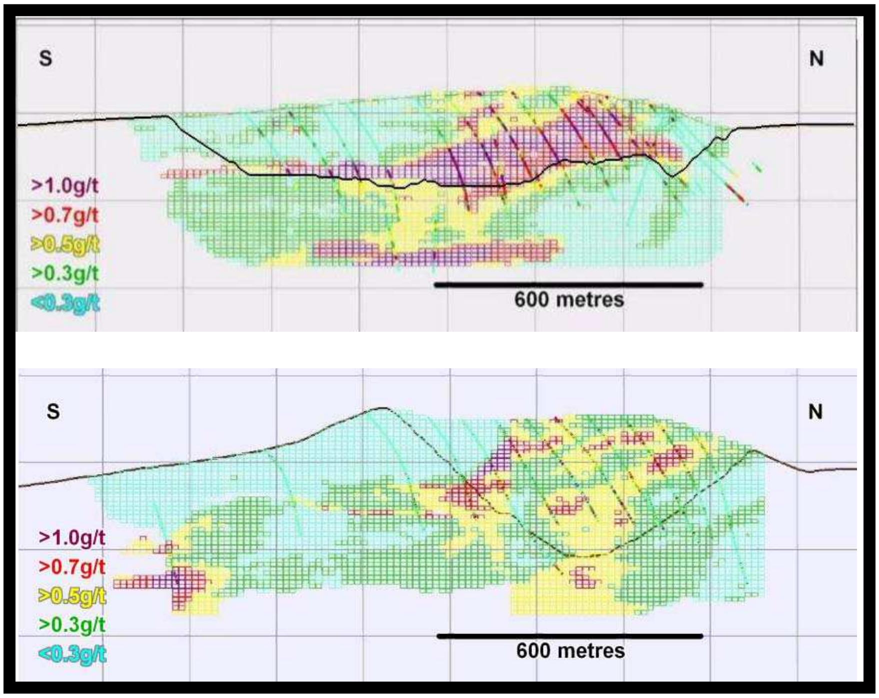

OCTOBER 2009
SUMMARY REPORT ON THE LIVENGOOD PROJECT, TOLOVANA DISTRICT, ALASKA

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1.0 Summary

The Livengood property is the focus of ongoing exploration by International Tower Hill Mines Ltd. ("ITH"). To date, 311 diamond and reverse circulation holes have been drilled on the property, and provide the basis for reporting a gold resource estimated at approximately 8.1Moz indicated and approximately 4.4 Moz inferred at a 0.50 g/t Au cutoff (ITH, 2009). This report is the sixth in a series of technical reports and the fifth in support of a resource estimate that provides documentation of the geological, operational, and resource estimation procedures that have been undertaken by ITH as they continue to advance this project. The currently reported estimate includes mineralization in the Core Zone and the new Sunshine and Northeast zones. The Core Zone has comprised the main part of previous estimates.

This report also includes for the first time, results from an initial Preliminary Economic Assessment (PEA). Using Whittle® software, with inputs of recovery and cost estimates, a set of optimized pits for a heap leach operation has been developed. Using the $700/oz Whittle optimized pit shell as a base case the economic model was developed by making scheduling adjustments to establish a constant stripping rate and setting minimum cutoff grades for each of the 21 ore types in the model. The model was then run using a base case assuming an $850 gold price and that 100,000 ore tonnes per day were sent to a heap leach pad. The result is an In-Pit resource of 308Mt at 0.68 g/t Au (~6.7 M oz) and 132Mt at 0.71g/t Au (~3.0 M oz) indicated and inferred respectively recovering 459K ounces of gold per year for 12.6 years yielding a total of 5.8M oz. The model gives an overall strip ratio of 0.78:1 (waste to ore), an average recovery of 60% with a total cost per ounce of $533. Key cost inputs were a mining cost of $1.80/tonne and processing costs of $3.80/tonne and G&A of $0.6/tonne. A financial analysis was then done assuming an initial capital investment of $665M with sustaining capital of $297M. The model has an IRR of 14.6% and an NPV of $440M assuming a 5% discount rate. It is believed that these results are based on reasonable economic estimates and assumptions that are sufficient to demonstrate reasonable prospects for economic extraction. These results are preliminary in nature and include inferred resources that are considered too speculative geologically to have the economic considerations applied to them and there is no certainty that this preliminary assessment will be realized.

