204	6	5.40	5.40	RDXB
DC97-425	210	216	6	4.88	4.88	RDXB
DC97-425	216	222	6	4.98	4.98	RDXB
DC97-426	72	78	6	6.28	6.28	RDX
DC97-426	210	216	6	8.90	8.90	GWK
DC97-427	90	96	6	9.11	9.11	GWK
DC97-427	144	150	6	4.38	4.38	RDX
DC97-428	75.35	81.35	6	6.80	6.80	GWK
DC97-428	81.35	87.35	6	6.05	6.05	GWK
DC97-428	159.35	165.35	6	5.06	5.06	RDX
DC97-428	195.35	201.35	6	7.17	7.17	RDF
DC97-428	201.35	207.35	6	6.80	6.80	RDF
DC97-430	144	150	6	24.01	11.58	GWK
DC97-430	204	210	6	5.57	5.57	RDA
DC97-430	228	234	6	6.37	6.37	RDX
DC97-432	6	12	6	8.10	8.10	RDX
DC97-434	168	174	6	5.68	5.68	RDXL
DC97-434	180	186	6	4.11	4.11	RDXL
DC97-436	174	180	6	5.77	5.77	RDX
DC97-436	180	186	6	6.98	6.98	RDX
DC97-436	186	192	6	4.89	4.89	RDX
DC97-437	282	286.51	4.51	4.39	4.39	RDXB
DC97-442	104.9	110.9	6	7.17	7.17	RDX
DC97-445	186	192	6	5.52	5.52	RDX
DC97-445	192	198	6	7.05	7.05	SHL
DC97-447	222	228	6	4.19	4.19	RDX
DC97-448	222	228	6	32.04	20.89	RDXB
DC98-449	120	126	6	7.99	7.99	RDA
DC98-449	126	132	6	4.03	4.03	RDA
DC98-449	144	150	6	6.15	6.15	RDA
DC98-449	150	156	6	5.95	5.95	RDA
DC98-449	162	168	6	4.01	4.01	RDA
DC98-449	174	180	6	4.02	4.02	RDA
DC98-449	180	186	6	6.21	6.21	RDA
DC98-449	210	216	6	4.11	4.11	RDX
DC98-452	225	228.9	3.9	9.50	9.50	GWK
DC98-453	86.44	92.44	6	5.80	4.95	RDF
DC98-454	87	93	6	15.96	15.29	GWK
DC98-454	105	111	6	4.16	4.16	RDX
DC98-454	153	159	6	6.07	6.07	RDX
DC98-454	165	171	6	4.85	4.85	RDF
DC98-455	56.13	62.13	6	4.71	4.71	RDXB
DC98-455	158.13	164.13	6	7.62	7.62	GWK
DC98-455	182.13	188.13	6	4.17	4.17	GWK
DC98-455	194.13	200.13	6	5.84	5.84	GWK
DC98-455	212.13	218.13	6	9.78	9.78	GWK
DC98-455	218.13	224.13	6	7.66	7.66	RDA
DC98-455	248.13	254.13	6	4.93	4.93	RDA
DC98-456	48	54	6	6.16	6.16	GWK
DC98-456	102	108	6	10.14	8.61	GWK
DC98-456	240	246	6	8.03	8.03	RDF

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC98-458	146.13	152.13	6	21.86	8.89	GWK
DC98-458	224.13	230.13	6	4.65	4.65	MD
DC98-458	290.13	296.13	6	4.31	4.31	RDX
DC98-459	108	114	6	5.44	5.44	RDA
DC98-459	186	192	6	5.12	5.12	RDXB
DC98-459	192	198	6	13.44	13.44	RDXB
DC98-459	198	204	6	8.71	8.71	RDXB
DC98-459	204	210	6	15.67	15.67	RDXB
DC98-459	210	216	6	12.14	10.14	RDXB
DC98-460	162	168	6	4.92	4.92	GWK
DC98-460	168	174	6	4.44	4.44	GWK
DC98-460	192	198	6	4.04	4.04	GWK
DC98-460	204	210	6	9.15	9.15	GWK
DC98-460	210	216	6	13.49	13.49	GWK
DC98-460	228	234	6	5.66	5.66	GWK
DC98-462	126	132	6	4.12	4.12	RDF
DC98-462	150	156	6	5.59	5.59	GWK
DC98-462	198	204	6	7.44	7.44	GWK
DC98-462	210	216	6	8.46	8.46	GWK
DC98-463	75.05	81.05	6	5.88	5.88	SHL
DC98-463	105.05	111.05	6	5.45	5.45	SHL
DC98-463	183.05	189.05	6	8.48	8.48	RDA
DC98-463	189.05	195.05	6	4.20	4.20	GWK
DC98-463	219.05	225.05	6	8.13	8.13	GWK
DC98-463	225.05	231.05	6	4.97	4.97	GWK
DC98-463	255.05	261.05	6	6.57	6.57	RDX
DC98-463	291.05	297.05	6	4.13	4.13	RDX
DC98-463	333.05	339.05	6	8.96	8.96	GWK
DC98-463	339.05	345.05	6	11.89	11.89	RDX
DC98-463	345.05	351.05	6	20.96	20.96	RDX
DC98-463	375.05	381.05	6	4.50	4.50	SHL
DC98-465	102	108	6	4.27	4.27	GWK
DC98-467	96	102	6	9.77	9.77	RDX
DC98-467	150	156	6	5.22	5.22	RDX
DC98-467	228	234	6	9.36	7.29	RDX
DC98-467	234	240	6	4.48	4.48	RDF
DC98-468	218.13	224.13	6	4.99	4.99	RDX
DC98-468	272.13	278.13	6	5.91	5.91	GWK
DC98-468	278.13	284.13	6	4.61	4.61	GWK
DC98-468	296.13	302.13	6	4.29	3.63	RDXL
DC98-469	49.6	55.6	6	5.00	5.00	GWK
DC98-469	187.6	193.6	6	15.44	8.71	GWK
DC98-470	141	147	6	4.41	4.41	GWK
DC98-471	36	42	6	5.38	5.38	RDX
DC98-471	60	66	6	7.58	7.58	RDX
DC98-473	110.44	116.44	6	5.59	5.59	RDA
DC98-473	116.44	122.44	6	5.89	5.89	RDA
DC98-473	182.44	188.44	6	4.39	4.39	RDF
DC98-475	283.94	289.94	6	4.60	4.60	RDX
DC98-476	54	60	6	4.20	4.20	RDX
DC98-476	66	72	6	6.32	6.32	RDX
DC98-476	72	78	6	5.32	5.32	RDX
DC98-476	108	114	6	5.30	5.30	RDX
DC98-476	216	222	6	6.71	6.71	SLT

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC98-478	122.1	128.1	6	4.28	4.28	RDF
DC98-478	140.1	146.1	6	10.35	10.35	RDX
DC98-478	170.1	176.1	6	10.65	9.88	GWK
DC98-478	176.1	182.1	6	7.53	5.63	GWK
DC98-478	254.1	260.1	6	4.73	4.73	GWK
DC98-479	240	246	6	6.66	6.66	RDX
DC98-479	318	324	6	4.50	4.50	GWK
DC98-481	26.2	32.2	6	4.63	4.63	RDX
DC98-481	110.2	116.2	6	5.94	5.94	RDXB
DC98-481	158.2	164.2	6	4.96	4.96	RDXB
DC98-484	114	120	6	4.84	4.84	RDA
DC98-484	120	126	6	7.99	7.99	RDA
DC98-486	126	132	6	4.86	4.86	RDXB
DC98-486	138	144	6	7.23	7.23	RDXB
DC98-486	150	156	6	6.65	6.65	RDXB
DC98-486	198	204	6	13.04	13.04	RDX
DC98-486	252	258	6	4.07	4.07	RDX
DC98-486	276	282	6	10.90	10.90	RDX
DC98-487	138	144	6	4.23	4.23	RDXB
DC98-487	144	150	6	8.58	8.58	SHL
DC98-487	222	228	6	5.35	5.35	RDX
DC98-487	228	234	6	16.24	16.24	RDX
DC98-487	240	246	6	6.72	6.50	RDX
DC98-487	246	252	6	11.69	11.12	GWK
DC98-487	264	270	6	12.51	12.51	GWK
DC98-487	270	274.62	4.62	4.58	4.58	GWK
DC98-488	216	222	6	4.87	4.87	GWK
DC98-489	120	126	6	9.11	9.11	RDXB
DC98-489	126	132	6	9.01	9.01	RDXB
DC98-489	132	138	6	4.59	4.59	RDXB
DC98-489	138	144	6	4.47	4.47	RDXB
DC98-489	168	174	6	6.87	6.87	RDXB
DC98-489	174	180	6	7.73	7.73	RDXB
DC98-489	246	252	6	4.22	4.22	GWK
DC98-489	264	270	6	10.98	10.98	RDX
DC98-489	270	276	6	8.84	8.84	RDX
DC98-489	276	282	6	16.40	16.40	RDX
DC98-489	282	288	6	13.17	13.17	RDX
DC98-489	288	294	6	6.82	6.82	RDX
DC98-490	90	96	6	4.17	4.17	RDXB
DC98-490	120	126	6	4.08	4.08	RDXB
DC98-492	132	138	6	6.52	6.52	RDXB
DC98-492	138	144	6	5.11	5.11	RDXB
DC98-492	144	150	6	7.44	7.44	RDXB
DC98-492	150	156	6	9.60	9.60	RDXB
DC98-492	156	162	6	7.37	7.37	RDXB
DC98-492	162	168	6	6.76	6.76	RDXB
DC98-492	174	180	6	5.46	5.46	RDXB
DC98-492	198	204	6	5.96	5.96	RDXB
DC98-493	84	90	6	4.06	4.06	RDX
DC98-494	156.6	162.6	6	14.96	8.62	RDF
DC98-494	162.6	168.6	6	6.61	4.80	GWK
DC98-494	168.6	174.6	6	4.53	4.53	RDF
DC98-494	276.6	282.6	6	6.62	6.62	GWK

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC98-496	105.05	111.05	6	9.37	9.37	RDXB
DC98-496	111.05	117.05	6	8.53	8.53	RDXB
DC98-496	183.05	189.05	6	5.17	5.17	RDXB
DC98-497	134.74	140.74	6	4.96	4.96	RDXB
DC98-497	152.74	158.74	6	4.26	4.26	RDXB
DC98-497	188.74	194.74	6	9.89	9.89	RDX
DC98-498	78	84	6	4.08	4.08	RDXB
DC98-498	108	114	6	4.31	4.31	RDXB
DC98-498	138	144	6	9.65	9.65	RDXB
DC98-498	174	180	6	6.01	6.01	RDXB
DC98-500	132	138	6	4.95	4.95	RDX
DC98-501	192	198	6	5.43	5.43	RDXB
DC98-501	306	312	6	4.55	4.55	RDA
DC98-501	312	318	6	4.63	4.63	RDA
DC98-502	30	36	6	4.62	4.62	SHL
DC98-502	36	42	6	5.90	5.90	RDXB
DC98-502	60	66	6	6.70	6.70	RDX
DC98-502	66	72	6	14.60	14.60	RDX
DC98-502	102	108	6	5.75	5.75	RDX
DC98-502	240	246	6	7.29	7.29	RDX
DC98-502	246	252	6	15.59	15.59	RDX
DC98-502	270	276	6	5.74	5.74	RDX
DC98-502	276	282	6	55.07	24.67	RDX
DC98-502	282	288	6	6.77	6.77	SHL
DC98-503	56.8	62.8	6	16.25	13.43	GWK
DC98-503	122.8	128.8	6	4.26	4.26	RDXB
DC98-503	194.8	200.8	6	7.16	7.16	RDXB
DC98-503	200.8	206.8	6	17.36	10.85	RDXB
DC98-503	206.8	212.8	6	7.84	7.84	GWK
DC98-504	72	78	6	5.58	5.58	RDX
DC98-504	102	108	6	5.38	5.38	RDX
DC98-504	108	114	6	12.39	12.39	RDX
DC98-504	114	120	6	10.13	10.13	RDXB
DC98-504	186	192	6	5.81	5.81	RDXB
DC98-506	204	210	6	5.69	5.69	RDX
DC98-507	235.83	241.83	6	4.10	4.10	GWK
DC98-508	114	120	6	13.99	12.09	SHL
DC98-509	240	246	6	4.10	4.10	RDXB
DC98-511	64.72	70.72	6	4.61	4.61	GWK
DC98-511	316.72	322.72	6	14.43	12.07	SHL
DC98-512	84	90	6	5.39	5.39	RDXL
DC98-513	84	90	6	7.23	7.23	RDX
DC98-513	108	114	6	4.05	4.05	RDX
DC98-514	60	66	6	10.92	10.92	RDX
DC98-514	168	174	6	4.33	4.33	GWK
DC98-514	180	186	6	6.60	6.60	MD
DC98-514	312	318	6	11.00	11.00	MD
DC98-518	6	12	6	10.12	10.12	RDXB
DC98-518	90	96	6	6.70	6.70	RDA
DC98-518	102	108	6	5.83	5.83	RDA
DC98-518	270	276	6	10.97	10.97	RDX
DC98-519	134	140	6	10.57	10.57	RDA
DC98-519	200	206	6	4.23	4.23	RDX
DC98-519	260	266	6	16.82	16.82	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC98-519	266	272	6	11.42	11.42	RDX
DC98-519	272	278	6	6.84	6.84	RDX
DC98-519	278	284	6	4.83	4.83	RDX
DC98-520	240	246	6	5.60	5.60	MD
DC98-521	99.05	105.05	6	5.57	5.57	GWK
DC98-522	36	42	6	5.25	5.25	RDX
DC98-522	120	126	6	4.93	4.93	GWK
DC98-522	132	138	6	7.76	7.76	MD
DC98-522	192	198	6	7.74	7.74	RDF
DC98-522	198	204	6	21.70	15.58	RDX
DC98-522	234	240	6	5.01	5.01	RDX
DC98-522	240	246	6	5.42	5.42	RDX
DC98-522	282	288	6	5.59	5.59	RDX
DC98-522	300	306	6	5.99	5.99	RDX
DC98-523	0	6	6	9.41	9.41	RDA
DC98-523	6	12	6	10.27	10.27	RDA
DC98-523	108	114	6	10.84	10.84	RDA
DC98-525	54	60	6	4.56	4.56	GWK
DC98-527	42	48	6	12.75	12.75	RDX
DC98-528	42	48	6	5.00	5.00	RDXB
DC98-528	48	54	6	8.92	8.92	RDXB
DC98-528	54	60	6	6.41	6.41	RDXB
DC98-528	120	126	6	7.53	7.53	RDA
DC98-528	162	168	6	6.46	6.46	RDA
DC98-528	234	240	6	7.34	7.34	RDX
DC98-533	174	180	6	5.01	5.01	GWK
DC98-534	0	6	6	6.11	6.11	RDX
DC98-534	198	204	6	6.83	6.83	GWK
DC98-534	222	228	6	9.12	9.12	RDX
DC98-534	234	240	6	4.59	4.59	RDX
DC98-534	240	246	6	15.55	15.55	RDX
DC98-534	246	252	6	11.82	11.82	RDX
DC98-534	252	258	6	6.52	6.52	RDX
DC98-534	264	270	6	5.25	5.25	RDX
DC98-535	78	84	6	6.94	6.94	GWK
DC98-535	162	168	6	4.91	4.91	GWK
DC98-536	92.9	98.9	6	9.52	9.52	RDA
DC98-536	98.9	104.9	6	4.80	4.80	SHL
DC98-536	116.9	122.9	6	4.48	4.48	RDA
DC98-536	122.9	128.9	6	5.28	5.28	RDA
DC98-538	45.1	51.1	6	4.34	4.34	RDA
DC98-538	51.1	57.1	6	8.86	8.86	RDA
DC98-538	57.1	63.1	6	12.29	12.29	RDA
DC98-538	69.1	75.1	6	4.11	4.11	RDA
DC98-538	81.1	87.1	6	6.63	6.63	RDA
DC98-538	99.1	105.1	6	4.02	4.02	SHL
DC98-538	105.1	111.1	6	6.18	6.18	RDX
DC98-539	42	48	6	19.19	19.19	RDXB
DC98-539	48	54	6	4.22	4.22	RDF
DC98-539	84	90	6	4.74	4.74	SHL
DC98-539	90	96	6	7.08	7.08	RDA
DC98-539	96	102	6	7.51	7.51	RDA
DC98-539	102	108	6	5.04	5.04	GWK
DC98-539	150	156	6	11.75	11.75	GWK

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC98-539	156	162	6	6.72	6.72	GWK
DC98-539	180	186	6	5.14	5.14	RDA
DC98-539	186	192	6	5.51	5.51	RDA
DC98-539	204	210	6	4.01	4.01	RDA
DC98-541	30	36	6	16.06	7.35	GWK
DC98-541	216	222	6	7.40	7.40	RDX
DC98-541	222	228	6	6.73	6.73	RDX
DC98-542	270.1	276.1	6	4.54	4.54	RDX
DC98-542	276.1	282.1	6	6.37	6.37	RDX
DC98-543	144	150	6	8.71	8.71	RDX
DC98-543	150	156	6	5.18	5.18	RDX
DC98-543	162	168	6	6.80	6.80	RDA
DC98-543	168	174	6	6.04	6.04	RDA
DC98-543	180	186	6	4.72	4.72	RDA
DC98-543	186	192	6	7.52	7.52	RDA
DC98-543	240	246	6	6.59	6.59	RDX
DC99-545	140.74	146.74	6	6.23	6.23	RDA
DC99-545	176.74	182.74	6	4.90	4.90	RDA
DC99-546	72	78	6	9.02	9.02	RDXB
DC99-546	78	84	6	9.73	9.73	RDXB
DC99-546	102	108	6	5.54	5.54	RDXB
DC99-546	120	126	6	15.67	15.67	RDXB
DC99-546	258	264	6	4.03	4.03	RDX
DC99-546	288	294	6	5.63	5.63	RDX
DC99-546	294	300	6	6.95	6.95	RDX
DC99-546	300	306	6	19.55	19.55	SLT
DC99-547	101.6	107.6	6	7.70	7.70	RDA
DC99-547	161.6	167.6	6	5.94	5.94	RDX
DC99-547	167.6	173.6	6	8.02	8.02	RDX
DC99-548	72	78	6	4.32	4.32	RDX
DC99-548	210	216	6	8.85	8.85	RDX
DC99-548	258	264	6	11.89	11.89	RDX
DC99-549	96	102	6	8.26	8.26	RDX
DC99-549	150	156	6	11.00	11.00	RDX
DC99-549	162	168	6	13.90	13.90	RDX
DC99-550	36	42	6	5.19	5.19	RDA
DC99-550	48	54	6	4.22	4.22	RDA
DC99-550	78	84	6	5.88	5.88	RDA
DC99-550	90	96	6	4.75	4.75	RDA
DC99-550	156	162	6	4.79	4.79	GWK
DC99-550	174	180	6	5.57	5.57	RDXB
DC99-550	210	216	6	6.16	6.16	RDXB
DC99-550	234	240	6	5.83	5.83	RDXB
DC99-550	270	276	6	5.47	5.47	RDXB
DC99-550	276	282	6	4.52	4.52	RDXB
DC99-550	312	318	6	6.05	6.05	RDA
DC99-550	318	324	6	13.53	13.53	RDA
DC99-550	324	330	6	7.15	7.15	RDA
DC99-550	330	330.4	0.4	6.60	6.60	RDA
DC99-553	114	120	6	4.71	4.71	RDA
DC99-553	144	150	6	4.21	4.21	RDA
DC99-553	156	162	6	4.48	4.48	RDA
DC99-553	174	180	6	6.50	6.50	RDX
DC99-553	204	210	6	4.11	4.11	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC99-553	216	222	6	18.73	18.73	RDX
DC99-553	222	228	6	18.04	18.04	RDX
DC99-555	0	6	6	13.53	5.00	OVB
DC99-555	6	12	6	9.29	5.11	SLT
DC99-555	12	18	6	16.93	16.93	RDXB
DC99-555	18	24	6	14.75	14.75	RDXB
DC99-555	24	30	6	42.25	19.71	MD
DC99-555	48	54	6	8.74	8.74	RDX
DC99-555	54	60	6	8.44	8.44	RDX
DC99-555	60	66	6	5.68	5.68	RDX
DC99-555	66	72	6	7.69	7.69	RDX
DC99-555	72	78	6	7.26	7.26	RDX
DC99-555	132	138	6	7.09	7.09	RDA
DC99-555	138	144	6	6.18	6.18	RDA
DC99-555	144	150	6	4.03	4.03	RDX
DC99-555	150	156	6	6.50	6.50	SLT
DC99-555	228	234	6	10.74	10.74	RDXB
DC99-555	240	246	6	11.60	11.60	RDA
DC99-555	246	252	6	6.19	6.19	RDA
DC99-555	252	258	6	5.39	5.39	RDA
DC99-555	258	264	6	4.13	4.13	RDA
DC99-555	264	270	6	8.44	8.44	RDA
DC99-555	270	276	6	4.73	4.73	RDA
DC99-555	282	288	6	9.60	9.60	RDX
DC99-555	288	294	6	8.45	8.45	RDX
DC99-555	318	324	6	5.32	5.32	RDX
DC99-556	132	138	6	4.20	4.20	RDX
DC99-557	72	78	6	6.96	6.96	RDA
DC99-557	78	84	6	19.34	19.34	RDA
DC99-557	84	90	6	8.78	8.78	RDA
DC99-557	96	102	6	4.13	4.13	RDX
DC99-557	102	108	6	8.51	8.51	SLT
DC99-560	42	48	6	5.73	5.73	SLT
DC99-560	48	54	6	5.87	5.87	RDX
DC99-560	150	156	6	4.30	4.30	RDA
DC99-560	156	162	6	7.25	7.25	RDA
DC99-560	162	168	6	4.12	4.12	RDA
DC99-561	144	150	6	4.11	4.11	RDXL
DC99-561	150	156	6	4.32	4.32	RDXL
DC99-562	103.93	109.93	6	9.00	9.00	RDA
DC99-562	109.93	115.93	6	9.19	9.19	RDA
DC99-562	115.93	121.93	6	5.00	5.00	RDA
DC99-562	139.93	145.93	6	4.38	4.38	RDA
DC99-563	18	24	6	11.30	11.30	RDA
DC99-563	210	216	6	5.83	5.83	RDXL
DC99-563	216	222	6	7.89	7.89	RDXL
DC99-563	222	228	6	7.35	7.35	RDXL
DC99-564	48	54	6	4.03	4.03	RDA
DC99-564	240	246	6	5.68	5.68	RDX
DC99-565	26.16	32.16	6	4.25	4.25	RDXB
DC99-565	104.16	110.16	6	4.54	4.54	RDA
DC99-565	212.16	218.16	6	8.91	8.91	MD
DC99-565	218.16	224.16	6	8.11	8.11	MD
DC99-566	34.3	40.3	6	6.20	6.20	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DC99-566	40.3	46.3	6	5.29	5.29	RDX
DC99-566	58.3	64.3	6	7.22	7.22	MD
DC99-566	64.3	70.3	6	7.69	7.69	GWK
DC99-566	112.3	118.3	6	4.33	4.33	GWK
DC99-566	268.3	274.3	6	5.04	4.56	RDXL
DC99-567	222	228	6	7.12	7.12	RDX
DC99-567	228	234	6	5.77	5.77	RDX
DC99-567	240	246	6	7.06	7.06	RDX
DC99-568	96	102	6	5.72	5.72	RDA
DC99-568	126	132	6	5.07	5.07	SHL
DC99-568	132	138	6	4.52	4.52	RDA
DC99-568	216	222	6	7.12	7.12	RDX
DC99-569	6	12	6	4.21	4.21	RDX
DC99-569	12	18	6	4.80	4.80	RDX
DC99-569	150	156	6	4.69	4.69	RDXL
DC99-569	252	258	6	5.03	5.03	RDXL
DC99-569	258	264	6	6.16	6.16	GWK
DC99-571	42	48	6	4.56	4.56	RDX
DC99-571	54	60	6	4.57	4.57	RDX
DC99-571	114	120	6	6.91	6.91	RDX
DC99-573	257.8	263.8	6	4.60	4.60	RDA
DC99-573	311.8	317.8	6	9.22	9.22	RDA
DC99-573	341.8	347.8	6	5.39	5.39	RDA
DC99-573	347.8	353.8	6	6.80	6.80	RDA
DC99-573	359.8	365.8	6	4.82	4.82	RDA
DC99-577	528	534	6	5.88	5.88	GWK
DCR96-308	60	66	6	4.91	4.91	RDX
DCR96-309	48	54	6	15.02	13.16	GWK
DCR96-309	126.23	132.23	6	7.77	7.77	RDX
DCR96-311	6	12	6	5.83	5.83	RDX
DCR96-311	12	18	6	5.98	5.98	RDX
DCR96-311	42	48	6	6.17	6.17	RDX
DCR96-312	96	102	6	5.33	5.33	GWK
DCR96-314	28.57	34.57	6	10.94	10.94	RDX
DCR96-314	34.57	40.57	6	6.51	6.51	RDX
DCR96-321	30	36	6	5.65	5.65	RDF
DCR96-324	30	36	6	4.99	4.99	GWK
DCR96-324	78	84	6	8.03	8.03	RDX
DCR96-324	108	114	6	6.17	6.17	RDX
DCR96-324	156	162	6	4.39	4.39	RDX
DCR96-325	162	168	6	4.35	4.35	RDX
DCR96-325	204.17	210.17	6	4.11	4.11	RDX
DCR96-327	102	108	6	6.27	6.27	RDX
DCR96-327	108	114	6	5.34	5.34	RDX
DCR96-327	120	126	6	4.88	4.88	RDX
DCR96-327	138	144	6	7.31	7.31	RDX
DCR96-327	180	182.93	2.93	7.66	7.66	RDX
DCR96-328	30	36	6	8.60	7.74	RDX
DCR96-328	108	114	6	10.31	10.31	RDA
DCR96-328	132	138	6	4.43	4.43	RDA
DCR96-328	156	162	6	4.61	4.61	GWK
DCR96-328	180	182.93	2.93	12.22	12.22	GWK
DGT05-1017	116.45	122.45	6	4.54	4.54	GWK
DGT05-1017	152.45	158.45	6	4.70	4.70	GWK

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
DGT05-1017	362.45	368.45	6	4.49	4.49	GWK
DGT06-1158	440.21	446.21	6	9.75	9.75	GWK
DGT06-1158	446.21	452.21	6	20.25	20.25	RDXB
DR02-629	24.1	30.1	6	6.18	6.18	MD
DR02-630	19.72	25.72	6	5.81	5.81	GWK
DR02-668	30.1	36.1	6	5.50	3.12	OVB
DR02-752	27.05	33.05	6	12.66	8.61	RDX
DR02-753	74.39	79.25	4.86	4.66	4.66	SLT
DR02-785	59.34	65.34	6	4.00	4.00	RDX
DR02-786	9.05	15.05	6	6.76	6.76	RDXB
DR02-786	15.05	21.05	6	7.17	7.17	RDXB
DR02-792	66.1	72.1	6	4.12	4.12	NONE
DR02-796	21.05	27.05	6	5.13	5.13	RDX
DR02-796	27.05	33.05	6	4.78	4.78	RDX
DR02-806	16.57	22.57	6	4.77	4.77	RDX
DR02-807	72.5	76.2	3.7	7.04	7.04	GWK
DR02-816	60.77	66.77	6	6.62	6.62	MD
DR02-820	48.58	54.58	6	5.62	5.62	RD
DR02-820	60.58	66.58	6	5.84	5.84	RD
DR02-823	16.6	22.6	6	13.55	13.55	RDXB
DR02-823	22.6	28.6	6	8.59	8.59	RDXB
DR02-837	72.58	78.58	6	7.22	7.22	RDX
DR02-845	112.484	118.484	6	4.36	4.36	RDA
DR02-847	85.06	91.06	6	6.58	6.58	RDX
DR02-952	55.812	61.812	6	10.41	10.41	RDX
DR02-952	61.812	67.812	6	12.43	12.43	RDX
DR02-952	67.812	73.812	6	8.21	8.21	RDX
DR05-0996	37.62	43.62	6	13.55	13.55	RDX
DR05-0998	139.716	145.716	6	4.57	4.57	GWK
DR97-342	114	120	6	6.31	6.31	RDX
DR97-342	126	132	6	4.96	4.96	RDX
DR97-345	60	66	6	7.47	7.47	GWK
DR97-345	66	72	6	6.29	6.29	GWK
DR97-349	78	84	6	16.55	16.32	GWK
DR97-350	162	168	6	6.11	6.11	RDX
DR97-350	168	174	6	7.91	7.91	RDX
DR97-350	174	180	6	5.94	5.94	RDX
DR97-352	90	96	6	10.71	6.71	GWK
DR97-352	114	120	6	5.82	5.82	RDX
DR97-352	120	126	6	32.75	9.01	GWK
DR97-352	126	132	6	7.52	7.52	RDX
DR97-352	132	138	6	6.91	6.91	RDX
DR97-352	138	144	6	8.97	8.97	RDX
DR97-369	174	180	6	4.37	4.37	MD
DR97-369	180	186	6	4.58	4.58	RDX
DR97-379	84	90	6	4.09	4.09	GWK
DR97-382	96	102	6	4.30	4.30	GWK
DW02-831	34.67	40.67	6	5.93	5.93	GWK
LT-07	84	86	2	4.95	4.95	RDXB
LT-08A	24	30	6	4.95	4.95	RDX
LT-08A	30	36	6	4.17	4.17	RDX
LT-13	132	138	6	4.63	4.63	RDX
LT-13	138	144	6	4.80	4.80	RDX
LT-13	348	354	6	4.06	4.06	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
LT-13	384	390	6	4.71	4.71	RDX
LT-13	450	456	6	4.38	4.38	RDX
LT-14A	8	14	6	6.92	6.92	RDXB
LT-14E	56	62	6	4.74	4.74	RDX
LT-19A	30	36	6	4.21	4.21	RDXB
LT-26	234	240	6	5.95	5.95	RDX
LT-27B	48	54	6	4.33	4.33	RDX
LT-27B	66	72	6	4.08	4.08	RDX
LT-30	24	30	6	6.25	6.25	NONE
LT-30	90	96	6	4.27	4.27	NONE
LT-32	96	102	6	4.48	4.48	RDX
LT-32	114	120	6	7.33	7.33	RDX
LT01-61	48	54	6	4.68	4.68	RDX
LT96-29B-2	37	43	6	4.85	4.85	RD
LT96-29B-6	36	42	6	6.66	6.66	RD
LT97-03-1	6	10	4	10.64	10.64	SED
LT97-13A-1	36	42	6	5.34	5.34	RDX
LT97-13A1	54	60	6	6.02	6.02	RDX
LT97-13A1	60	66	6	7.74	7.74	RDX
LT97-13A19	6	12	6	6.45	6.45	RDX
LT97-13B-1	30	36	6	4.38	4.38	RDX
LT97-17B-6	108	114	6	4.29	4.29	RDX
LT97-17B-6	156	162	6	4.58	4.58	SED
LT97-17B24	18	24	6	9.32	9.32	RDX
LT97-17B24	24	30	6	5.65	5.65	RDX
LT97-24	0	6	6	6.39	6.39	SED
LT97-24	90	96	6	5.20	5.20	RDX
LT97-30-4	72	78	6	6.01	6.01	RDX
LT97-30-4	78	84	6	7.93	7.93	RDX
LT97-30-4	90	96	6	4.96	4.96	RDX
LT97-30-4	96	102	6	13.58	13.58	RDX
LT97-39-16	6	12	6	5.15	5.15	RDX
LT97-40B	30	36	6	11.13	11.13	RDX
LT97-40B	42	48	6	4.74	4.74	RDX
LT97-40B	48	54	6	7.22	7.22	RDX
LT97-41-1	90	96	6	8.31	8.31	SED
LT97-42-14	0	6	6	5.34	5.34	RDX
LT97-42-7	64	68	4	5.44	5.44	RDX
LT97-44-27	30	36	6	6.91	6.91	RDX
LT97-46-29	36	42	6	7.83	7.83	RDX
LT97-46-29	72	78	6	5.00	5.00	RDX
LT98-27-00	12	18	6	4.53	4.53	RDX
LT98-27-00	30	36	6	5.48	5.48	RDX
LT98-27-00	36	42	6	16.05	16.05	RDX
LT98-27-45	0	6	6	9.84	9.84	RDX
LT98-27-45	6	12	6	4.39	4.39	RDX
LT98-32-1	42	48	6	5.79	5.79	RDX
LT98-32-3	24	27	3	5.87	5.87	GWK
LT98-54-00	36	38	2	4.12	4.12	RDX
LT98-54-38	0	6	6	4.65	4.65	RDX
LT98-54184	18	24	6	9.34	9.34	RDX
LT98-55B-4	18	24	6	5.80	5.80	RDX
LT98-55B28	66	72	6	4.57	4.57	RDX
LT99-59-1	49	52	3	4.30	4.30	RDX

Hole ID	From	To	length	Au g/t	Capped Au g/t	Rock Type
QT-04	48	54	6	4.92	4.92	GRH
QT-04	54	60	6	5.55	5.55	GRH
QT99-11-7	84	90	6	8.12	8.12	GWK
RC-009	68	74	6	8.92	8.92	GWK
RC-009	74	80	6	10.91	10.91	GWK
RC-009	80	86	6	7.42	7.42	RHY
RC-013	6	12	6	4.37	4.37	GWK
RC-013	12	18	6	8.40	8.40	GWK
RC-014	42	48	6	4.25	4.25	GWK
RC-086	36	42	6	12.44	12.23	ARH
RC-087	42	48	6	7.05	4.79	GWK
RC-090	35	41	6	4.14	4.14	RHY
RC-091	30	36	6	16.63	12.55	GWK
RC-091	108	114	6	4.73	4.73	RHY
RC-100	4	10	6	7.61	7.61	GWK
RC-101	94	100	6	26.61	22.48	GWK
RC-103	48	54	6	6.77	6.77	GWK
RC-103	54	56	2	4.27	4.27	SLT
RC-113	68	74	6	8.64	8.64	RHY
ST-09	187	193	6	7.95	7.95	GRH
ST-09	193	199	6	4.34	4.34	GRH
ST-17C	18	24	6	4.28	4.28	NONE

APPENDIX C

2006 – 2007 QA/QC Report

NovaGold Resources Inc.

MEMO

To: Kevin Francis

From: Melissa Zack

Date: February 1, 2008

Subject: Donlin Creek 2006/2007 QAQC

A review was conducted of the 2006 and 2007 Donlin Creek QAQC Program which included blanks, standards and duplicate samples. Data used in this review was provided by Penny Hobby and Rich Harris of Barrick Gold Corp. in January 2007. An insertion rate of one in twenty for blanks, standards and duplicates was quoted in the report "Sampling Method and Approach (after the 43-101 report)"; however, this cannot be confirmed as results for non-control samples were not provided in the data package.

Blanks

Blank data was provided within the file QAQC_STD_070128_pch.csv. Two types of blank material were used during the program; a product (Blnk) from Anchorage Sand and Gravel for the beginning of the 2006 season, followed by granite chips (Blnk-Gr) purchased from Lowe's for later 2006 and 2007. Each blank type was analysed for Au by two different assay protocols, Au-AA23 and Au-AA25, and can be divided as follows:

Blnk Type	Total Count
Blnk AA23	36
Blnk AA25	5
Blnk-Gr AA23	1435
Blnk-Gr AA25	2083
Sum	**3559**

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According to an excerpt from the report "Sampling Method and Approach (after the 43-101 report)", acceptable limits for the blank material was typically <0.05ppm Au until late 2006, after which the limit was set at <0.02 ppm Au. For the purpose of this review "Late 2006" was interpreted to mean following a date of September 1, 2006. Blank performance was also reviewed in terms of a more typical upper limit of three times the detection limit in the element of interest.

At an upper limit of three times the detection limit for Au, a total of 246 blanks were found to exceed this, resulting in a failure rate of 6.9%. The detection limits for Au-AA23 and Au-AA25 are 0.005 ppm and 0.01ppm Au, respectively. However, when examining the blanks in terms of the cut-off values outlined by Barrick Gold Corp., the failure rate is more modest. By dividing the samples into two time frames, a total of 5 samples or 0.1% of blanks failed before September 2006 and 163 samples or 4.6% of blanks failed after September 2006. The data that was extracted from the Donlin Creek AcQuire database included comments regarding many of the failed blanks. Of the five failures before September 2006, only one contained a comment. Of the 163 blanks that failed following September 2006, seventy-five had attached comments, leaving eighty-eight without. Examples of such comments are as follows:

'High Au value for Blank; but within acceptable limits'
'Failed QAQC for Au by a high margin on original certificate'
'Blank slightly out of range'
'Failed QAQC slightly for Blank-Granite, probably due to cross over contamination from surrounding samples.'

Although these comments do not always explain why the sample was not sent for re-assay, it at least indicates that the failure was noted, and was consciously ignored. A definition of "acceptable limits" as mentioned in the comments above would be helpful in reducing the ambiguity of some of these statements. However, there are still a total of ninety-two blanks (just over 50% of all blank failures), that have no comment and have not been accounted for. The maximum failures were found to be 0.135ppm, 0.438ppm and 0.64ppm for Blnk-AA23, Blnk-Gr AA23 and Blnk-Gr AA25 respectively, and their significance in terms of ore-grade cut-off should be determined. A number of the blank failures could be a result of smear from proceeding higher-grade samples; however, this could not be investigated as proceeding assay sample data was not provided as part of the data package.

Charts depicting each blank type, further dived by assay protocol and differing detection limits can be seen in Appendix I. Failure lines can be seen at 0.05ppm Au, 0.02ppm Au and three times the detection limit (0.015ppm for Au-AA23 and 0.03ppm for Au-AA25). Please note that these charts would be altered slightly if further separated into the two date ranges of before and after September 1, 2006.

Duplicates

Duplicate data was obtained from the file QAQC_DUP_070128_pch.csv. A total of 3885 duplicate pairs obtained from coarse reject material were taken at Donlin Creek during the 2006 and 2007 programs. Successful or "passing" duplicates were identified by calculating the Absolute Relative Difference (ARD):

$$ARD = \frac{|A-B|}{0.5*(A+B)}$$

For coarse reject duplicates, the following criteria (outlined in 'Assay Quality Assurance-Quality Control Program for Drilling Projects at the Pre-Feasibility to Feasibility Report Level 5th Edition', S. Long, 2007) must be met: ARD must be <0.2 for 90% of the pairs, and if the Pair Mean <15(DET), the Pair Difference must be ≤ 3(DET) to pass. Detection limits are 0.005ppm and 0.01ppm Au for Au-AA23 and Au-AA25 respectively, and results below detection limit were assigned a value equal to half the detection limit, to avoid the ARD equation becoming undefined. After dividing the duplicate pairs by assay method, and applying the ARD equation, the pairs were found to be 93% passing.

Method	DET	Total Pairs	Pairs Passing	% Passing
Au-AA23	0.005	1474	1371	93.0
Au-AA25	0.01	2411	2249	93.3

Charts were plotted to show the duplicate pairs and their Pass/Fail lines. The max value of each pair was plotted against the min value in order to reduce bias and plot above the y=x line. Ideally, all Donlin Creek duplicates should plot between the y=x line and the 20% Pass/Fail (P/F) line. Pass/Fail lines cross the y-axis at three times the detection limit of the element of interest. Duplicate performance is also represented as an ARD (or AVRD: Absolute Value of the Relative Difference) vs. Cumulative Frequency (in Percentile) plot. Outliers in the duplicate pairs can be visually identified as points that fall to the right of the inflection point, where the slope of the line steepens greatly. For both assay methods, less than 10% of the pairs occur to the right of this point. All plots can be seen in Appendix II.

Standards

A total of 3573 results from fifteen different SRMs or standard reference materials were provided in the file QAQC_STD_070128_pch.csv. For each standard, the total number of samples as well as the mean and standard deviation of the returned assay results can be seen in the chart below. The standards were also divided by assay method, and upper and lower performance brackets are provided for both two and three standard deviations from the mean.

Standard	Method	Count	Mean	Std Dev	Mean - 2SD	Mean + 2SD	Mean - 3SD	Mean + 3SD
LS1	Au-AA23	27	0.0049	0.0005	0.0039	0.0059	0.0035	0.0064
Std-G	Au-AA23	686	1.4775	0.0638	1.3500	1.6051	1.2862	1.6688
	Au-AA25	92	1.4423	0.1427	1.1569	1.7276	1.0142	1.8703
Std-H	Au-AA23	523	2.9535	0.1256	2.7024	3.2046	2.5768	3.3302
	Au-AA25	138	2.9478	0.1092	2.7293	3.1662	2.6201	3.2754
Std-IC	Au-AA25	121	1.2316	0.0311	1.1693	1.2938	1.1382	1.3249
Std-IH	Au-AA25	44	7.0702	0.2019	6.6665	7.4740	6.4646	7.6759
Std-IL	Au-AA25	132	0.1030	0.0073	0.0884	0.1175	0.0811	0.1248
Std-IM	Au-AA25	146	3.2318	0.6953	1.8413	4.6224	1.1460	5.3177
Std-OX	Au-AA25	28	3.0282	0.0890	2.8502	3.2062	2.7612	3.2952
Std-SC	Au-AA25	119	0.9920	0.2405	0.5110	1.4729	0.2705	1.7134
Std-SH	Au-AA25	29	6.6928	0.1516	6.3895	6.9960	6.2379	7.1476
Std-SL	Au-AA25	716	0.0958	0.0081	0.0795	0.1121	0.0713	0.1202
Std-SM	Au-AA25	72	2.9657	0.4931	1.9795	3.9519	1.4864	4.4450
CDN-GS-3	Au-AA23	251	0.7959	0.0463	0.7032	0.8886	0.6569	0.9349
	Au-AA25	441	0.7959	0.0439	0.7081	0.8837	0.6642	0.9276
CDN-GS-3B	Au-AA25	6	3.5183	0.1556	3.2071	3.8296	3.0515	3.9852
CDN-GS-1P5A	Au-AA25	2	1.0650	0.3748	0.3155	1.8145	-0.0593	2.1893

Round-Robin results were provided from Barrick for all standards, excluding CDN-GS-3B, CDN-1P5A and LS1. However, CDN-GS-3B and 1P5A had only six and two samples respectively. Standard LS1 was used only for limestone drilling which did not support the assays encompassed in the resource estimate. The best value and standard deviation, as well as the number of assay labs participating in the Round-Robin, were used to determine the 95% confidence interval of each standard. Please note that these confidence intervals may vary from those previously provided by Barrick; the confidence intervals for this review were calculated using n=number of labs, not n=number of results, so that the uncertainty in each SRM would reflect the level of disagreement between the labs rather than the differences in analytical precision or how many times the material was assayed. Round-Robin data was also given for both 30 and 50g fire assay methods. Since the assay protocols used for the Donlin Creek data, Au-AA23 and Au-AA25, are both 30g fire-assays, the best values and confidence intervals used for this review are highlighted in yellow. Also note that the best value and standard deviation were not recalculated for the purpose of this study, as individual assay results from the Round-Robin were not provided for the majority of the standards.