This work is based partly on cost estimates and initial metallurgical test work on gold recovery. ITH envisions a combined heap leach and mill recovery system for optimized gold recovery. Tests are currently underway to determine what recoveries might be expected from the mill so that this can be modeled. However, at this time, insufficient information is available to ITH to adequately model an optimized combination heap leach and mill scenario. Therefore, the results reported here are for a heap leach-only option.

Initial metallurgical test work has been completed on the main Livengood mineralization types and indicates that the oxidized and partially oxidized materials respond well to simple cyanidation. Shale-hosted mineralization responds well to cyanide leaching but will require CIP processing due to the presence of preg-robbing carbon. Ore blocks containing shale were treated as waste in the economic model. Most of the mineralization also displays good results when subjected to gravity separation techniques. A more extensive metallurgical test work program for the Livengood mineralization is currently underway to more completely identify the mineral processing parameters required to treat the various Livengood mineralization types.

This initial PEA for the Livengood deposit has demonstrated that the project has strong potential based on a heap leach only option investigated. It is important to note that this PEA covers only the heap leach, oxide component of the October 13, 2009 resource with approximately 40% of the deposit (un-oxidized mineralization) excluded pending the possible addition of a milling circuit. Milling metallurgical test work currently in progress suggests that high gold recoveries can be obtained from all ore types utilizing an initial gravity concentration circuit followed by standard CIP processing of the tails.

The Livengood property is located approximately 115 km northwest of Fairbanks, Alaska in the Tolovana mining district within the Tintina Gold Belt. The project area is centered on a local high point named Money Knob. This feature and the adjoining ridge lines have been considered by many to be the lode gold source for the Livengood placer deposits which lie in the adjacent valley to the north where they have been actively mined since 1914 with production of more than 500,000 ounces of gold.

The property has been prospected and explored by several companies and private individuals since the 1970's. Geochemical surveys by Cambior in 2000 and AngloGold Ashanti (U.S.A.) Exploration Inc. ("AGA") in 2003 and 2004 outlined a 1.6 x 0.8 km area with anomalous gold in soil. Scattered anomalous samples continue along strike for an additional 2 km to the northeast and 1.6 km to the southwest. Eight reverse circulation holes were drilled by AGA in 2003 and a further 4 diamond core holes were drilled in 2004 to evaluate this anomaly. Favourable results from these holes revealed wide intervals of gold mineralization (BAF-7: 138.7m @ 1.07 g/t Au; MK-04-03: 55.3m @ 0.51 g/t Au) along with lesser intervals over a broad area. Over the past 4 years, exploration by ITH through its wholly owned Alaskan subsidiary, Talon Gold Alaska, Inc., has been aimed at assessing this area of mineralization through drilling diamond core and reverse circulation holes.

ITH controls 100% of its 44 square kilometre Livengood land package, which is primarily made up of fee land leased from the Alaska Mental Health Trust and a number of smaller private mineral leases.

Rocks at Livengood are part of the Livengood Terrane, an east–west belt, approximately 240 km long, consisting of tectonically interleaved assemblages of various ages. These assemblages include the Amy Creek Assemblage, which is a sequence of latest Proterozoic and early Paleozoic basalt, mudstone, chert, dolomite, and limestone. In thrust contact above the Amy Creek Assemblage lies an early Cambrian ophiolite sequence of mafic and ultramafic sea floor rocks. Structurally above these rocks lies a sequence of Devonian shale, siltstone, conglomerate, volcanic, and volcaniclastic rocks which are the dominant host to the mineralization currently under exploration at Livengood. The Devonian assemblage is overthrust by more Cambrian ophiolite rocks. All of these rocks are intruded by Cretaceous multiphase monzonite, diorite, and syenite stocks, dikes, and sills. Gold mineralization is believed to be related to this intrusive event.