Standard	Assay Method	Best Value Au (ppm)	Standard Deviation	Number of Labs	95% C.I.	95% C.I (N=# of labs)
		Provided	Provided	Provided	Provided	Calculated
Std-G	30g FA	1.444	0.093	7	0.18	0.0689
Std-H	30g FA	2.92	0.18	7	0.36	0.1333
Std-IC	30g FA	1.213	0.03	7	NA	0.0222
Std-IH	30g FA	6.929	0.25	7	NA	0.1852
	50g FA	6.954	0.29	7	NA	0.2148
Std-IL	30g FA	0.100	0.004	7	NA	0.0030
Std-IM	30g FA	3.124	0.16	7	NA	0.1185
	50g FA	3.079	0.12	7	NA	0.0889
Std-OX	30g FA	3.042	0.10	7	NA	0.0741
	50g FA	2.943	0.11	7	NA	0.0815
Std-SC	30g FA	0.994	0.04	7	NA	0.0296
Std-SH	30g FA	6.442	0.25	7	NA	0.1852
	50g FA	6.496	0.30	7	NA	0.2222
Std-SL	30g FA	0.097	0.01	7	NA	0.0074
Std-SM	30g FA	3.006	0.07	7	NA	0.0519
	50g FA	2.981	0.10	7	NA	0.0741
CDN-GS-3	30g FA	0.79	0.037	6	0.07	0.0296
CDN-GS-3B	NA	NA	NA	NA	NA	NA
CDN-GS-1P5A	NA	NA	NA	NA	NA	NA
LS1	NA	NA	NA	NA	NA	NA

A control chart for each standard with sufficient data was created, and can be found in Appendix III. The control charts show lines defined by the best value and the best value ± 5%, which are compared to the lab's global and moving averages. As per S. Long (Amec Consultant, 2007) the laboratory's accuracy is considered acceptable if the lab's moving average stays within ± 5% of the best value. Standard results were also plotted against order of assay to determine any relative bias based on prolonged periods of high or low values. If these periods of high or low values, or "clusters", are more than 10% above or below the Best Value, the accuracy of the lab during that time frame is off by more than 10%, which is not acceptable for measuring ore grade and these clusters would be flagged with red or blue lines. Results from ore grade samples of the principle commodity should show a low bias; generally speaking, a bias of <5% is widely accepted and a bias of >10% widely rejected.

In 2006, Barrick set acceptable limits for all standards at the mean ± 2 standard deviations. Failed standards were to be re-assayed until the results were acceptable or validated by duplication. The results of the control charts from this review are summarized in the table below:

Standard	Method	Date Range	No. of Outliers	Relative Bias (%)	Time Frames w/ Bias	No. of Samples Exceeding 2SD	No. Exceeding 3SD w/ Comments	No. of Samples Exceeding 3SD	No. Exceeding 3SD w/ Comments	Comments
Std-G	Au-AA23	Jan-Nov 2006	2	1.7	0	14	2	1	0	*Comments required for failed standards.
	Au-AA23	Dec 2006-Feb 2007	3	3.5	0					
	Au-AA25	NA	2	1.4	0	8	5	4	4	*3 outliers removed, obvious std swap.
Std-H	Au-AA23	2006	3	1.0	0	28	2	5	1	*Comments required for failed standards.
	Au-AA23	2007	0	3.4	0					
	Au-AA25	NA	2	1.0	0	5	1	1	1	*Comments required for failed standards.
Std-IC	Au-AA25	NA	0	1.5	0	5	5	0	0	
Std-IH	Au-AA25	NA	0	2.0	0	3	0	0	0	*Comments required for failed standards.
Std-IL	Au-AA25	NA	1	3.0	1	6	6	0	0	
Std-IM	Au-AA25	NA	2	0.4	0	3	3	0	0	*3 outliers removed, obvious std swap.
Std-OX	Au-AA25	NA	0	-0.5	0	1	0	0	0	*Comments required for failed standards.
Std-SC	Au-AA25	NA	0	-0.8	0	3	3	3	3	
Std-SH	Au-AA25	NA	0	3.9	0	0	0	0	0	
Std-SL	Au-AA25	Jan-Aug 2007	7	-2.2	0	16	10	5	5	*Comments required for failed standards.
	Au-AA25	Sept-Dec 2007	3	-0.4	0					
Std-SM	Au-AA25	NA	0	1.4	0	2	2	2	2	*2 outliers removed, obvious std swap.
CDN-GS-3	Au-AA23	NA	0	0.7	0	11	5	0	0	*Comments required for failed standards.
	Au-AA25	NA	0	0.9	0	18	14	0	0	
CDN-GS-3B	Au-AA25	NA	NA	NA	NA	NA	NA	NA	NA	*Only 6 samples.
CDN-GS-1P5A	Au-AA25	NA	NA	NA	NA	NA	NA	NA	NA	*Only 2 samples.
LS1	Au-AA23	NA	NA	NA	NA	NA	NA	NA	NA	* Limestone Drilling only, not included in resource model.

Factors that warrant further investigation or are cause for concern are highlighted in pink. All remaining outliers should be examined and determined whether or not they are a logical result of a standard swap, etc. If so, these outliers should be removed from the data set and the control charts should be rerun. A total of 123 standards exceeded the range of ± two standard deviations; fifty-eight (or 47%) were flagged with a comment regarding the failure. These comments commonly referred to a standard swap, or that the result failed by "X %" but was "within acceptable limits". Again, a definition for "acceptable limits" should be determined to remove ambiguity. The other sixty-five failed standards should be re-examined and flagged with appropriate comments. All failures that cannot be explained logically, are not within "acceptable limits", or do not fall within material defined as "waste" should be sent for re-assay. If one looks at the performance of the SRMs in terms of three standard deviations, instead of two, only twenty-one samples exceed the tolerated range, sixteen of which have an applicable comment. This leaves only five samples to be re-examined.

Summary

Blanks:

- Five samples or 0.1% failed before September 2006.
- 163 samples or 4.6% failed after September 2006.
- 92 failures require applicable comments.
- "Within acceptable limits" needs to be defined.
- The significance of the failed blanks should be determined in terms of ore cut-off.
- Blank carry-over or "smear" should be analysed to see if it can account for some of the blank failures.

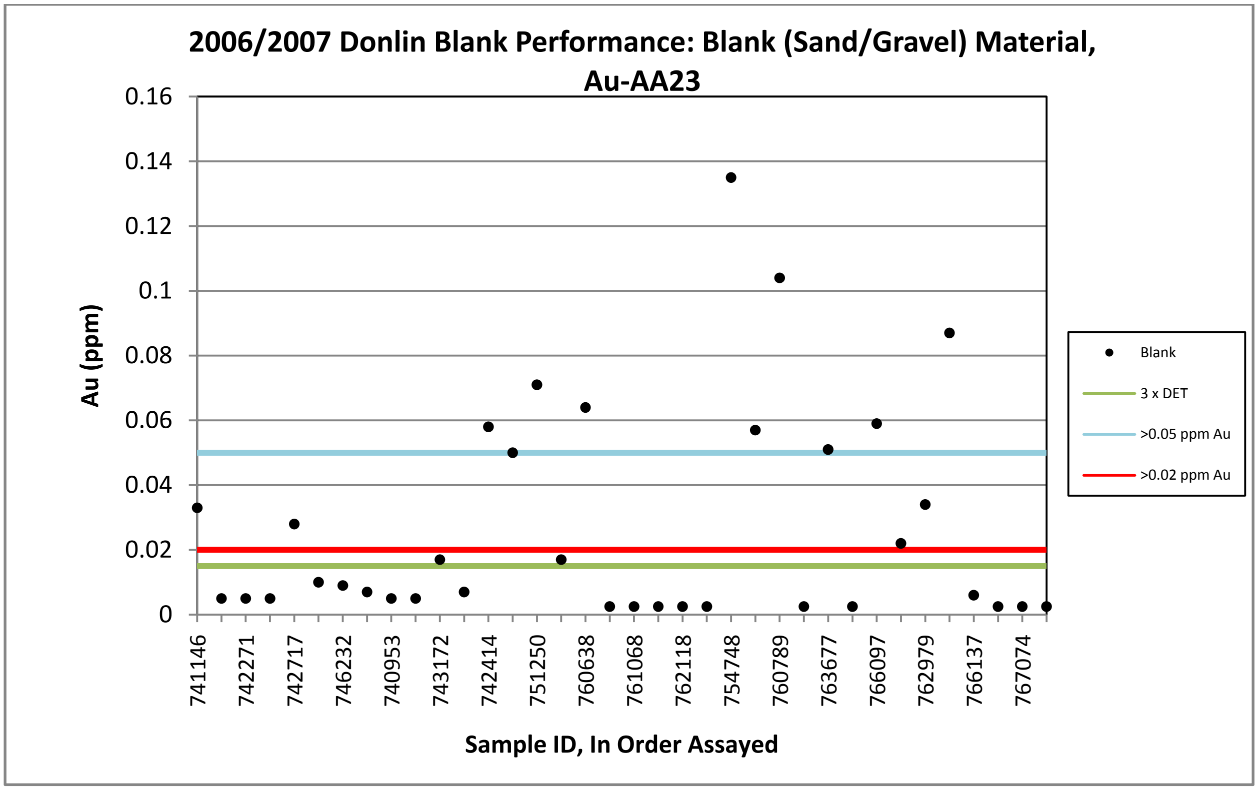
Duplicates:

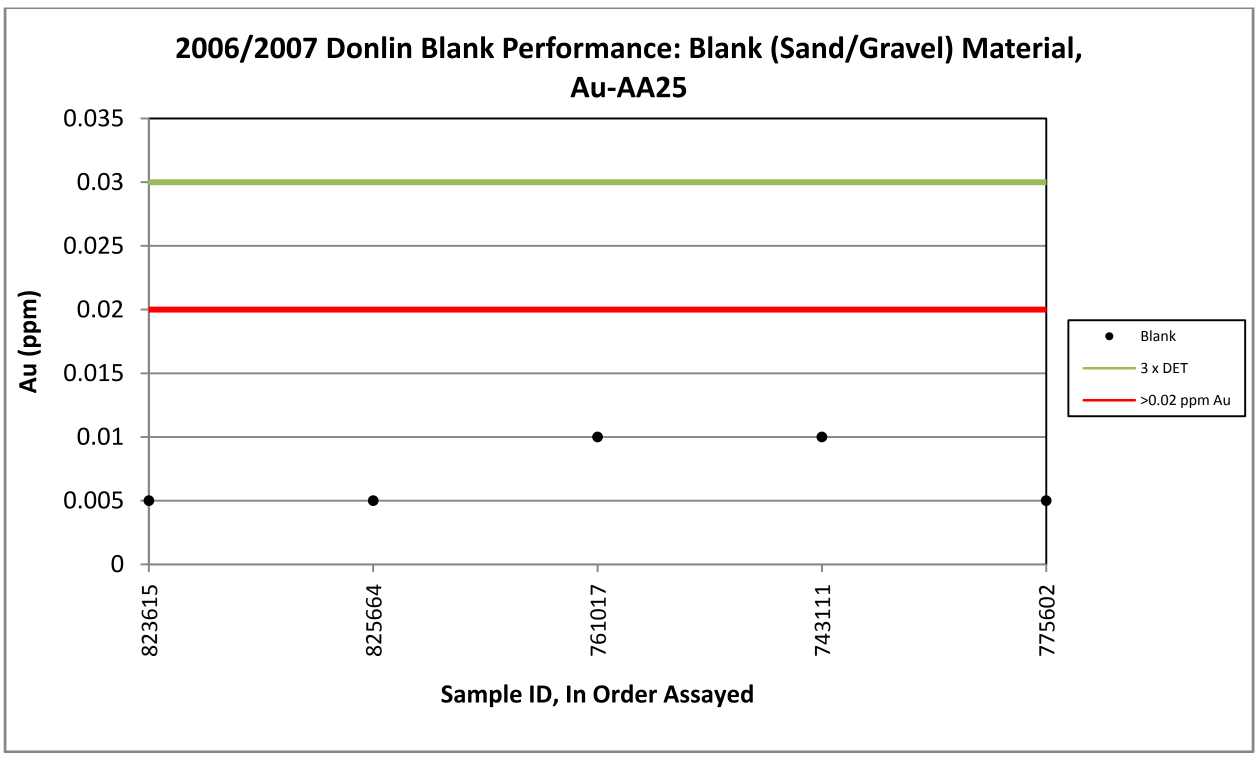
- Duplicate samples performed well and the pairs were 93% passing for both Au-AA23 and Au-AA25.

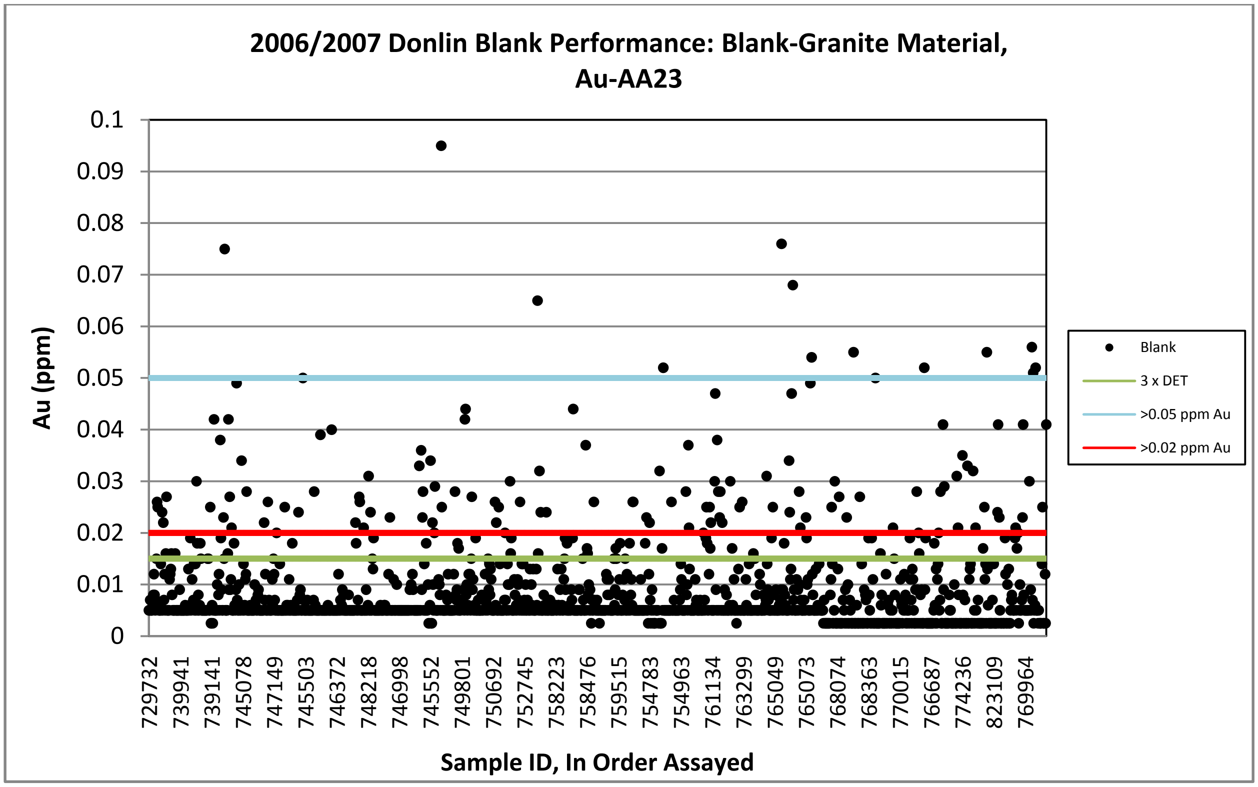
Standards:

- 123 samples exceeded the range of the mean ± 2 standard deviations; 65 of which have no applicable comment and should be re-examined.
- 21 samples exceeded the range of the mean ± 3 standard deviations; 5 of which have no applicable comment. At the very least, these five should be re-examined and dealt with in an appropriate manner.
- "Within acceptable limits" needs to be defined.

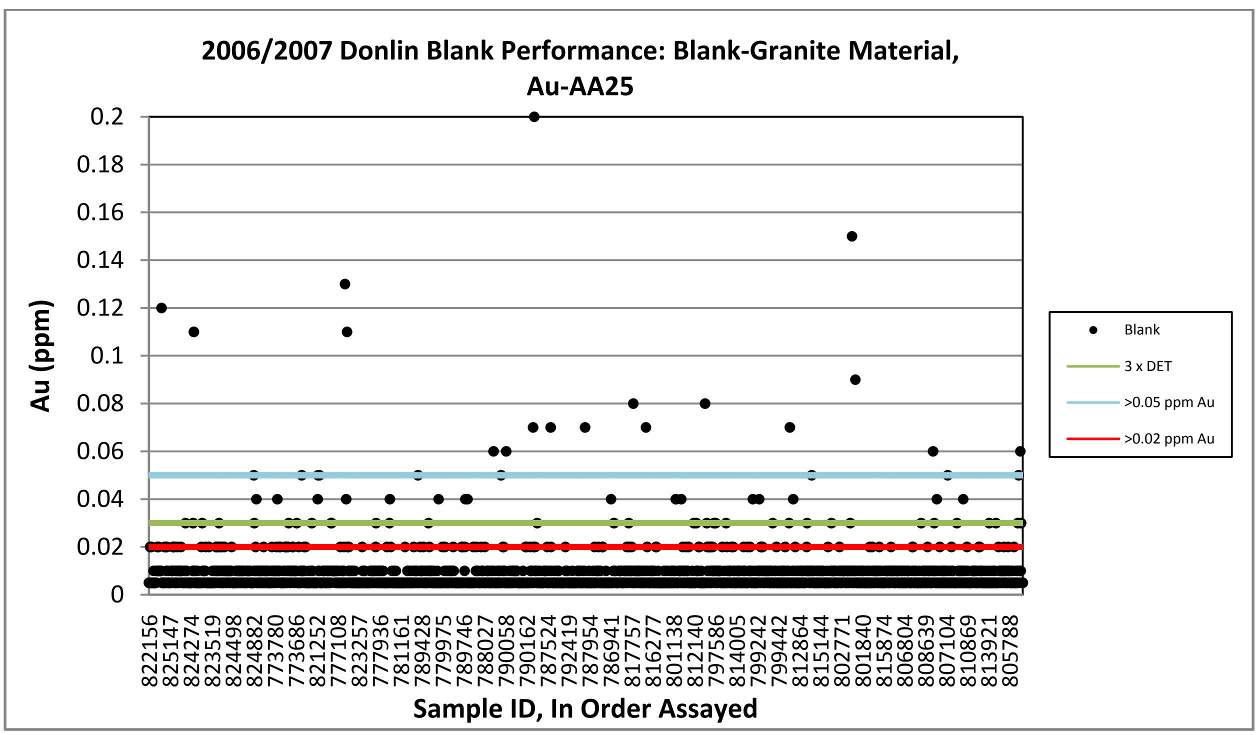
Appendix I: Blank Charts





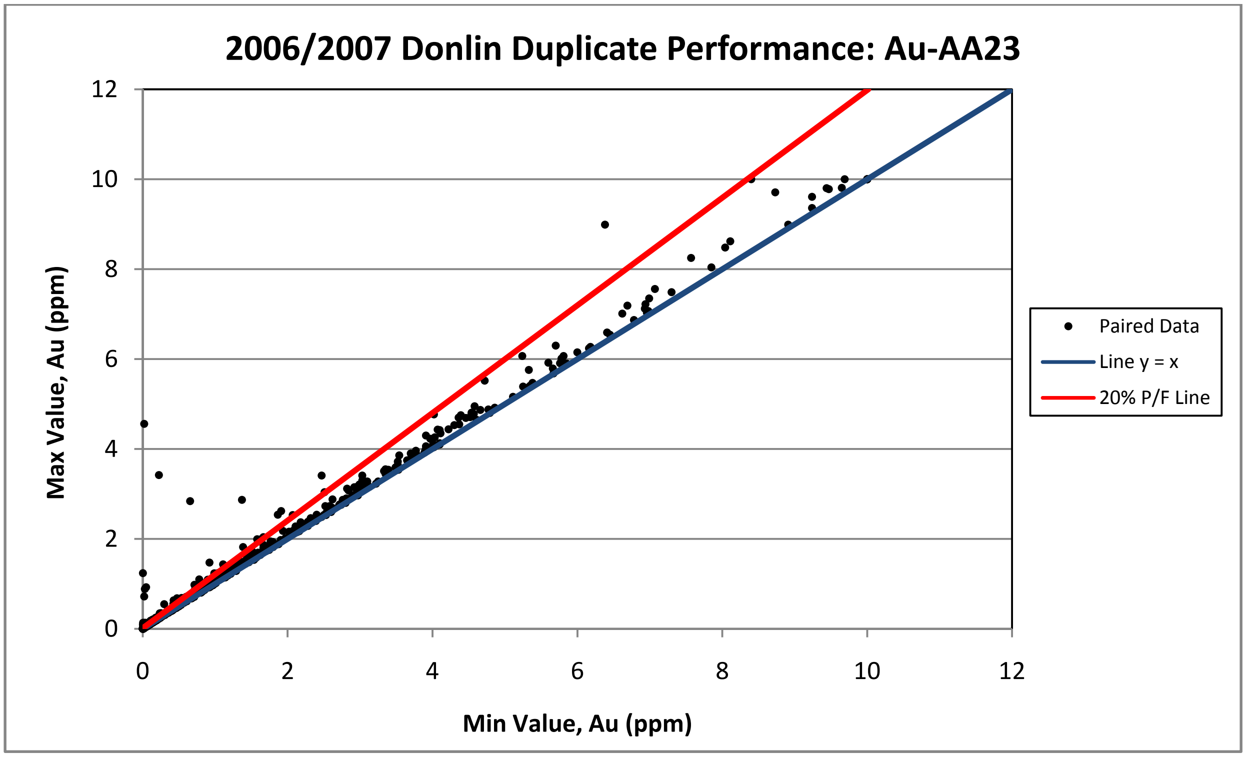


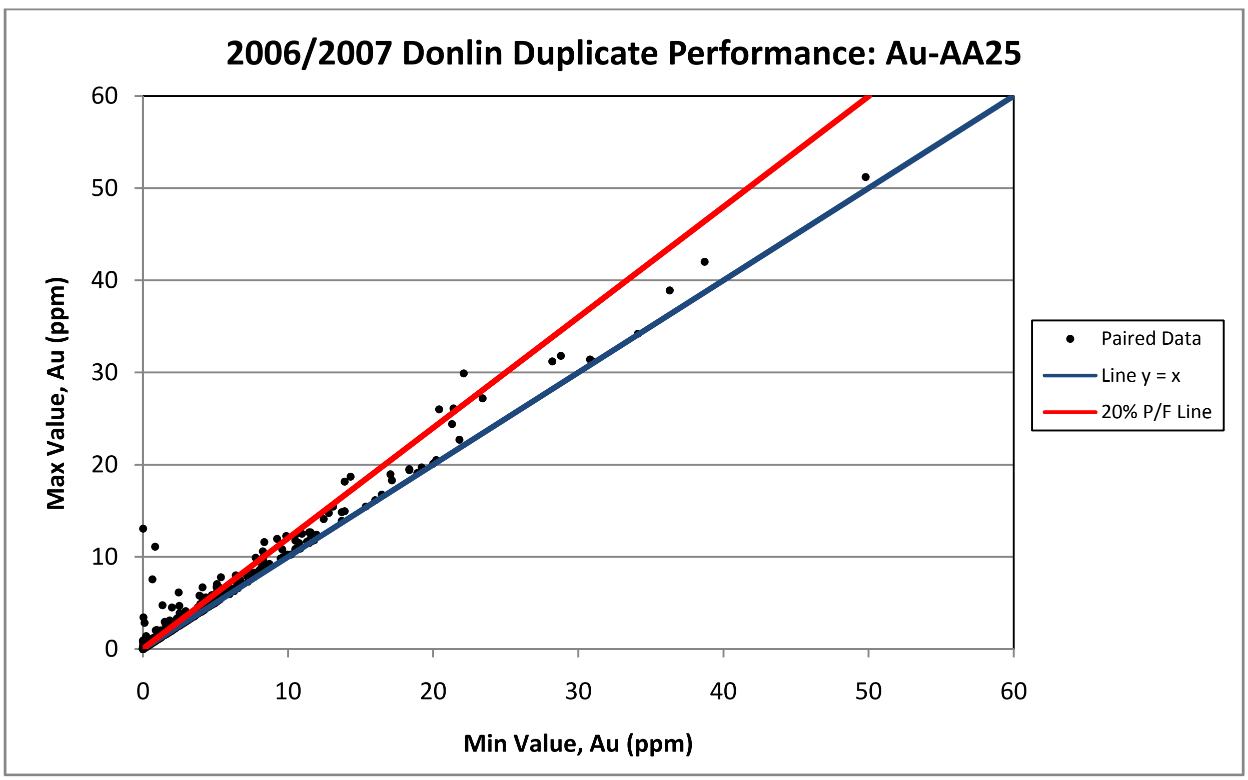
2006/2007 Donlin Blank Performance: Blank-Granite Material, Au-AA23
Au (ppm)
Sample ID, In Order Assayed
Blank
3 x DET
>0.05 ppm Au
>0.02 ppm Au