Gold mineralization occurs in two styles: as multistage fine quartz veins occurring in all lithologies (commonly near intrusive dikes and sills), and as diffuse mineralization within volcanic, intrusive, sedimentary, and mafic-ultramafic rocks without a clear quartz vein association. Four principal stages of alteration are currently recognized. These are an early biotite stage followed by albite-black quartz, followed by a sericite-quartz, and finally a carbonate stage. Arsenopyrite apparently has been introduced during all stages, and gold correlates strongly with arsenopyrite, but it is not clear whether gold was introduced during all four stages or preferentially during one or more stages.

Mineralization is interpreted to be intrusion-related, consistent with other gold deposits of the Tintina Gold Belt, and has a similar As-Sb geochemical association. Thrust-fold architecture is apparently key to providing pathways for magma (dikes and sills) and hydrothermal fluid.

Local fault and contact limits to mineralization have been identified, but overall the deposit has not been closed off in any direction. The current resource and area drilled represents only about 25% of the total area which shows anomalous gold in surface soil samples.

2.0 Introduction and Terms of Reference

2.1 Introduction

Mineral Resource Services Inc. ("MRS"), Reserva International ("RI") and Pennstrom Consulting Inc. ("PCI") have been requested by International Tower Hill Mines Ltd. ("ITH") to provide an independent technical report on the Livengood gold project in the Tolovana Mining District of Interior Alaska. The Livengood property is currently being explored by ITH through its wholly-owned subsidiary, Talon Gold Alaska, Inc. ("TGA").

This report on the Livengood project presents an updated resource estimate and Preliminary Economic Assessment (PEA) prepared by RI, initial metallurgical and capital cost findings prepared by PCI, and updated geological and project information described by MRS. The resource estimate is based on drill hole and surface data through September 25, 2009. Each author is a Qualified Person and is responsible for various sections of this report according to their expertise and contribution. Dr. Klipfel of MRS is responsible for all sections of this report except sections 16 and 17 as well as compilation of information. Mr. Willian Pennstrom Jr. is solely responsible for section 16, except section 16.11. Mr. Timothy Carew is solely responsible for section 16.11 and section 17. Each author has contributed figures, tables, and portions of section 1 based on their respective contributions to this report.

The work presented here builds on and revises previous geologic, metallurgical and resource information reported in six previous technical reports for the project (Klipfel, 2006; Klipfel and Giroux, 2008a; Klipfel, Giroux and Puchner 2008; Klipfel and Giroux, 2008b; Klipfel and Giroux, 2009; Klipfel, Barnes, and Pennstrom, 2009). Gold assays and analyses of other elements along with geological, structural, engineering, metallurgical data is from 311 holes drilled by ITH and previous explorers, including 99 RC holes and 6 diamond core holes drilled between June 1and September 6, 2009 as well as data from previous drilling programs.

Information presented in this report is based on data provided to MRS, RI, and PCI by ITH as of September 25, 2009. Data generated prior to 2006 was provided to ITH by AngloGold Ashanti (U.S.A.) Exploration Inc. ("AGA"). This report also relies on personal observations made by:

- Paul Klipfel in the course of six field visits.
- Tim Carew in the course of two site visits and generation of modelling data from primary data provided by ITH.
- Bill Pennstrom, who made one site visit Livengood and one visit to Fort Knox to identify operating costs at that mine.
- and on general geologic information available to the public through peer review journals as well as publications by the U.S. Geological Survey and agencies of the State of Alaska.

2.2 Terms of Reference

Dr. Paul Klipfel of MRS, in Reno, Nevada, Mr. Tim Carew, of RI in Reno Nevada, and Mr. William Pennstrom Jr. of PCI in Denver, were commissioned by ITH to prepare this report on the Livengood project. This report is based on data generated and results received by ITH through September 25, 2009 and is in support of resource information released to the public on October 13, 2009. The 2009 summer drilling program is mostly completed however drilling and results received after October 25th are not utilized in this report.

Dr. Klipfel, Mr. Carew, and Mr. Pennstrom are independent consultants and are Qualified Persons (QP) for the purposes of this report as defined by Canadian Securities Administrators National Instrument 43-101 ("NI 43-101").

2.3 Glossary of Key Abbreviations

ADNR	Alaska Department of Natural Resources
AGA	AngloGold Ashanti (U.S.A.) Exploration Inc.
AMHLT	Alaska State Mental Health Land Trust
BES	Barnes Engineering Services, Inc
BLM	U.S. Bureau of Land Management
g/t	grams/tonne
IRGS	Intrusion Related Gold System
ITH	International Tower Hill Mines Ltd.
KWh/T	kilowatt-hours per Ton
M	million
MRS	Mineral Resource Services Inc.
MTpa	million tonnes per annum
MW	megawatts
Opt	troy ounces per Ton
oz(s)	troy ounce(s)
PEA	Preliminary Economic Analysis
PCI	Pennstrom Consulting Inc.
QA/QC	Quality Assurance/Quality Control
QP	qualified person
ROM	run of mine
t	tonne
TGA	Talon Gold Alaska, Inc.
tpa	tonnes per annum
tpd	tonnes per day
tph	tonnes per hour
USACE	US Army Corps of Engineers
$ or USD	United States dollars

2.4 Purpose of Report

The purpose of this report is to provide an independent evaluation of the Livengood project, its exploration history, resource and mine development potential based on exploration work through September 25, 2009, a resource assessment based on that data, the discovery opportunity based on known geology and current exploration results, and to provide recommendations for future work. This report conforms to the guidelines set out in NI 43-101.

2.5 Sources of Information

Information for this report was provided to the authors by ITH and consists of data generated by ongoing exploration by ITH and initial data from 2006 and earlier which was provided to ITH by AGA. In addition, Dr. Klipfel has spent an aggregate of twenty two days on the site during six visits reviewing core, examining outcrop, and discussing the project with on-site geologic staff and with Mr. Jeffrey Pontius, President of ITH. In addition, Dr. Klipfel has undertaken independent petrographic evaluation of samples from the project.

Drilling, sampling, QA/AC control, logging and sampling, and other exploration activities have been performed by contract geologic staff under the direction of Dr. Russell Meyers, Ph.D. (ITH VP Exploration) and Mr. Chris Puchner M.Sc. (ITH Chief Geologist; AIPG CPG 07048). Both persons are Qualified Persons as per guidelines set out in NI 43-101. Support for logistics, surveying, camp management, and digital modeling have been provided by Northern Associates of Alaska Inc. and their geologic, survey, and IT staff. External consultants and engineering firms have been contracted for numerous functions including Giroux Consultants Ltd. of Vancouver, B.C., (previous resource evaluations), Barnes Engineering Services (previous resource evaluation), Mineral Resource Services, Inc. (petrographic evaluation), Three Parameters Plus, Inc. (environmental studies), Northern Land Use Research Inc. (archaeological surveys), ABR Inc. (environmental studies), Kappes Cassiday and Associates, (metallurgical test work), and Hazen Research Inc. (metallurgical test work).

Gold assay and multi-element ICP data from drill hole samples used in the resource evaluation are from ALS-Chemex. ALS-Chemex operates to international quality standards including compliance with ISO 17025 (www.ALSglobal.com). The ALS Chemex analyses have been validated annually through cross-lab checks using SGS, ACT Labs, and Alaska Assay Laboratories. Florin Analytical Services LLC. have provided analytical services for test work done by Kappes Cassiday. SRK has contributed to cost estimates.