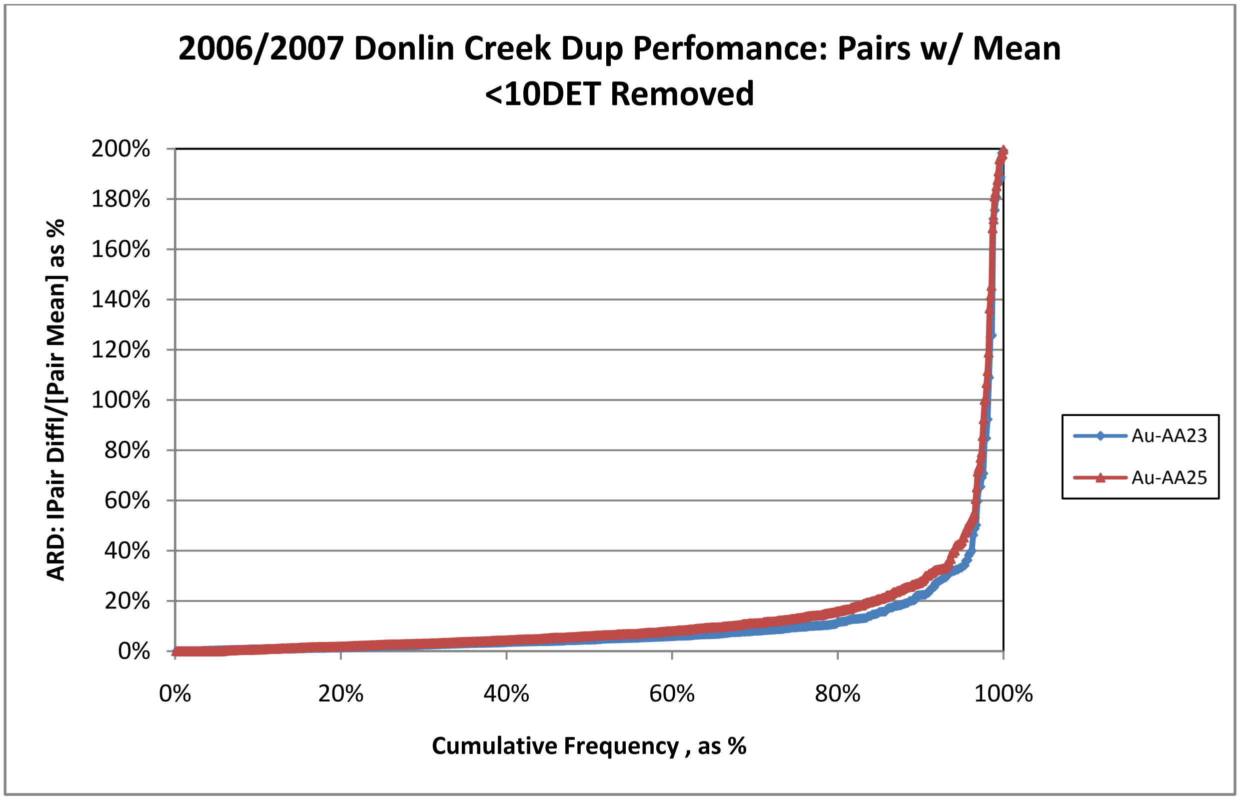


2006/2007 Donlin Blank Performance: Blank-Granite Material, Au-AA25
Au (ppm)
Sample ID, In Order Assayed
Blank
3 x DET
>0.05 ppm Au
>0.02 ppm Au

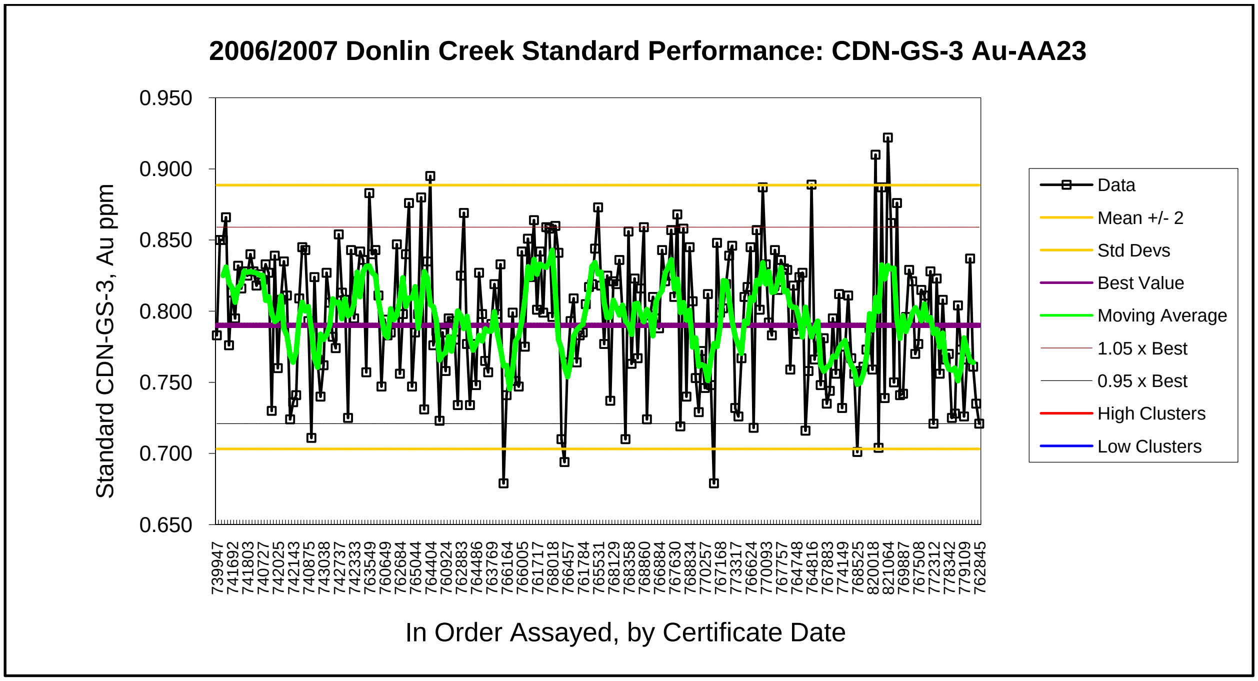
Appendix II: Duplicate Charts

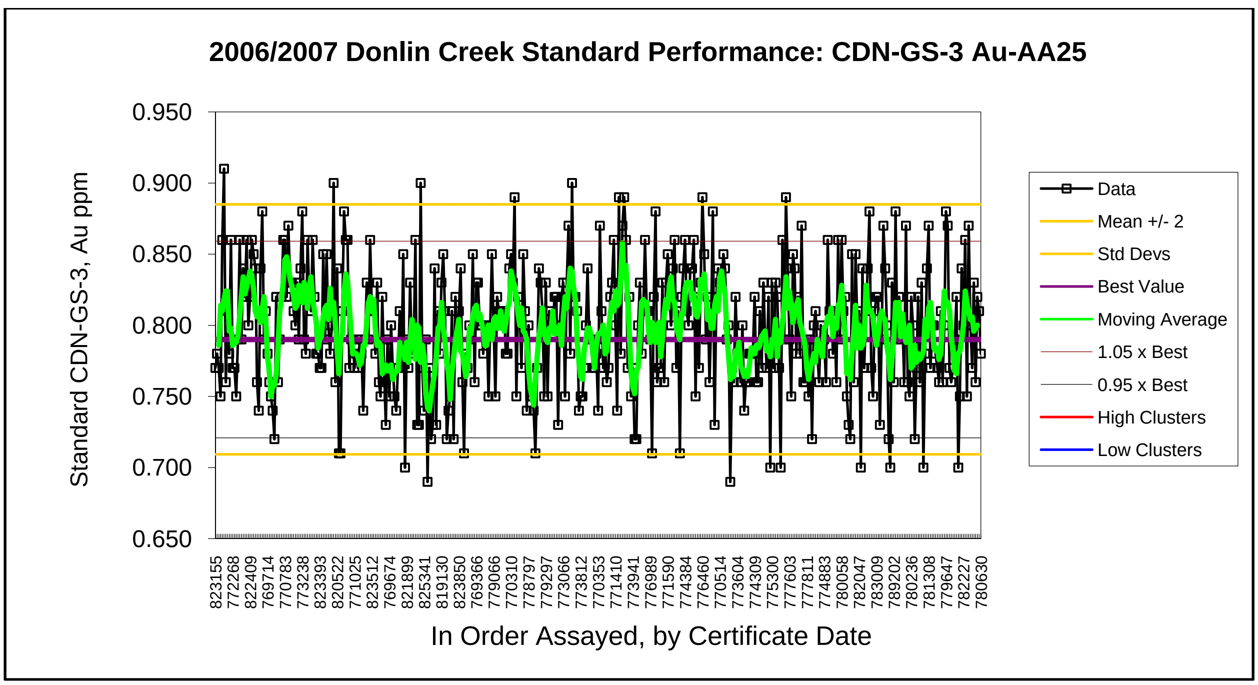


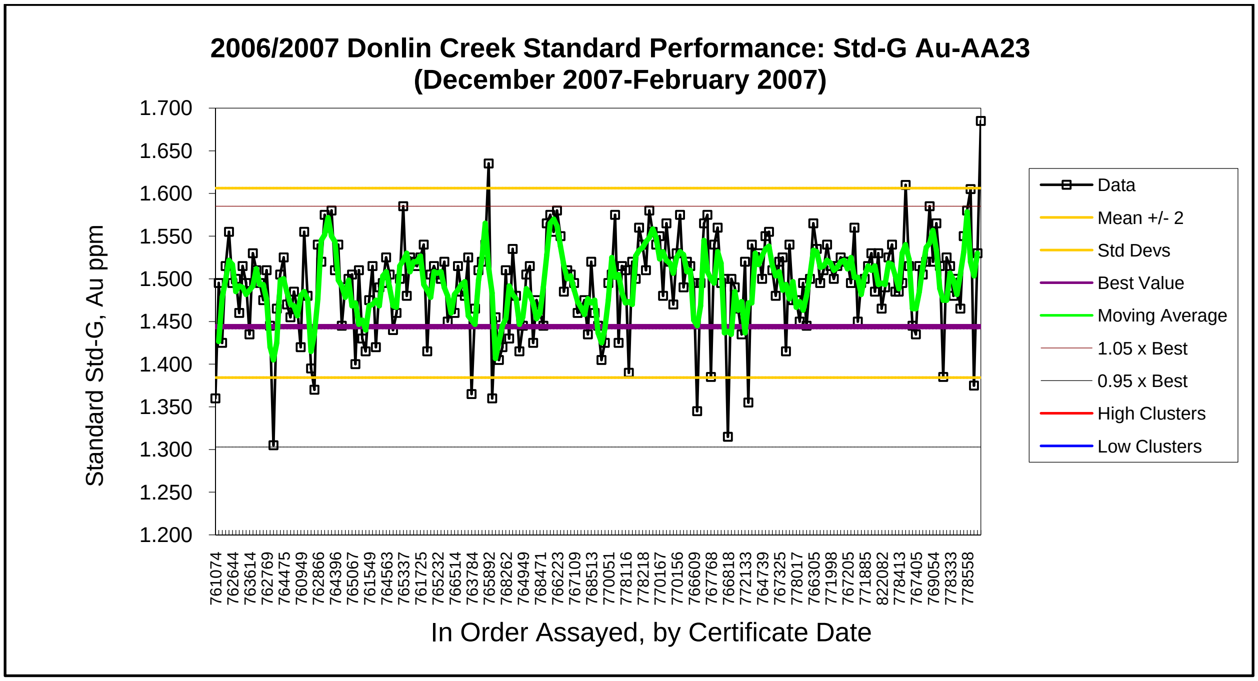


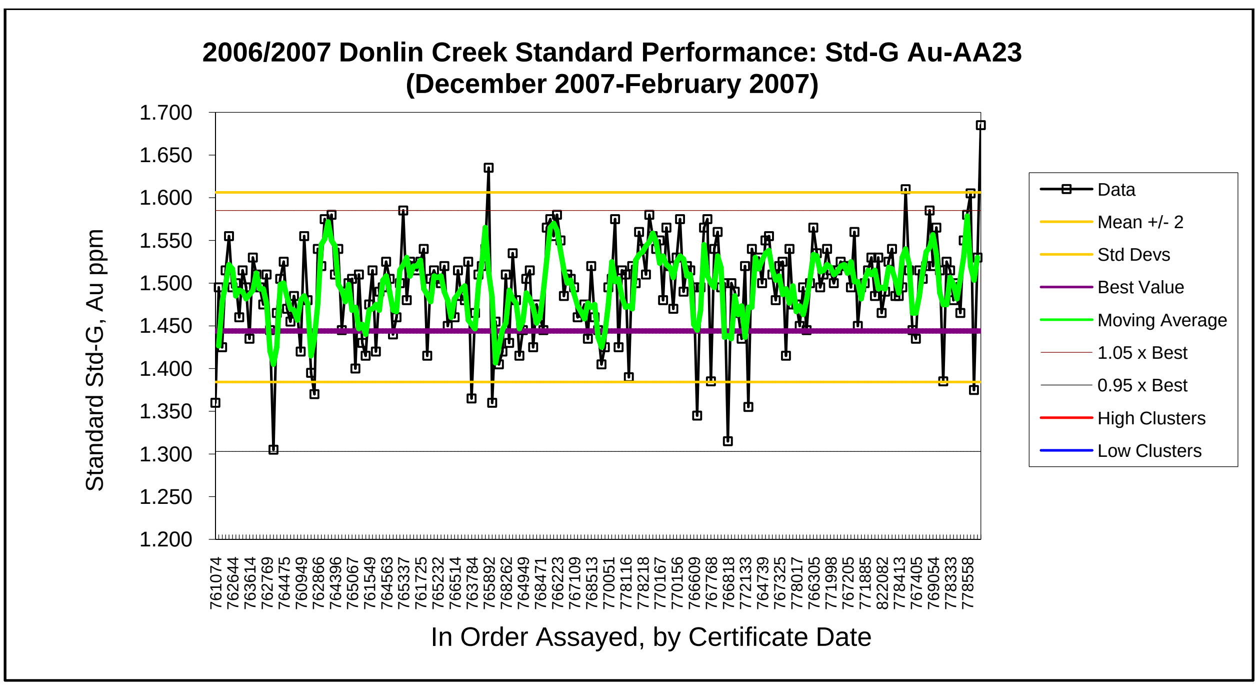

2006/2007 Donlin Creek Dup Perfomance: Pairs w/ Mean <10DET Removed
ARD: IPair DiffI/[Pair Mean] as %
200%
180%
160%
140%
120%
100%
80%
60%
40%
20%
0%
0%
20%
40%
60%
80%
100%
Cumulative Frequency , as %
Au-AA23
Au-AA25

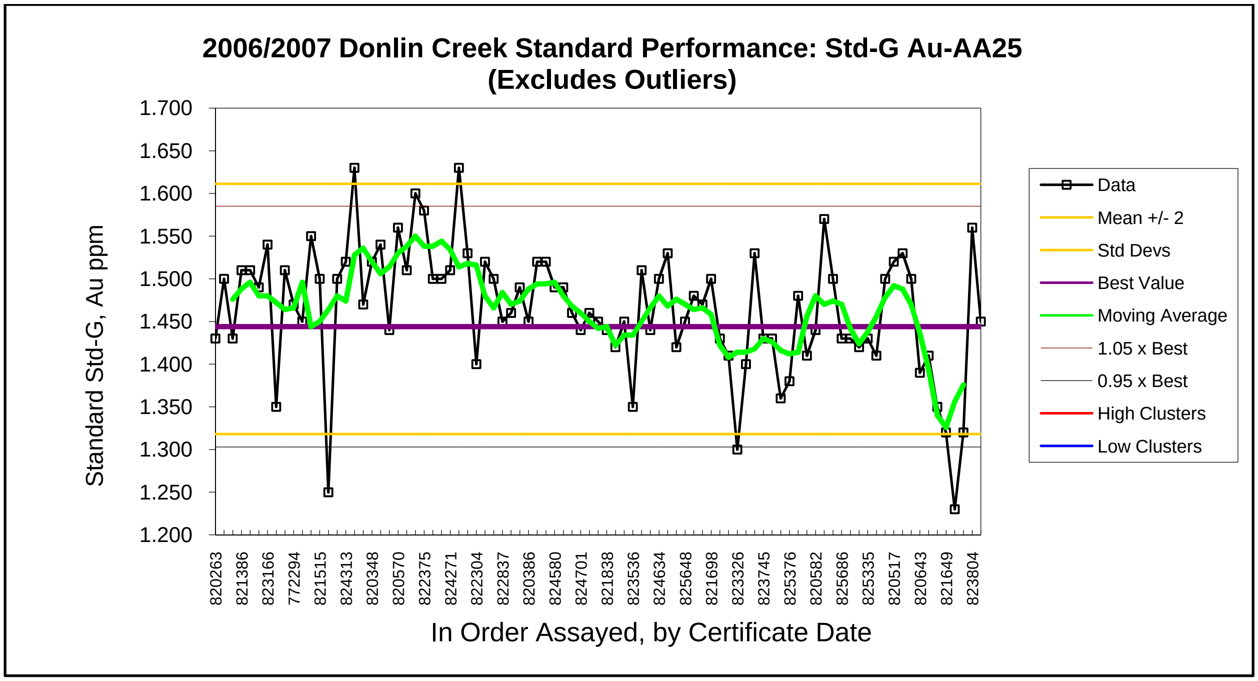
Appendix III: Standard Charts



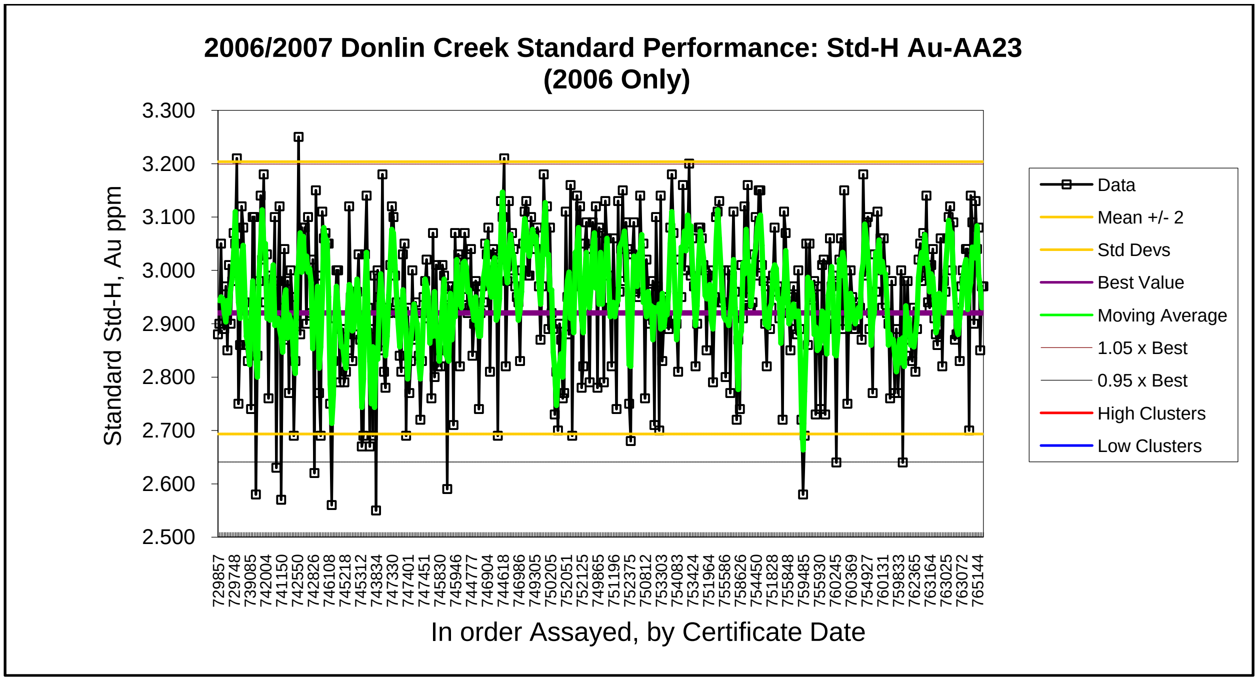



2006/2007 Donlin Creek Standard Performance: Std-G Au-AA23
(December 2007-February 2007)
Standard Std-G, Au ppm
1.700
1.650
1.600
1.550
1.500
1.450
1.400
1.350
1.300
1.250
1.200
761074
762644
763614
762769
764475
760949
762866
764396
765067
761549
764563
765337
761725
765232
766514
763784
765892
768262
764949
768471
766223
767109
768513
770051
778116
778218
770167
770156
766609
767768
766818
772133
764739
767325
778017
766305
771998
767205
771885
822082
778413
767405
769054
778333
778558
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters

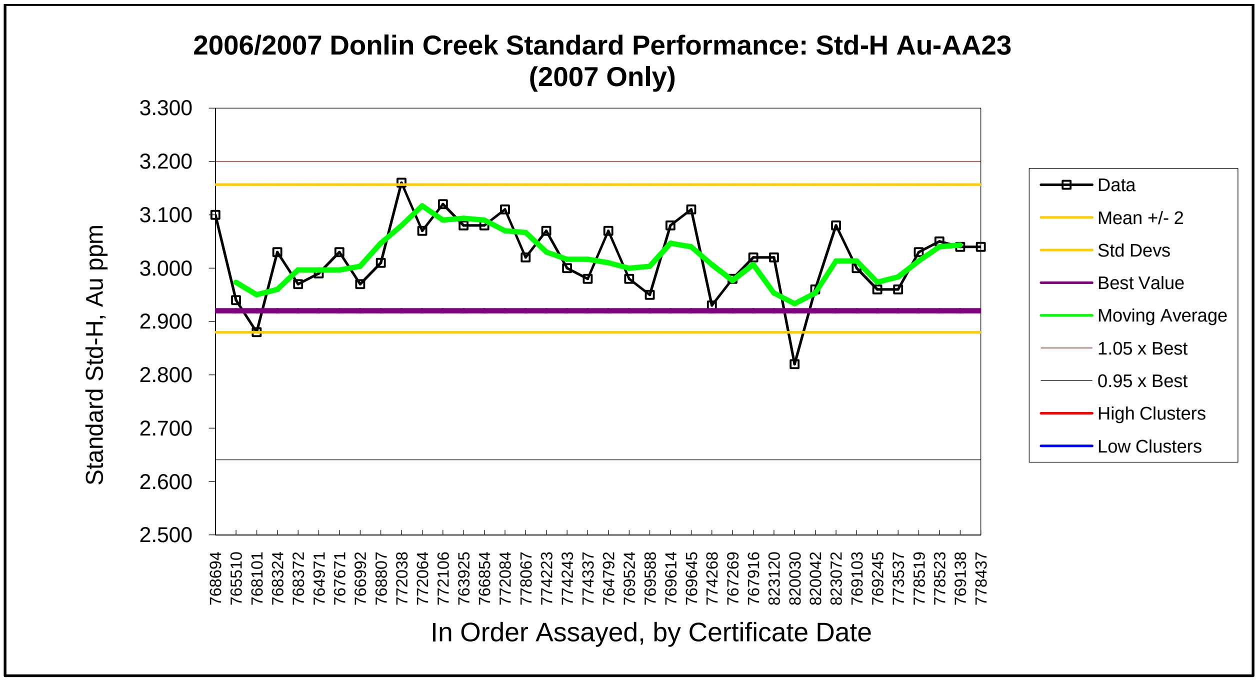

2006/2007 Donlin Creek Standard Performance: Std-G Au-AA23
(December 2007-February 2007)
Standard Std-G, Au ppm
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters

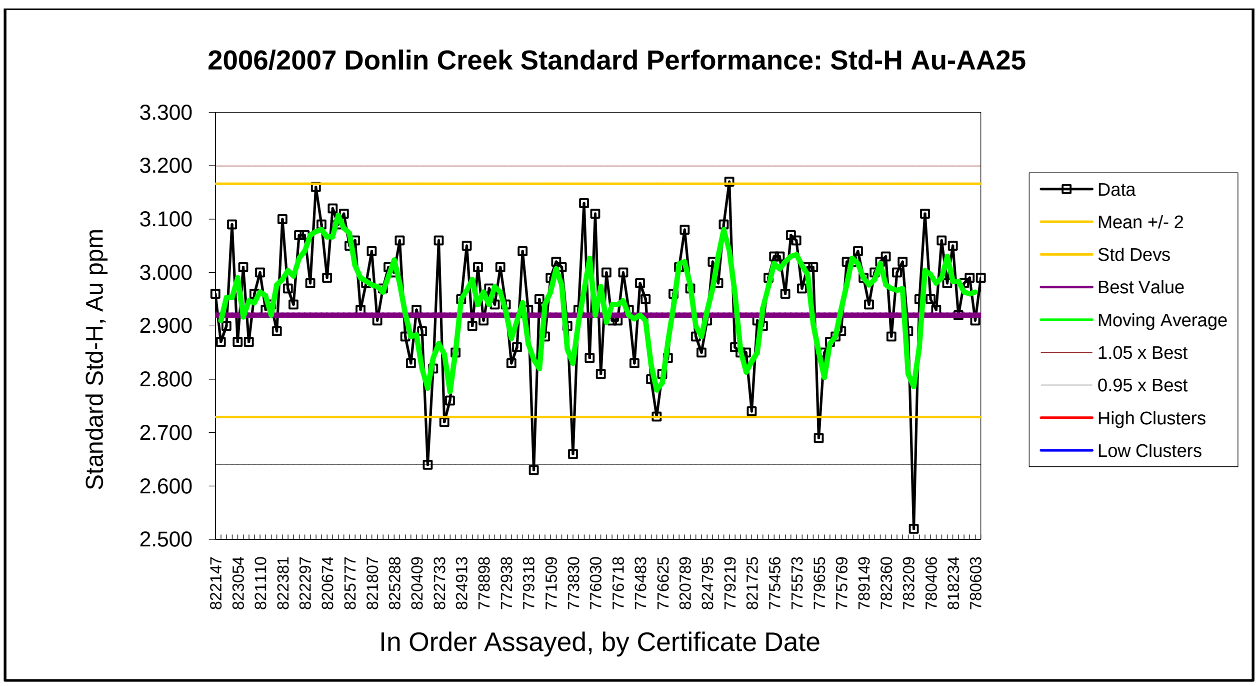


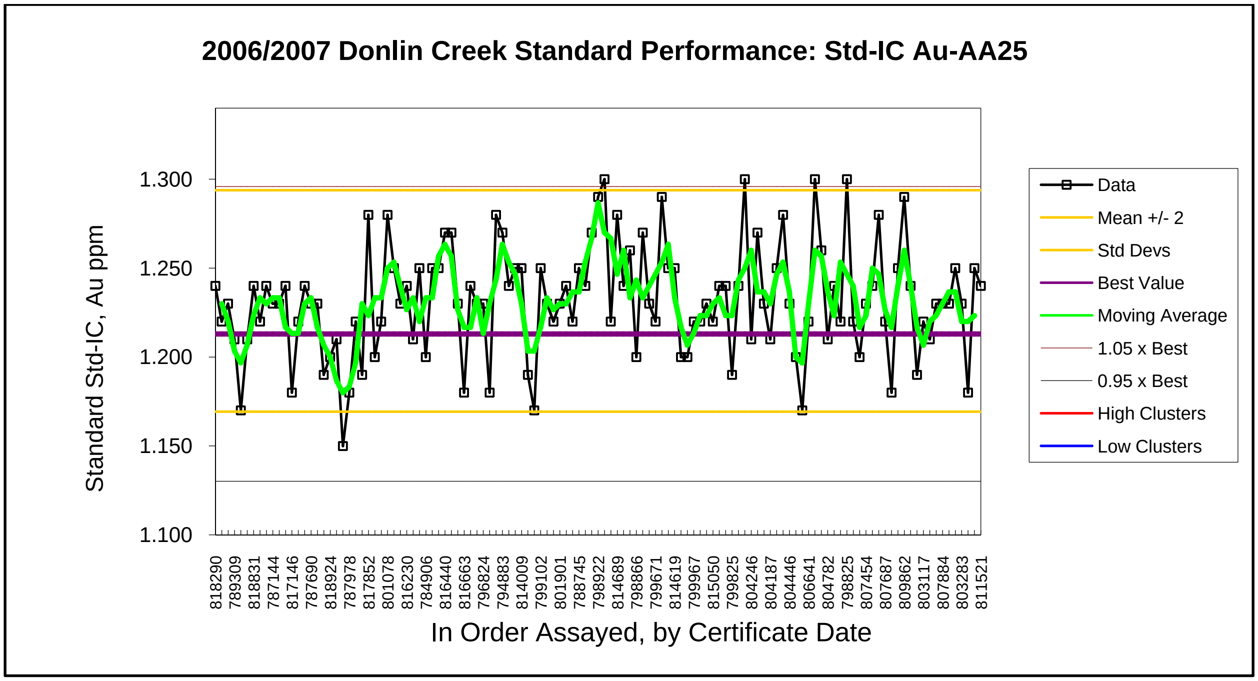
2006/2007 Donlin Creek Standard Performance: Std-G Au-AA25
(Excludes Outliers)
Standard Std-G, Au ppm
1.700
1.650
1.600
1.550
1.500
1.450
1.400
1.350
1.300
1.250
1.200
820263
821386
823166
772294
821515
824313
820348
820570
822375
824271
822304
822837
820386
824580
824701
821838
823536
824634
825648
821698
823326
823745
825376
820582
825686
825335
820517
820643
821649
823804
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters



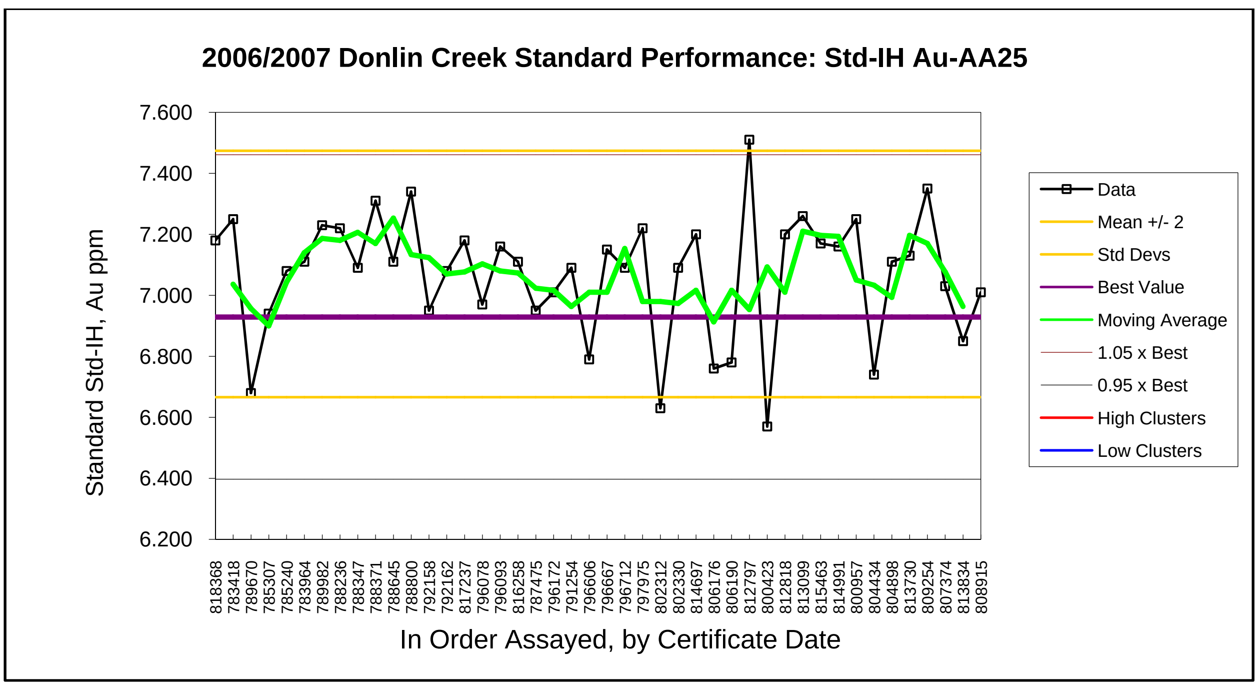
2006/2007 Donlin Creek Standard Performance: Std-H Au-AA23
(2006 Only)
Standard Std-H, Au ppm
3.300
3.200
3.100
3.000
2.900
2.800
2.700
2.600
2.500
729857
729748
739085
742004
741150
742550
742826
746108
745218
745312
743834
747330
747401
747451
745830
745946
744777
746904
744618
746986
749305
750205
752051
752125
749865
751196
752375
750812
753303
754083
753424
751964
755586
758626
754450
751828
755848
759485
755930
760245
760369
754927
760131
759833
762365
763164
763025
763072
765144
In order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters


2006/2007 Donlin Creek Standard Performance: Std-H Au-AA23
(2007 Only)
Standard Std-H, Au ppm
3.300
3.200
3.100
3.000
2.900
2.800
2.700
2.600
2.500
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters

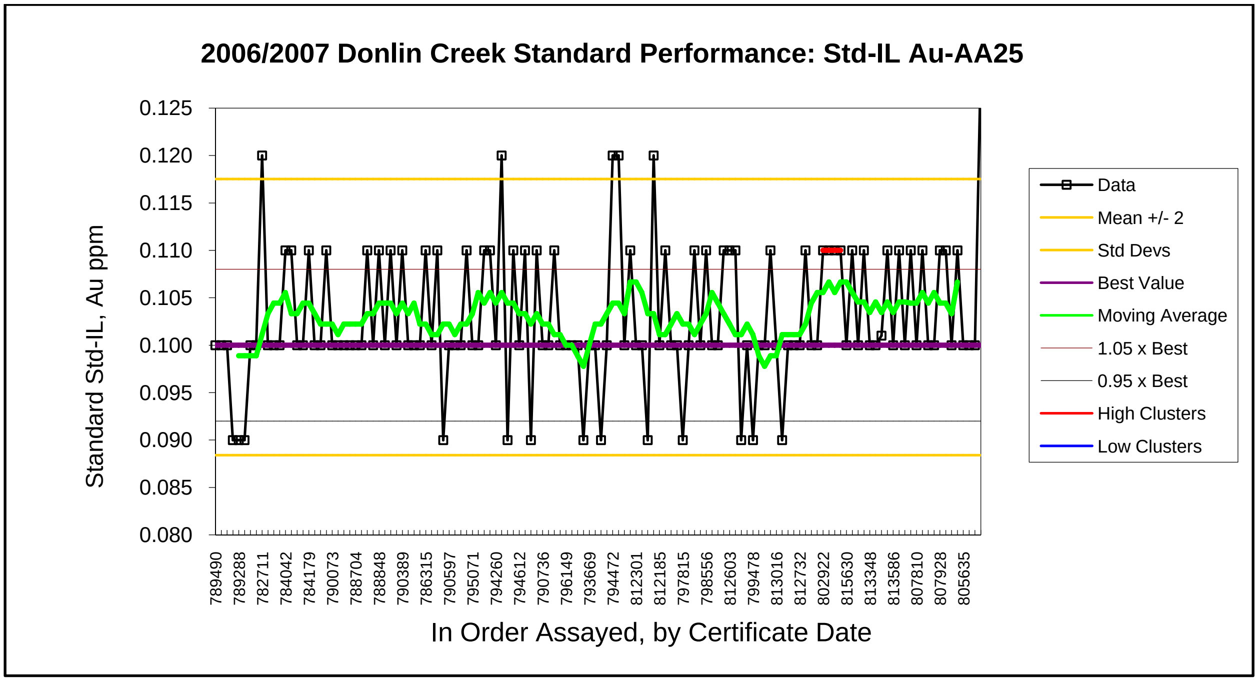

2006/2007 Donlin Creek Standard Performance: Std-H Au-AA25
Standard Std-H, Au ppm
3.300
3.200
3.100
3.000
2.900
2.800
2.700
2.600
2.500
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters



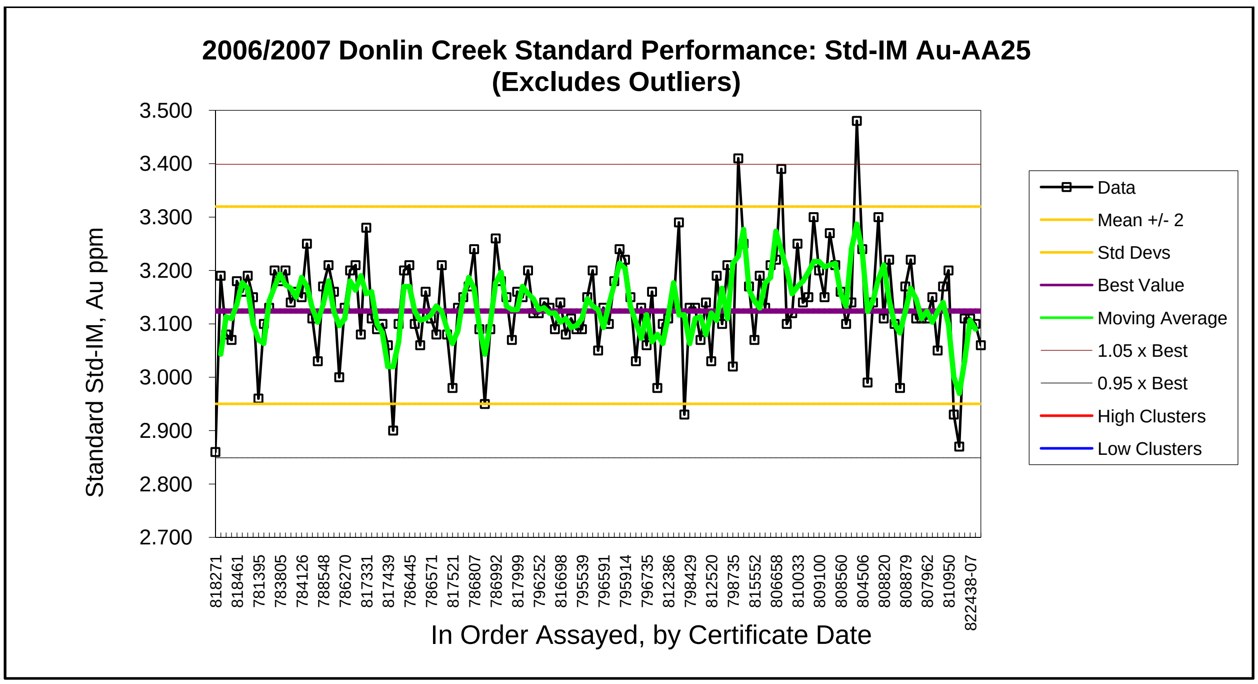
2006/2007 Donlin Creek Standard Performance: Std-IC Au-AA25
Standard Std-IC, Au ppm
1.300
1.250
1.200
1.150
1.100
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters



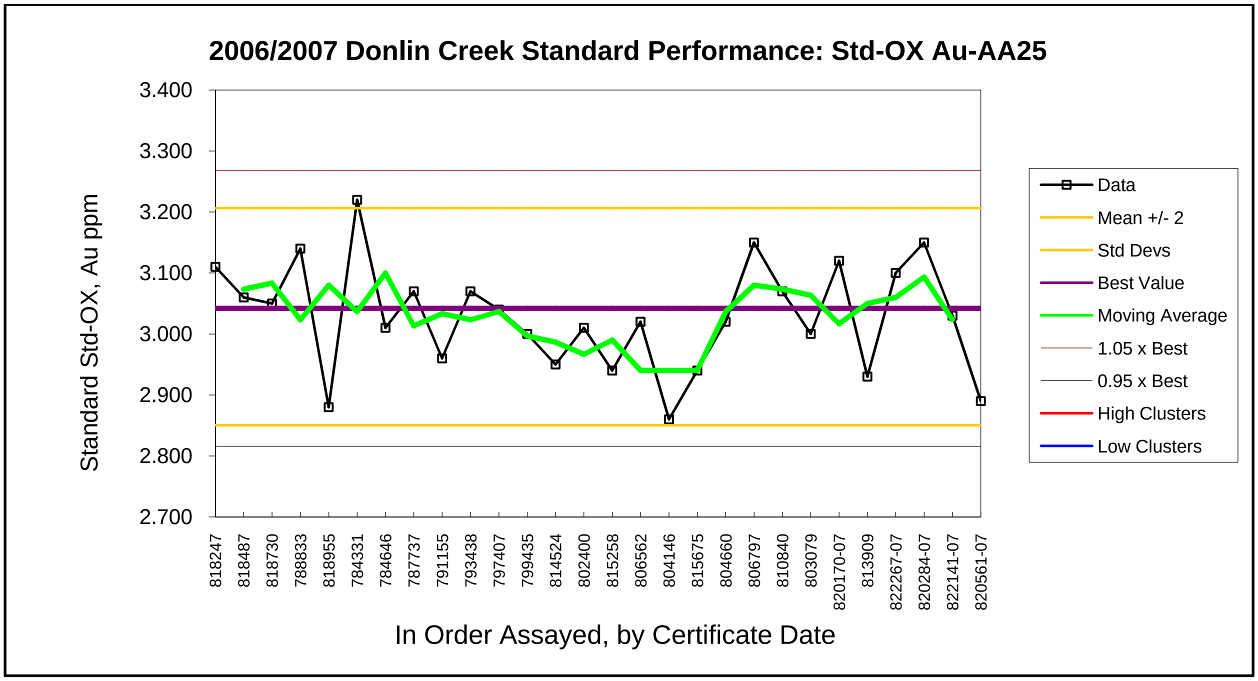
2006/2007 Donlin Creek Standard Performance: Std-IH Au-AA25
Standard Std-IH, Au ppm
7.600
7.400
7.200
7.000
6.800
6.600
6.400
6.200
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters



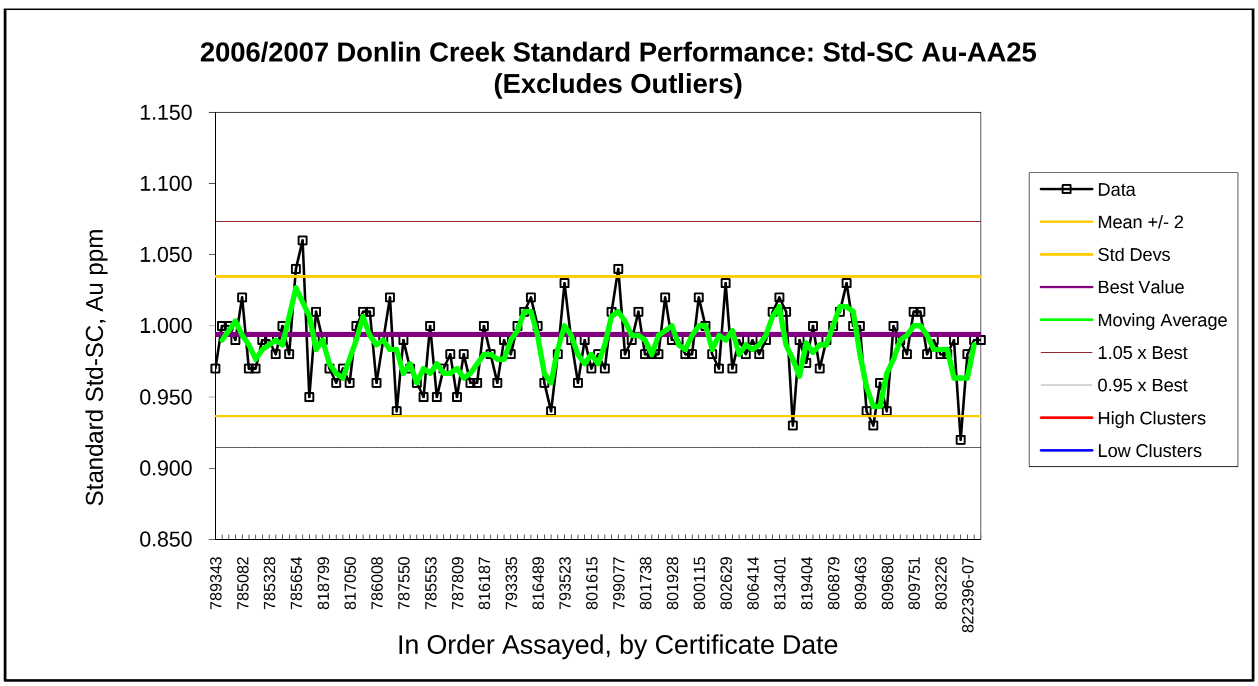
2006/2007 Donlin Creek Standard Performance: Std-IL Au-AA25
Standard Std-IL, Au ppm
0.125
0.120
0.115
0.110
0.105
0.100
0.095
0.090
0.085
0.080
789490
789288
782711
784042
784179
790073
788704
788848
790389
786315
790597
795071
794260
794612
790736
796149
793669
794472
812301
812185
797815
798556
812603
799478
813016
812732
802922
815630
813348
813586
807810
807928
805635
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters



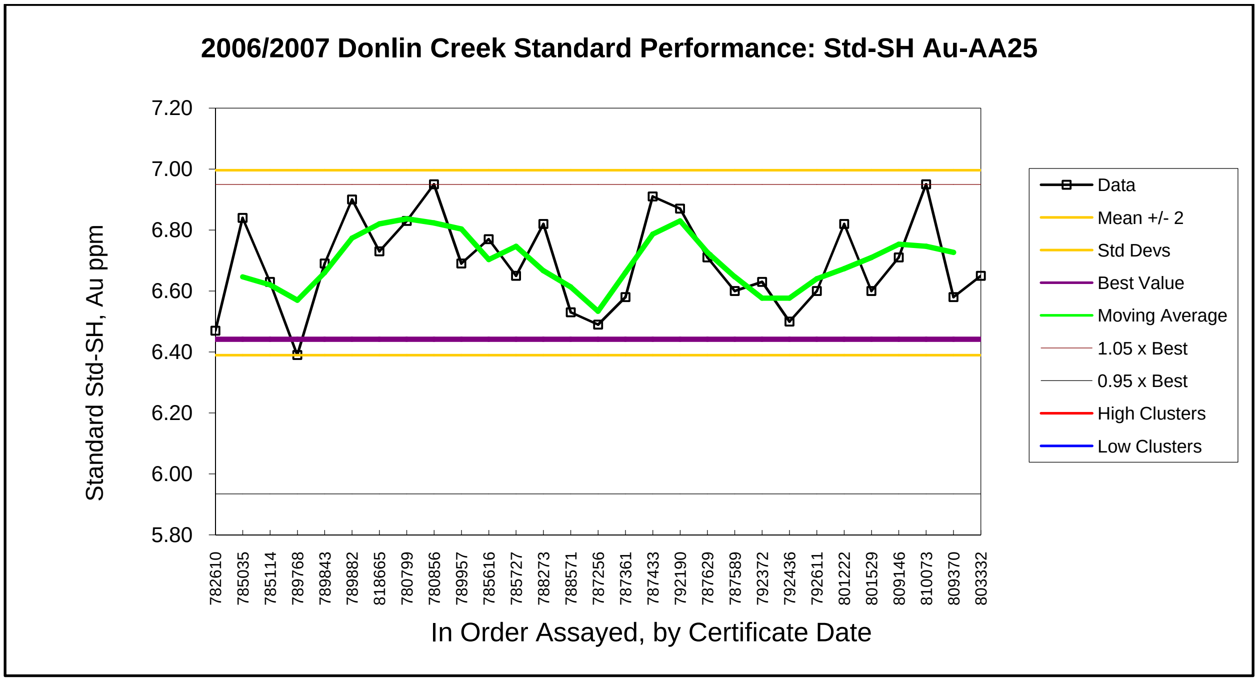
2006/2007 Donlin Creek Standard Performance: Std-IM Au-AA25
(Excludes Outliers)
Standard Std-IM, Au ppm
3.500
3.400
3.300
3.200
3.100
3.000
2.900
2.800
2.700
818271
818461
781395
783805
784126
788548
786270
817331
817439
786445
786571
817521
786807
786992
817999
796252
816698
795539
796591
795914
796735
812386
798429
812520
798735
815552
806658
810033
809100
808560
804506
808820
808879
807962
810950
822438-07
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters

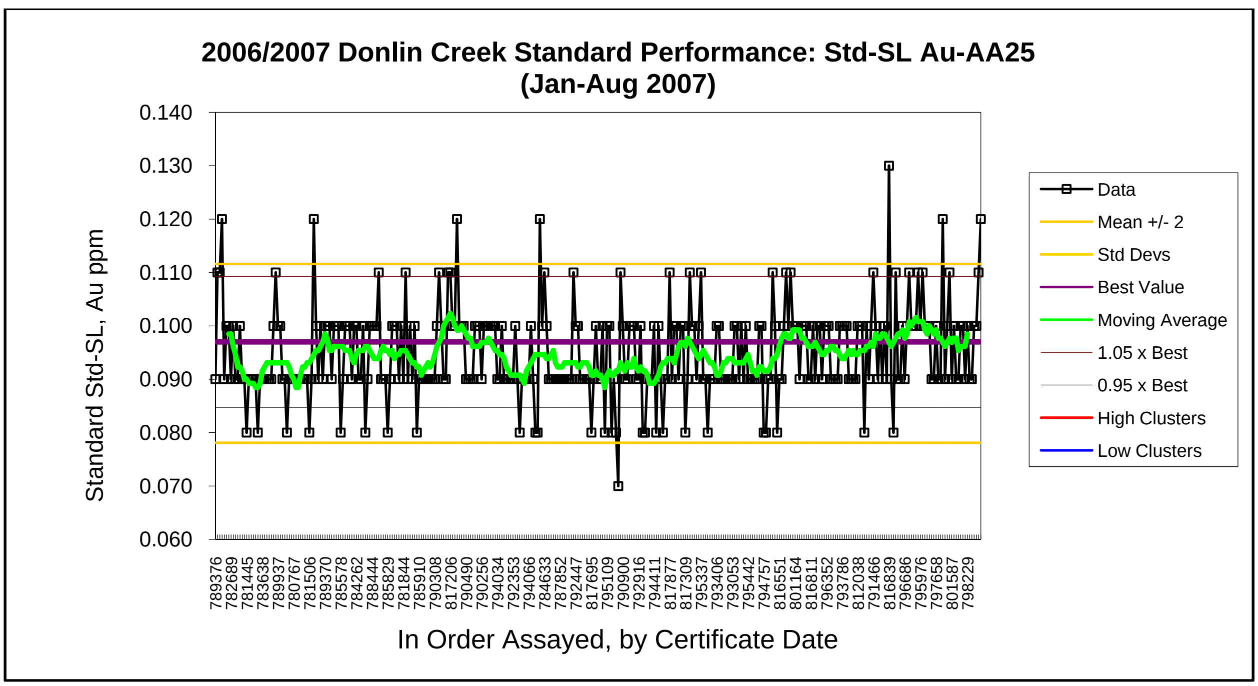


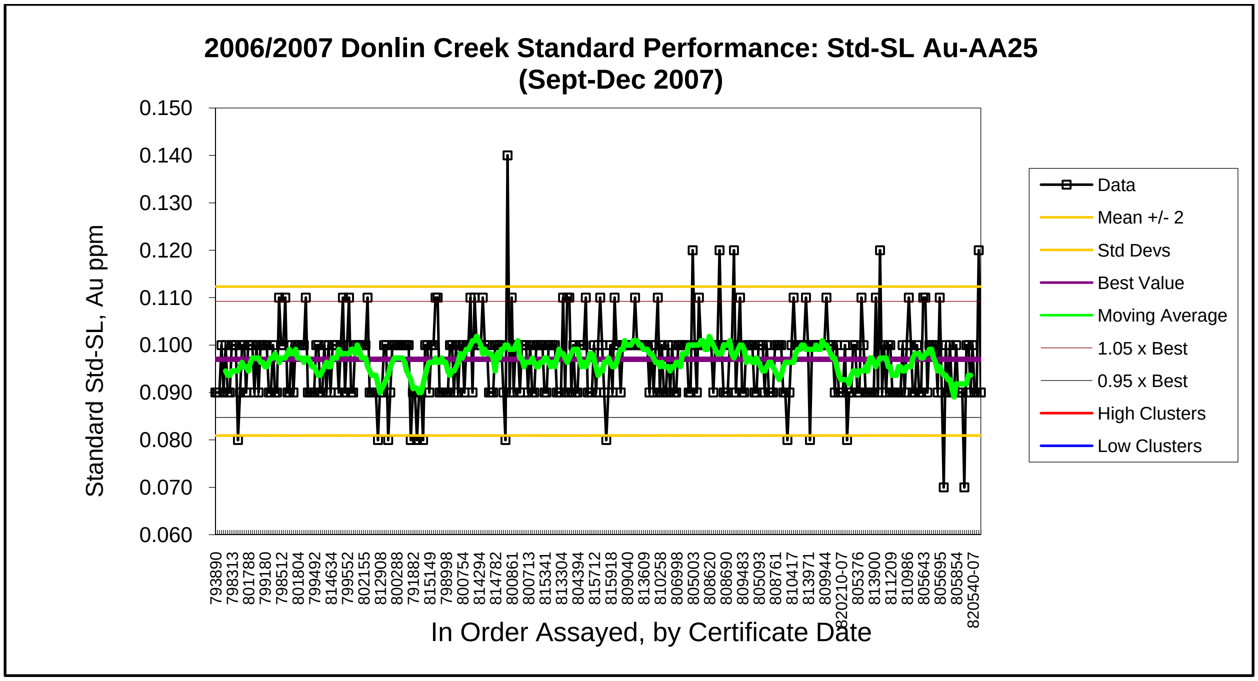
2006/2007 Donlin Creek Standard Performance: Std-OX Au-AA25
Standard Std-OX, Au ppm
3.400
3.300
3.200
3.100
3.000
2.900
2.800
2.700
818247
818487
818730
788833
818955
784331
784646
787737
791155
793438
797407
799435
814524
802400
815258
806562
804146
815675
804660
806797
810840
803079
820170-07
813909
822267-07
820284-07
822141-07
820561-07
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters

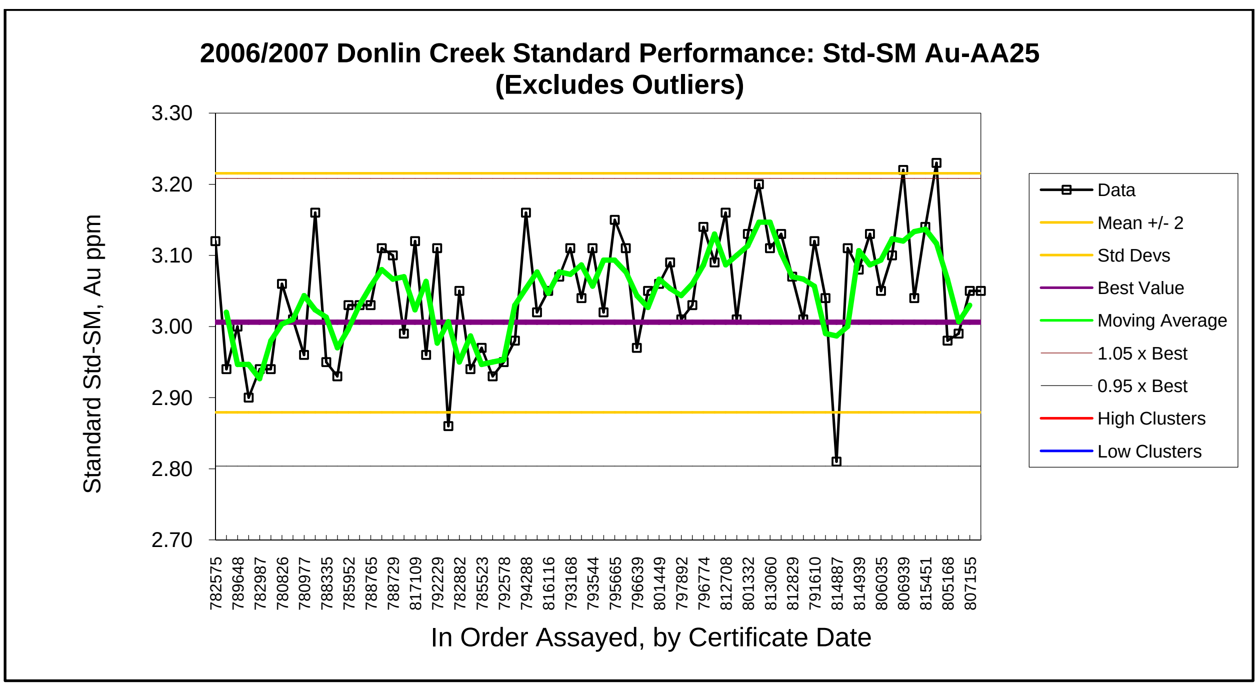


2006/2007 Donlin Creek Standard Performance: Std-SC Au-AA25
(Excludes Outliers)
Standard Std-SC, Au ppm
1.150
1.100
1.050
1.000
0.950
0.900
0.850
789343
785082
785328
785654
818799
817050
786008
787550
785553
787809
816187
793335
816489
793523
801615
799077
801738
801928
800115
802629
806414
813401
819404
806879
809463
809680
809751
803226
822396-07
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters


2006/2007 Donlin Creek Standard Performance: Std-SH Au-AA25
Standard Std-SH, Au ppm
7.20
7.00
6.80
6.60
6.40
6.20
6.00
5.80
782610
785035
785114
789768
789843
789882
818665
780799
780856
789957
785616
785727
788273
788571
787256
787361
787433
792190
787629
787589
792372
792436
792611
801222
801529
809146
810073
809370
803332
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters


2006/2007 Donlin Creek Standard Performance: Std-SL Au-AA25
(Jan-Aug 2007)
Standard Std-SL, Au ppm
0.140
0.130
0.120
0.110
0.100
0.090
0.080
0.070
0.060
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters

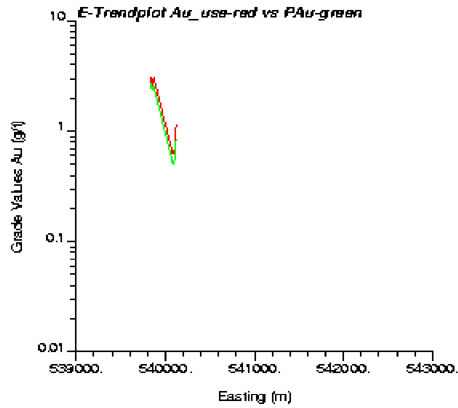

2006/2007 Donlin Creek Standard Performance: Std-SL Au-AA25
(Sept-Dec 2007)
Standard Std-SL, Au ppm
0.150
0.140
0.130
0.120
0.110
0.100
0.090
0.080
0.070
0.060
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters

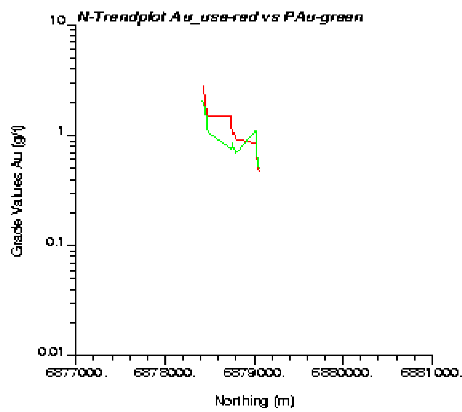

2006/2007 Donlin Creek Standard Performance: Std-SM Au-AA25
(Excludes Outliers)
Standard Std-SM, Au ppm
3.30
3.20
3.10
3.00
2.90
2.80
2.70
In Order Assayed, by Certificate Date
Data
Mean +/- 2
Std Devs
Best Value
Moving Average
1.05 x Best
0.95 x Best
High Clusters
Low Clusters

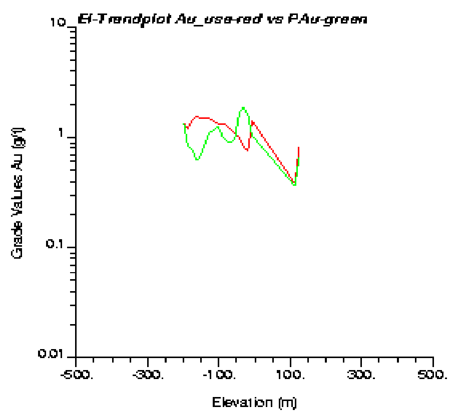
APPENDIX D

Block Model Swath Plots by Rock Type

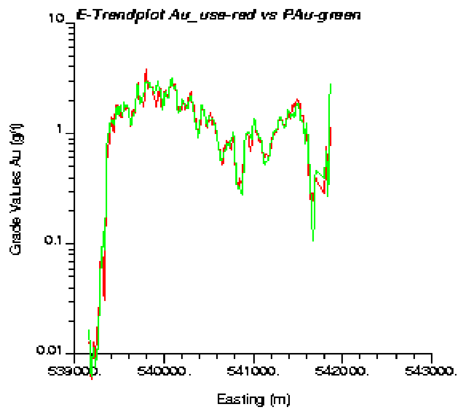
DonlinCreek Trend Plots for rck01_other

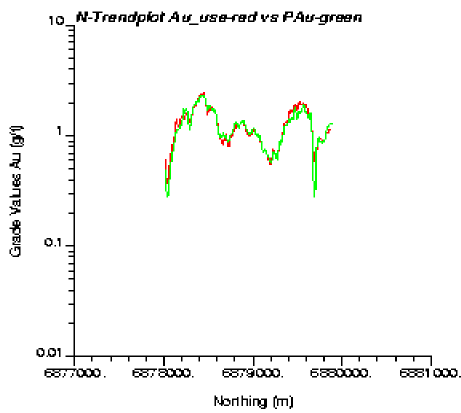


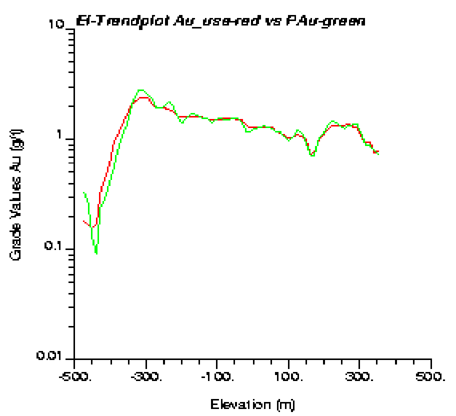




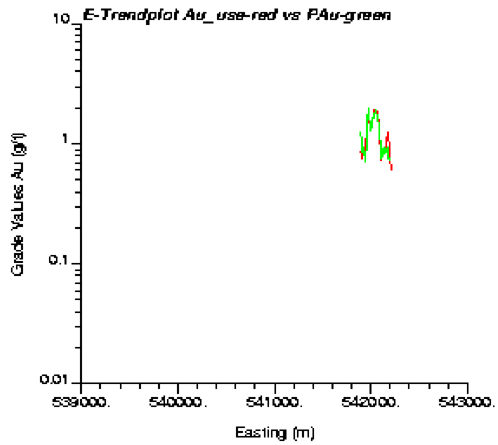
DonlinCreek Trend Plots for rck02_rda

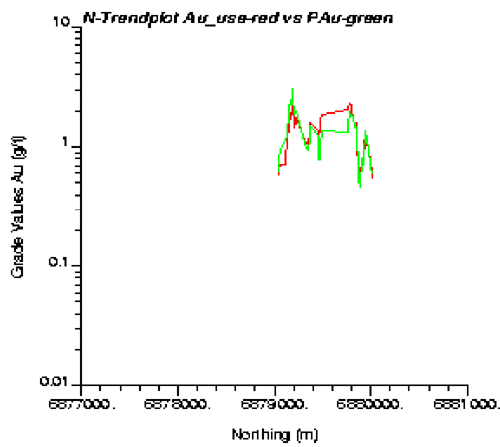


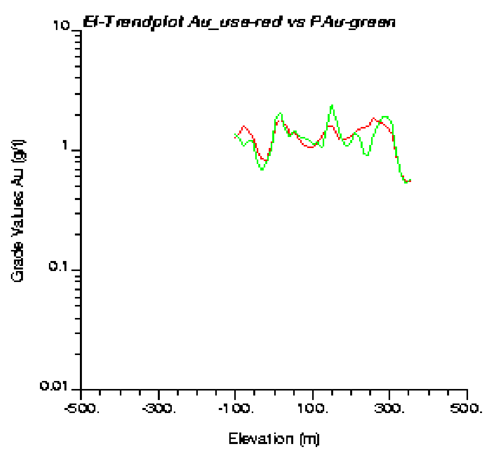




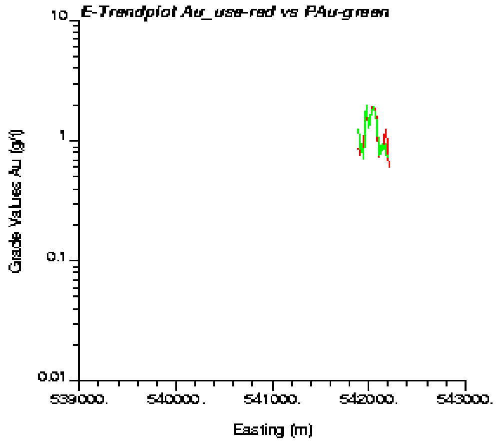
DonlinCreek Trend Plots for rck03_rdf

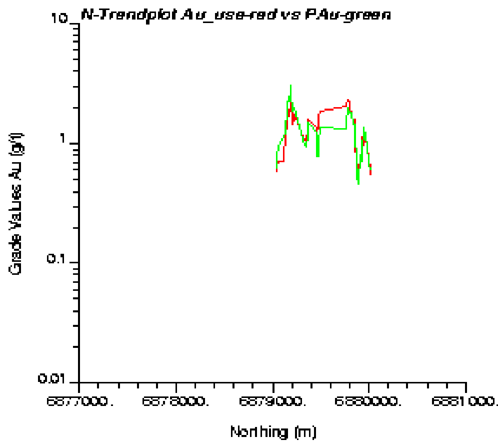


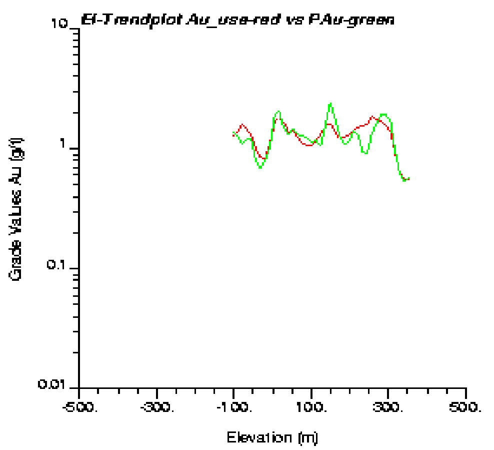




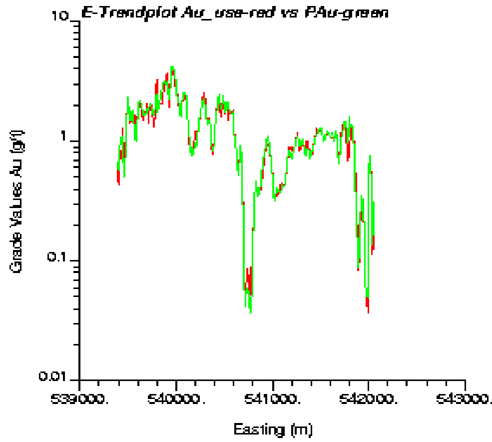
DonlinCreek Trend Plots for rck04_rdx

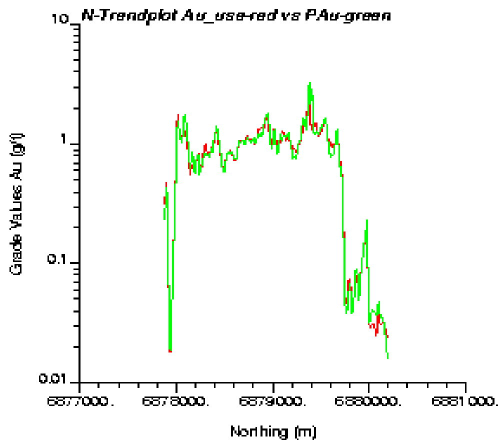


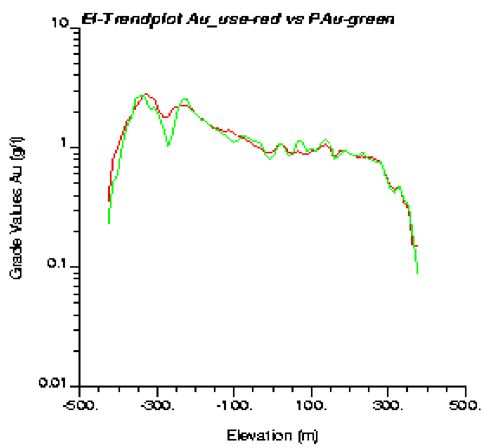




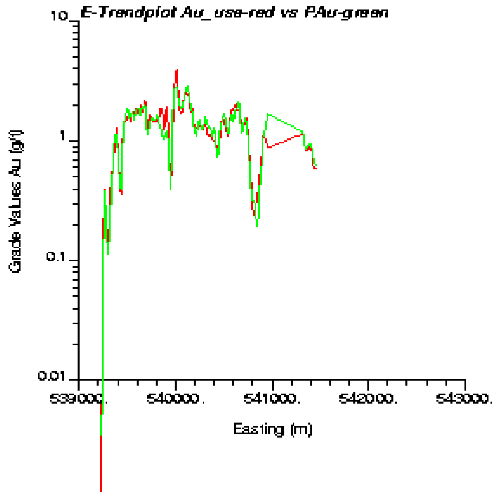
DonlinCreek Trend Plots for rck05_rdxb

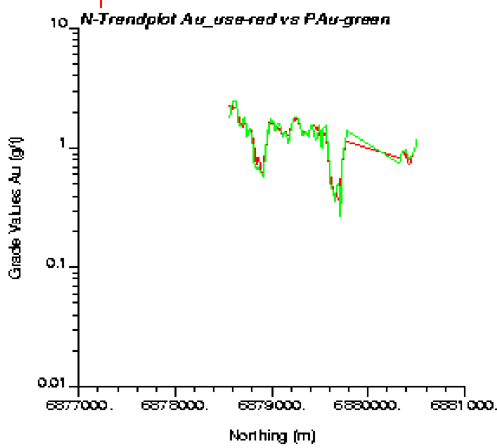


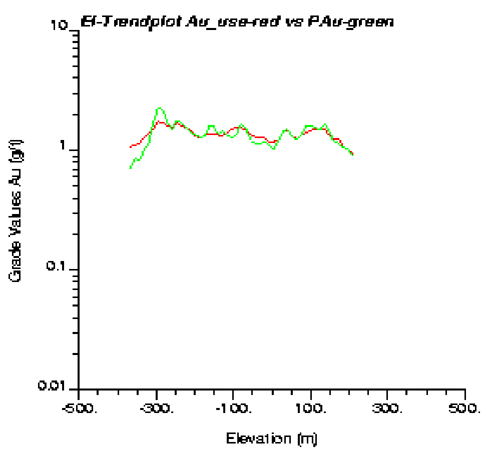




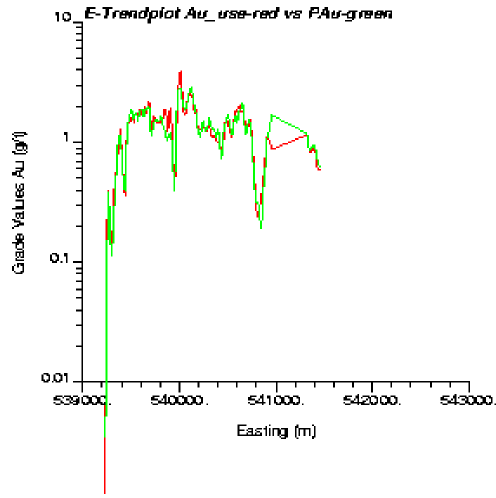
DonlinCreek Trend Plots for rck06_rdxl

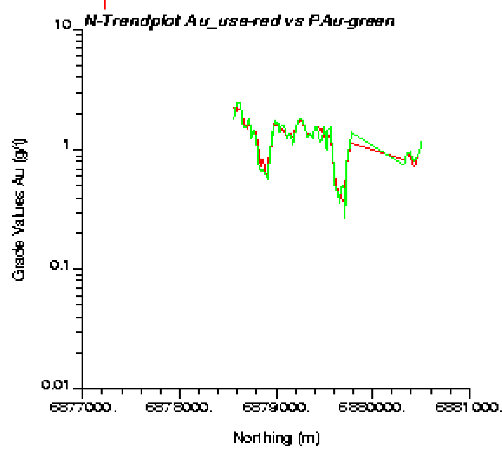


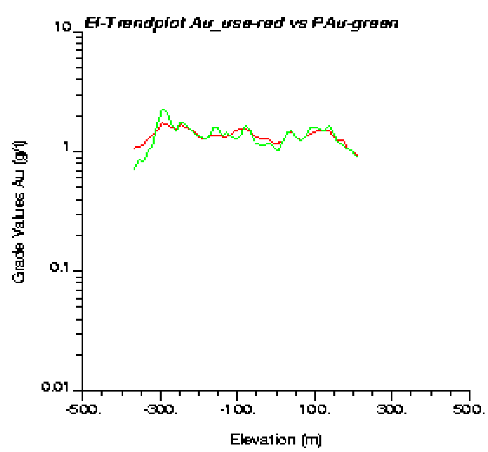




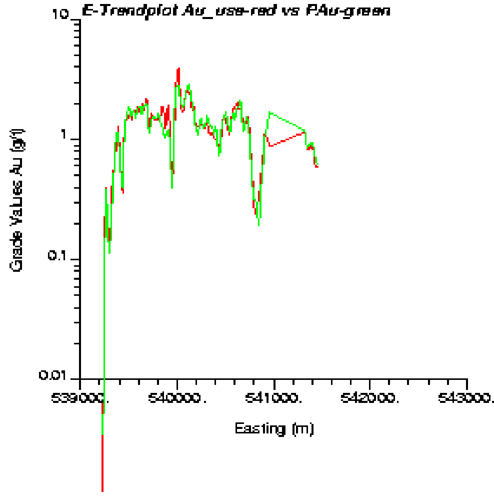
DonlinCreek Trend Plots for rck07_shale

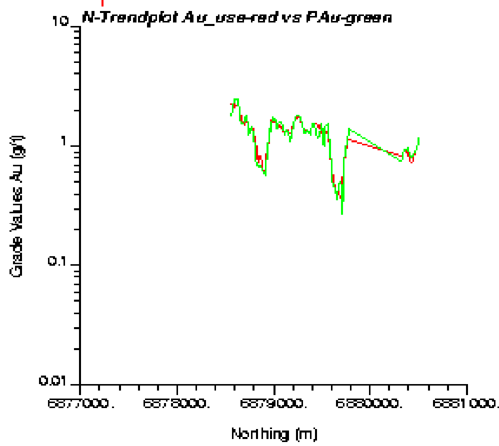


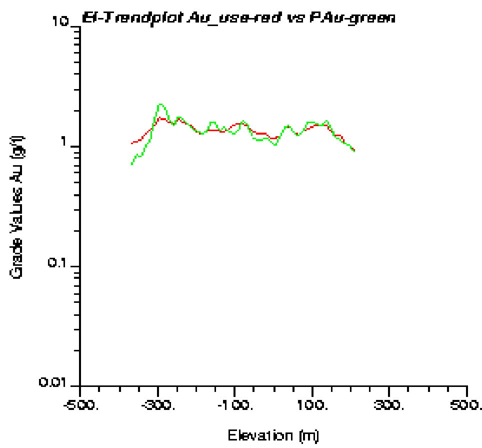




DonlinCreek Trend Plots for rck08_gwk

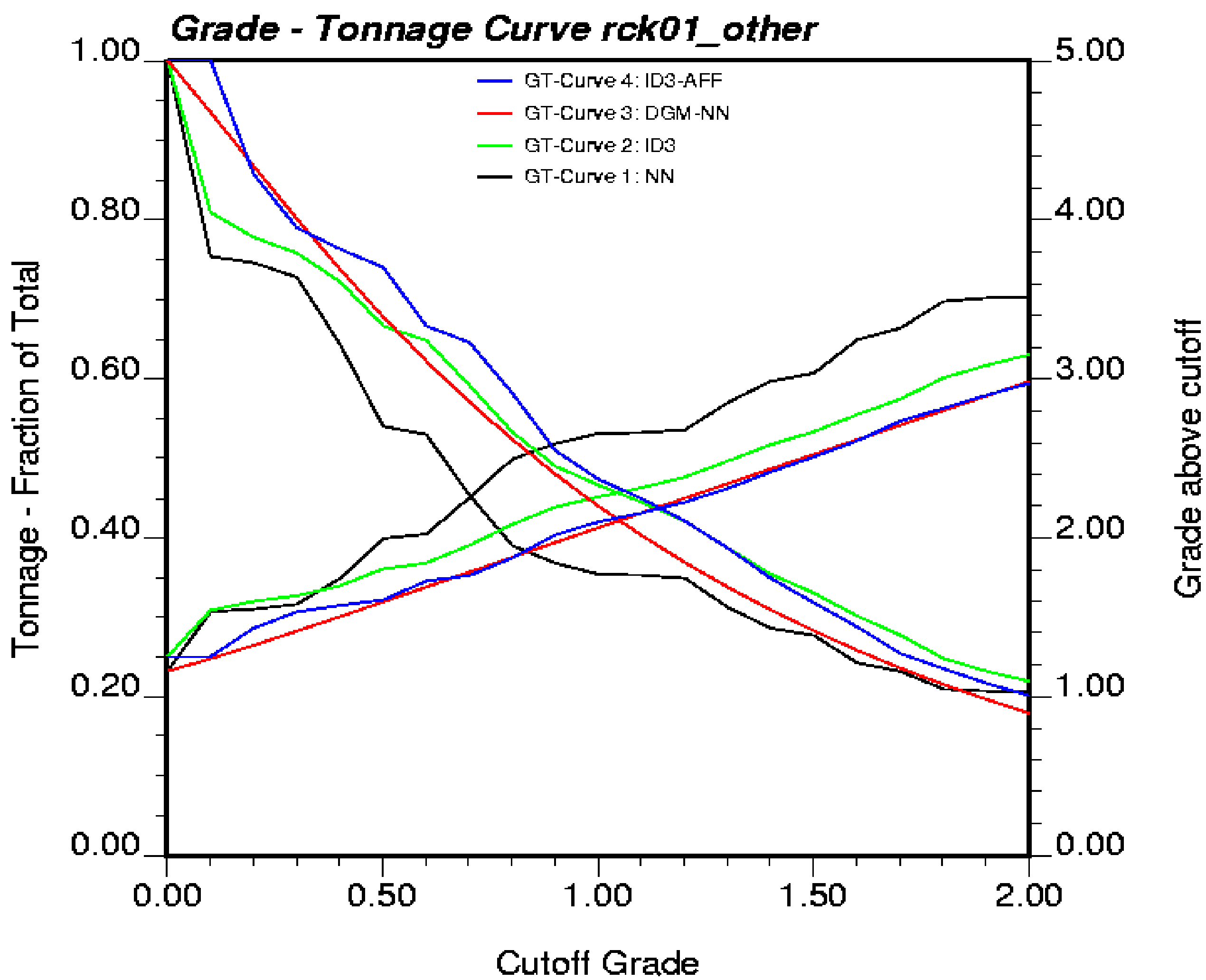




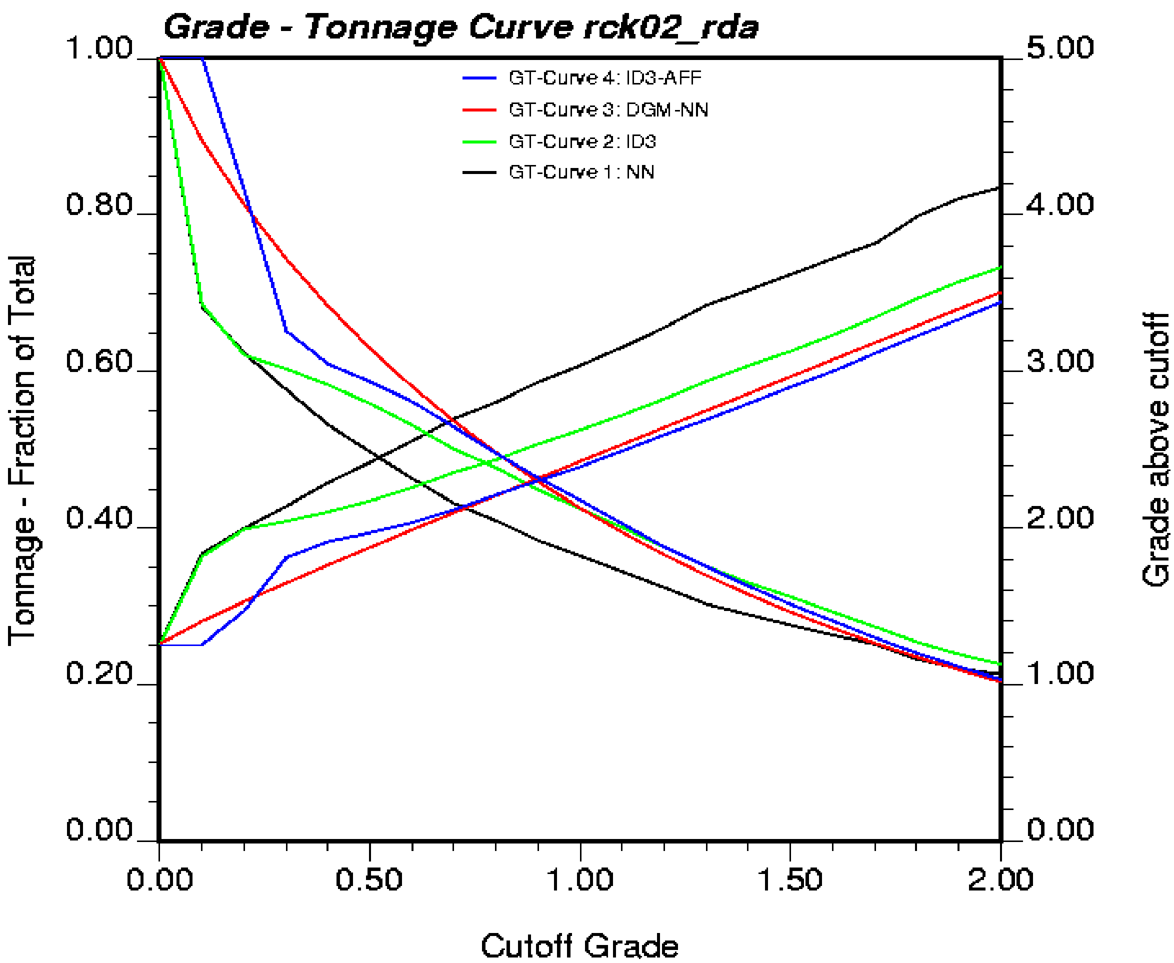


APPENDIX E

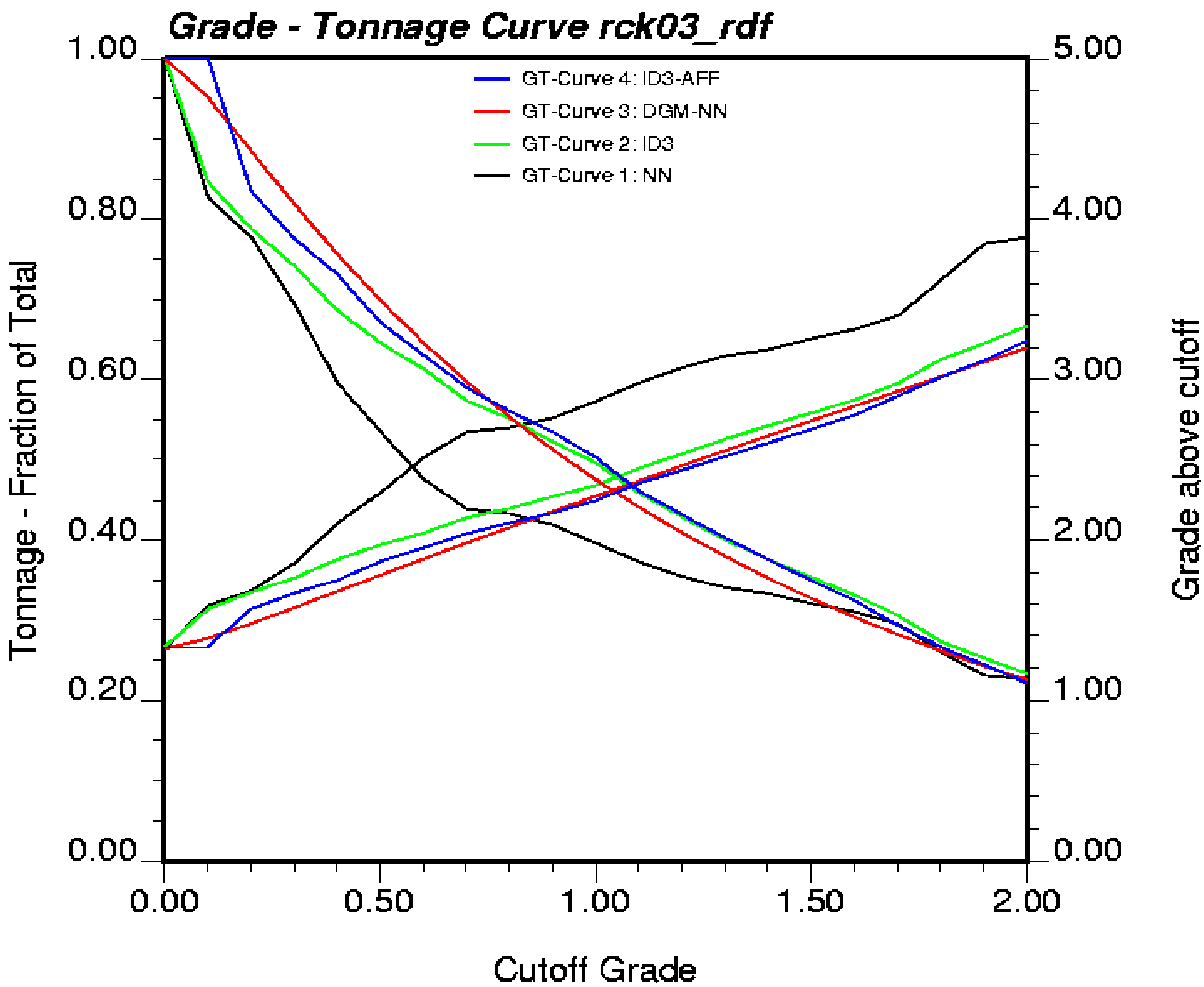
Change of Support Curves by Rock Type



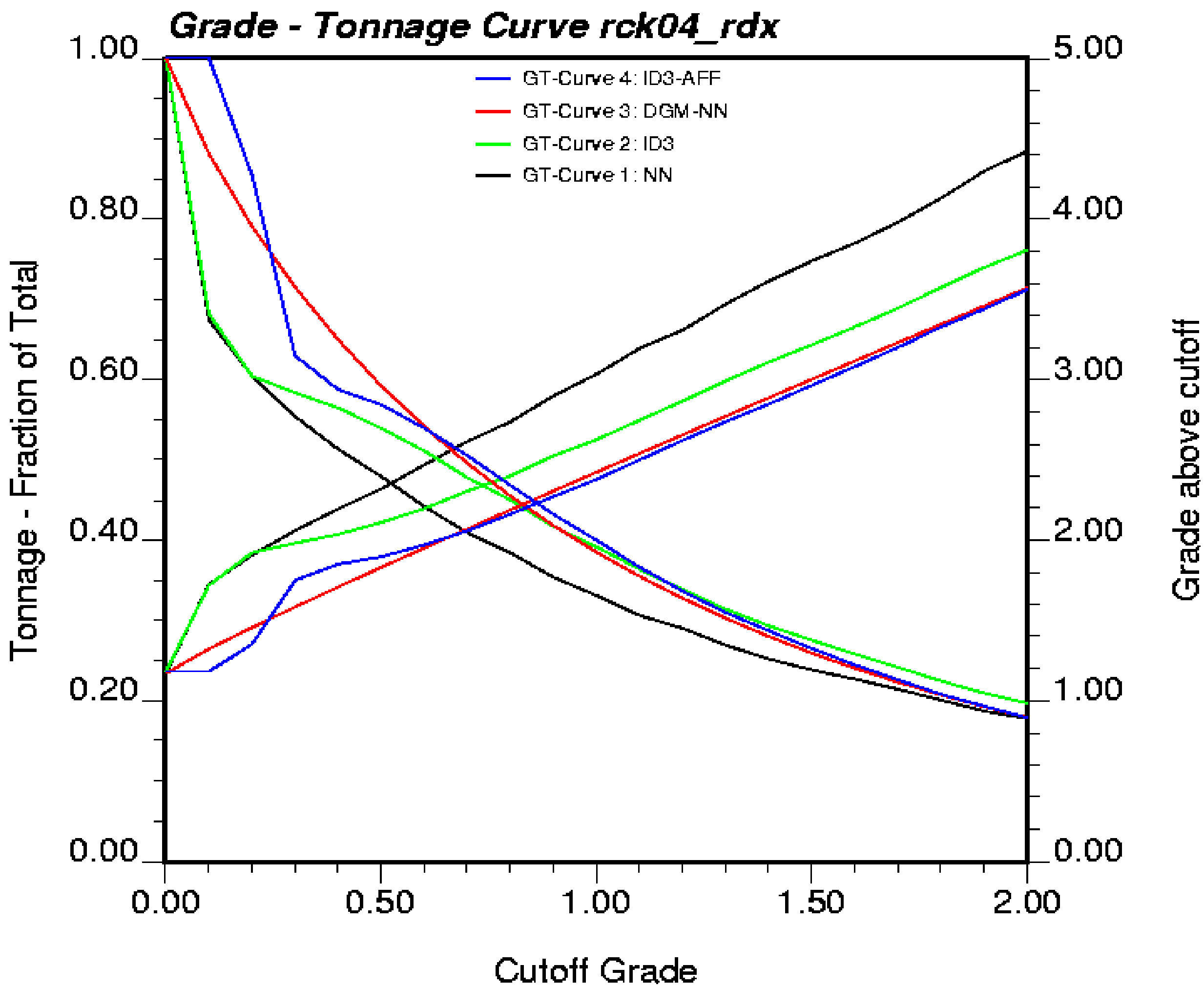
Grade - Tonnage Curve rck01_other
GT-Curve 4: ID3-AFF
GT-Curve 3: DGM-NN
GT-Curve 2: ID3
GT-Curve 1: NN
Tonnage - Fraction of Total
Grade above cutoff
Cutoff Grade
1.00
0.80
0.60
0.40
0.20
0.00
5.00
4.00
3.00
2.00
1.00
0.00
0.00
0.50
1.00
1.50
2.00

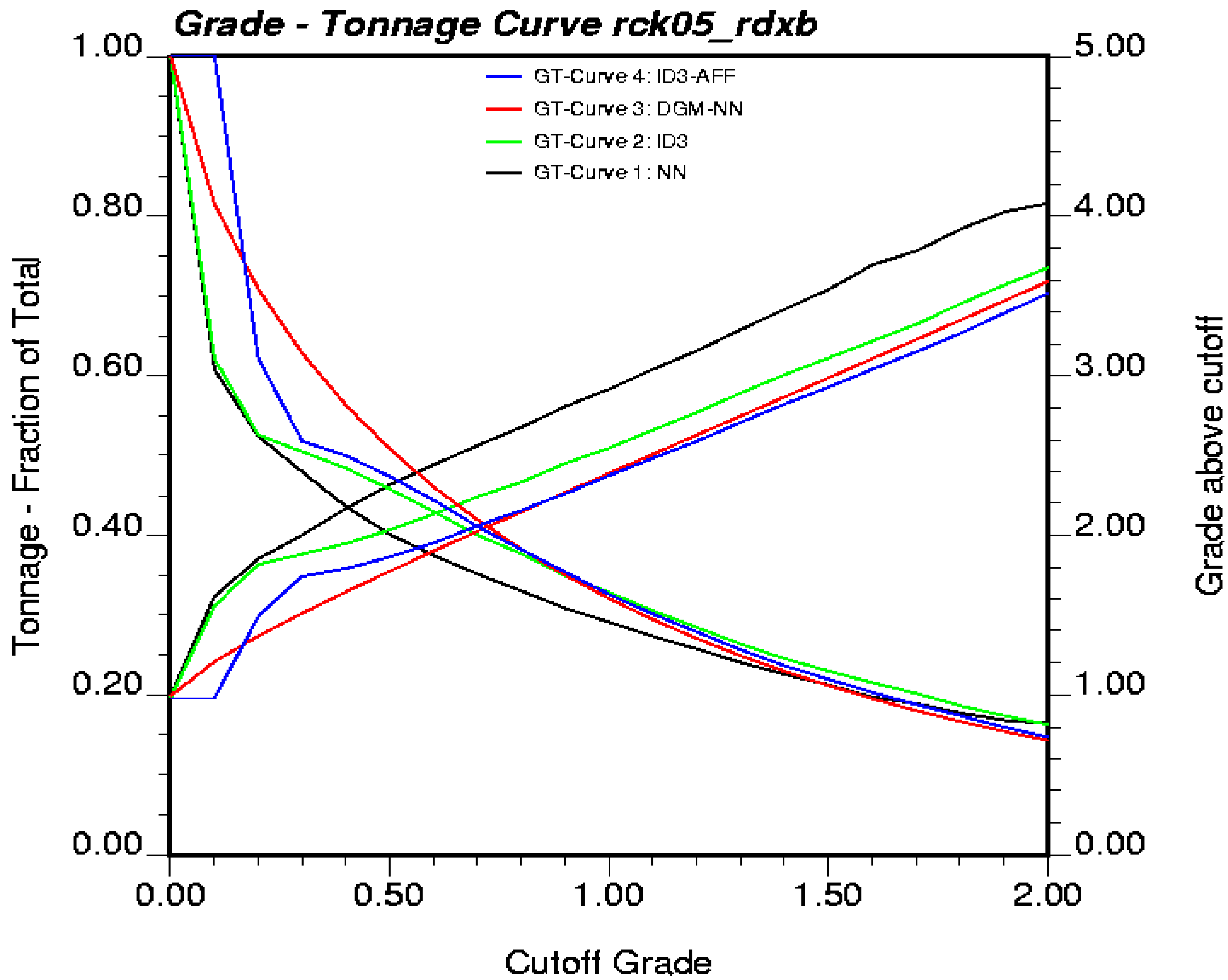