2.6 Field Examination

Dr. Klipfel has visited the property six times, with the most recent visit from October 5th through October 8th, 2009. These visits included sequential updating of data, exploration activities, review of geologic sections, and interpretations of geologic staff. Visits also included review of the physiographic, geologic and tectonic setting of the property, drill hole collar locations, surface and down-hole survey procedures, core orientation procedures as well as detailed examination of outcrop, drill core and RC chips. Previous visits were during the following periods: June 17-24, 2009, September 22 – 26, 2008, June 30 – July 3, 2008, October 4-5, 2007, and June 6-7, 2006. Independent check samples were collected during each of these visits and are described further in section 14.

Tim Carew has visited Livengood for a total of 9 days on two separate trips in October, 2009. During the course of these visits, modelling work was conducted collaboratively with ITH geologic staff, database information and contained data were reviewed and validated.

Mr. Pennstrom spent two days on site in May of 2009. Site characteristics were reviewed with ITH staff.

3.0 Reliance on Other Experts

The preparation of this report has relied upon public and private information gathered independently by the authors and data provided by ITH and AGA regarding the property. The authors assume and believe that the information provided and relied upon for preparation of this report is accurate and that interpretations and opinions expressed in them are reasonable and based on current understanding of mineralization processes and the host geologic setting. The authors have used this information to develop their own opinions and interpretations along with external and independent understanding of geologic, metallurgical processing, and resource evaluation concepts and best practices. The authors have endeavoured to be diligent in their examination of the data provided by ITH and the conclusions derived from review of that information or generated using that information.

4.0 Property Description and Location

4.1 Area and Location

The Livengood project is located approximately 115 km by road (85 km by air) northwest of Fairbanks in the northern part of the Tintina Gold Belt (**Figure 4.1**). At this location, the property straddles, but lies predominantly to the north of, the Elliott Highway, the main road connecting Fairbanks with the Alaskan far north. The property lies in numerous sections of Fairbanks Meridian Township 8N and Ranges 4W and 5W. Money Knob, the principal geographic feature within the area being explored, lies near the center of the claim block and is located at 65^{o}30’52’’N, 148^{o}27’50’’W.

The key area of interest and resource reported here lies on the northwest flank of Money Knob and is a zone of gold mineralization with, as yet, undetermined extent. This area lies within, and to the south of, a 1.6 x 0.8 km northeast-trending soil sample anomaly that was the initial target of interest for drill assessment. The surface geochemical anomaly is situated within in a broader area of less pronounced anomalism that extends a further 2 km to the northeast and 1.6 km to the southwest. This zone is described further in Section 9.0. Continued drilling success has lead to several rounds of resource evaluation, the latest of which is the subject of this report. At this time, mineralization continues to be identified as the area drilled expands outwards from an initial core zone centered over the geochemical soil anomaly. Identified mineralization has local boundaries such as faults or contacts, but overall, the limits of this mineralized system have not been identified with mineralization effectively open in all directions. The area with anomalous gold in soil samples has only been partially tested.