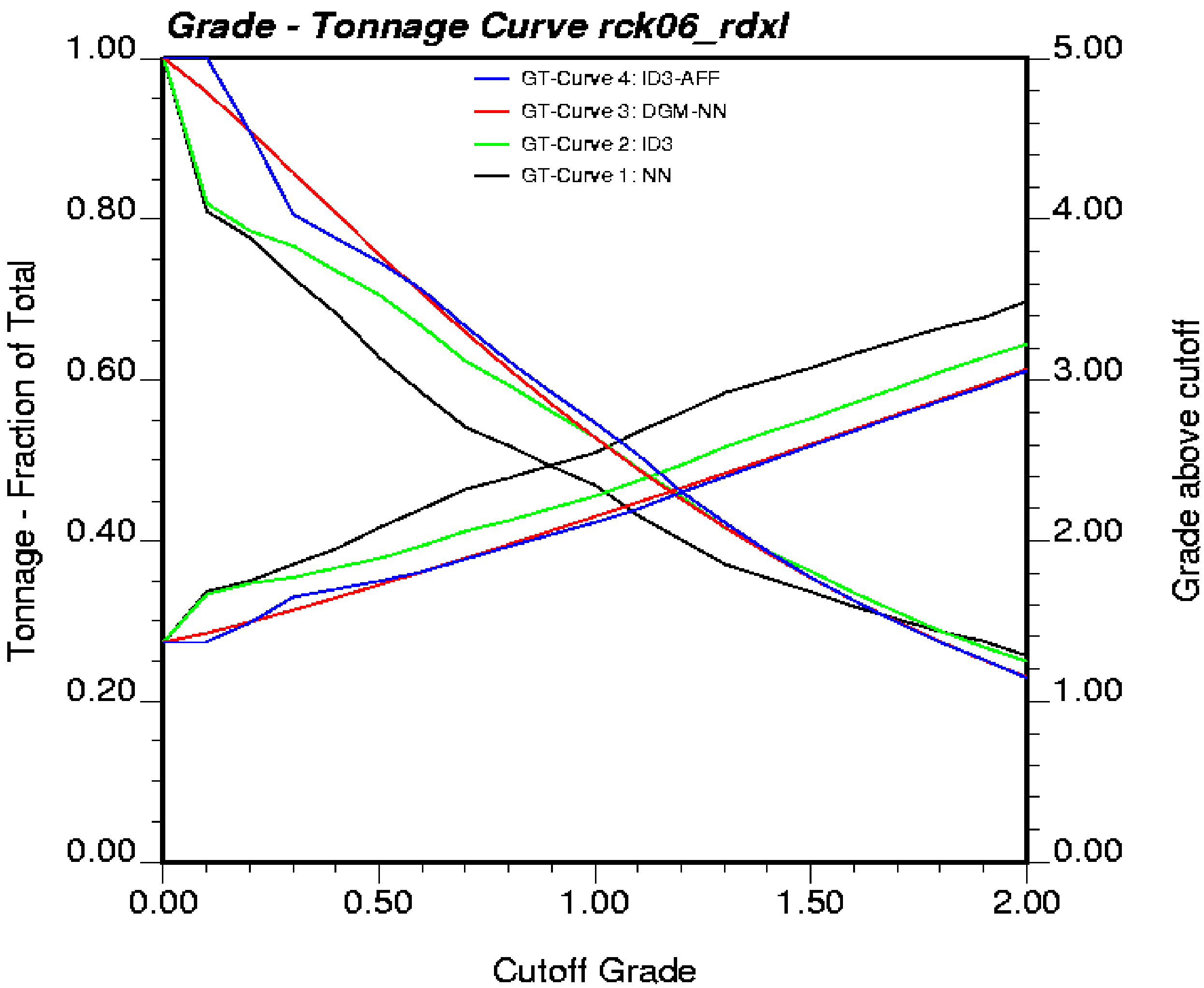
Grade - Tonnage Curve rck02_rda
GT-Curve 4: ID3-AFF
GT-Curve 3: DGM-NN
GT-Curve 2: ID3
GT-Curve 1: NN
Tonnage - Fraction of Total
Grade above cutoff
Cutoff Grade
1.00
0.80
0.60
0.40
0.20
0.00
5.00
4.00
3.00
2.00
1.00
0.00
0.50
1.50
2.00



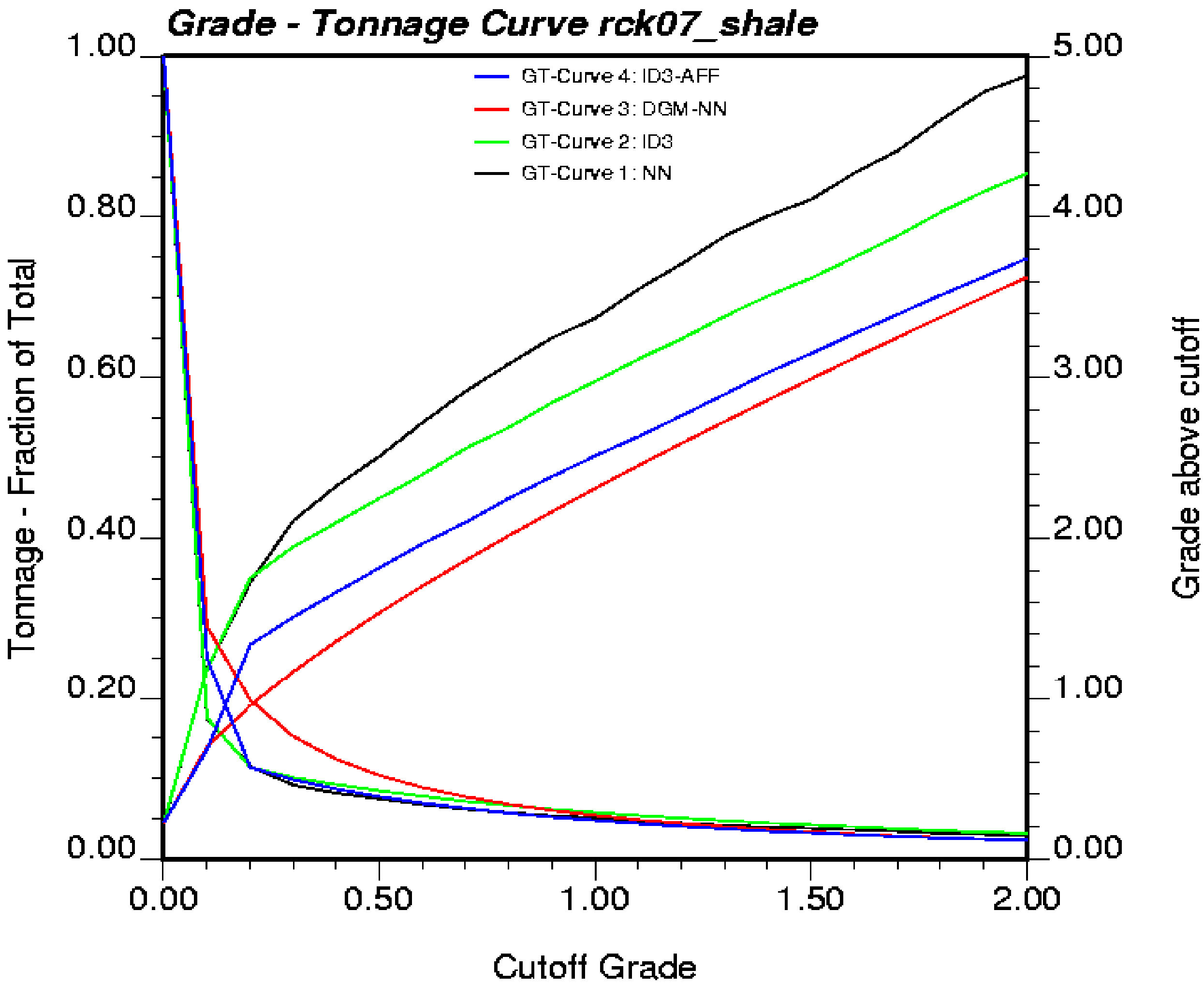
Grade - Tonnage Curve rck03_rdf
GT-Curve 4: ID3-AFF
GT-Curve 3: DGM-NN
GT-Curve 2: ID3
GT-Curve 1: NN
Tonnage - Fraction of Total
Grade above cutoff
Cutoff Grade
1.00
0.80
0.60
0.40
0.20
0.00
5.00
4.00
3.00
2.00
1.00
0.00
0.00
0.50
1.00
1.50
2.00


Grade - Tonnage Curve rck04_rdx
GT-Curve 4: ID3-AFF
GT-Curve 3: DGM-NN
GT-Curve 2: ID3
GT-Curve 1: NN
Tonnage - Fraction of Total
Grade above cutoff
Cutoff Grade
1.00
0.80
0.60
0.40
0.20
0.00
5.00
4.00
3.00
2.00
1.00
0.00
0.00
0.50
1.00
1.50
2.00

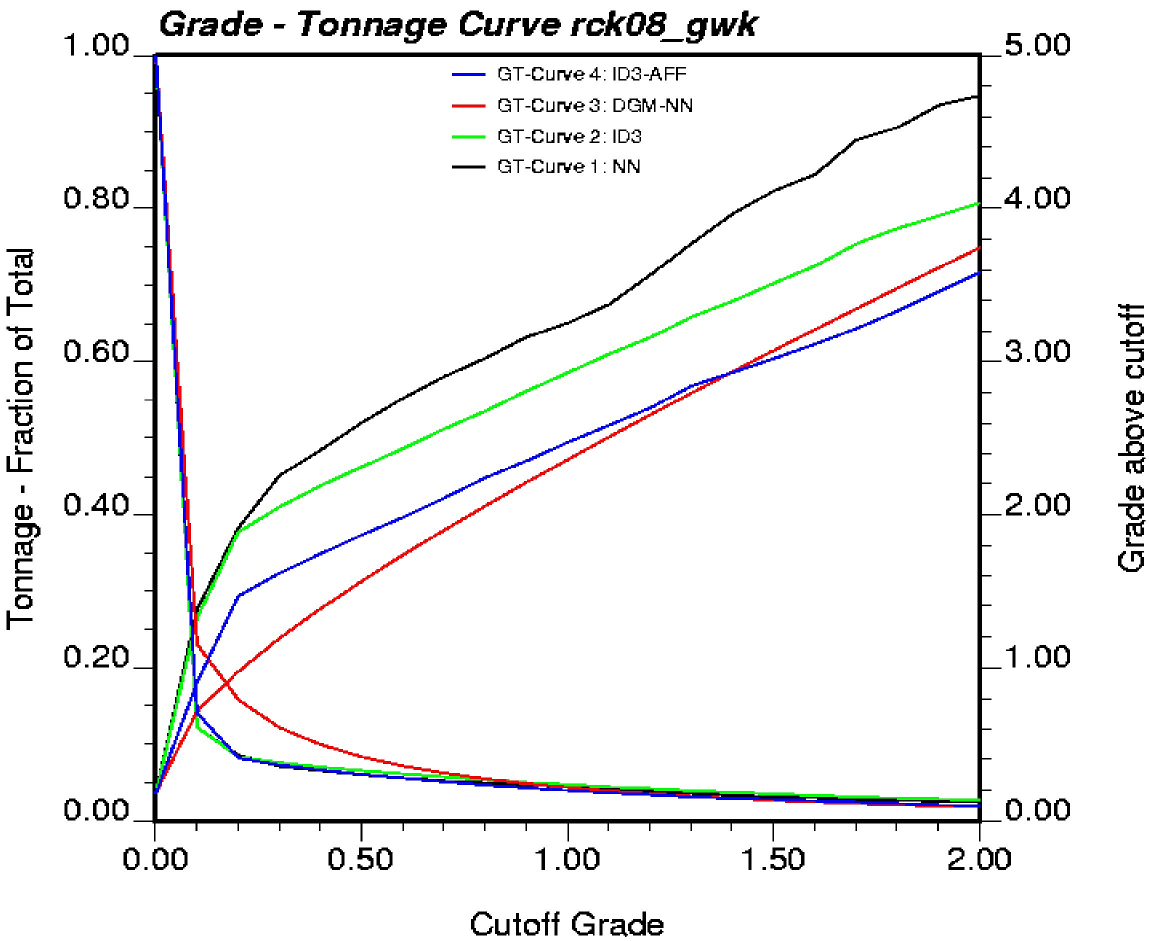

Grade - Tonnage Curve rck05_rdxb
GT-Curve 4: ID3-AFF
GT-Curve 3: DGM-NN
GT-Curve 2: ID3
GT-Curve 1: NN
Tonnage - Fraction of Total
Grade above cutoff
Cutoff Grade
0.00
0.20
0.40
0.60
0.80
1.00
1.00
2.00
3.00
4.00
5.00
0.50
1.50
2.00



Grade - Tonnage Curve rck06_rdxl
GT-Curve 4: ID3-AFF
GT-Curve 3: DGM-NN
GT-Curve 2: ID3
GT-Curve 1: NN
1.00
0.80
0.60
0.40
0.20
0.00
5.00
4.00
3.00
2.00
1.00
0.00
0.00
0.50
1.00
1.50
2.00
Tonnage - Fraction of Total
Grade above cutoff
Cutoff Grade



Grade - Tonnage Curve rck07_shale
GT-Curve 4: ID3-AFF
GT-Curve 3: DGM-NN
GT-Curve 2: ID3
GT-Curve 1: NN
Tonnage - Fraction of Total
Grade above cutoff
Cutoff Grade
1.00
0.80
0.60
0.40
0.20
0.00
5.00
4.00
3.00
2.00
1.00
0.00
0.00
0.50
1.00
1.50
2.00



Grade - Tonnage Curve rck08_gwk
GT-Curve 4: ID3-AFF
GT-Curve 3: DGM-NN
GT-Curve 2: ID3
GT-Curve 1: NN
Tonnage - Fraction of Total
Grade above cutoff
Cutoff Grade
1.00
0.80
0.60
0.40
0.20
0.00
5.00
4.00
3.00
2.00
1.00
0.00
0.00
0.50
1.00
1.50
2.00