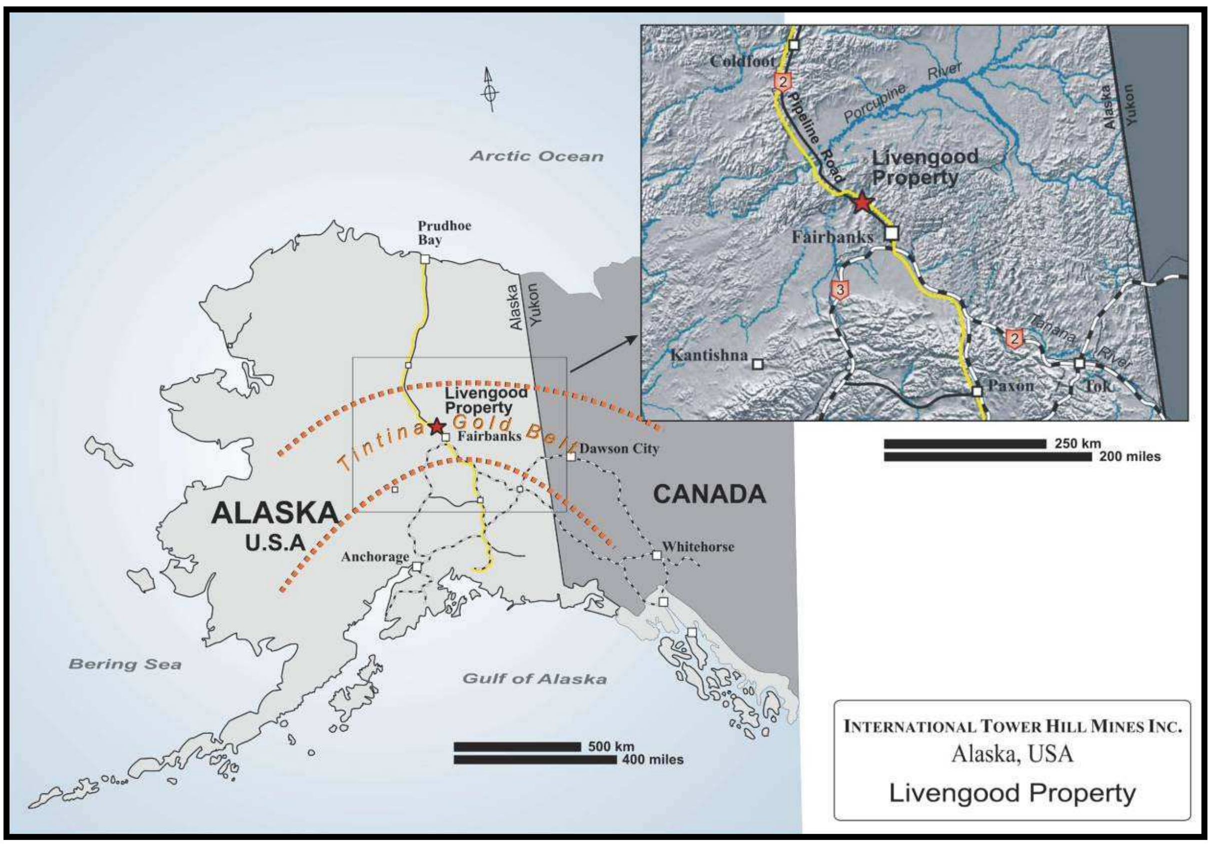

Figure 4.1. Location map showing the location of the Livengood project and the Tintina Gold Belt (orange dashed lines enclose the belt).

4.2 Claims and Agreements

The Livengood Property (**Figure 4.2**) consists of an aggregate area of approximately 10,593 acres (4,287 hectares) controlled through agreements between TGA and the State of Alaska as well as between TGA and various private individuals who hold state and federal patented and unpatented mining and placer claims. All property and claims controlled through agreements are summarized in **Table 4.1** and listed in **Appendix 1**. These agreements are with the AMHLT, Richard Hudson and Richard Geraghty, the estate of Ron Tucker, the Griffin heirs, and Karl Hanneman and the Bergelin Family Trust. The AMHLT Trust Land Office manages approximately 1 million acres of Alaska land through the Department of Natural Resources (www.mhtrust.org) and generates revenue for the AMHLT through land leasing and fees for a range of resources.

The AMHLT lease (#9400248), signed July 1, 2004 by AGA and assigned to TGA on August 4, 2006, includes advance royalty payments of $5/acre/year which escalates to $15/acre in years 4-6 and $25/acre in years 7-9. The lease has a work commitment of $10/acre in years 1-3, $20/acre in years 4-6, and $30/acre in years 7-9. The lease carries a sliding scale production royalty of 2.5% @ $300 gold up to 5% for a gold price more than $500. In addition, an NSR production royalty of 1% is payable to AMHLT with respect to the unpatented federal mining claims subject to the Hudson & Geraghty and the Hanneman and Bergelin Family Trust lease. AMHLT owns both the surface and subsurface rights to the land under lease to TGA.

The Hudson and Geraghty lease, signed April 21, 2003 by AGA and assigned to TGA on August 4, 2006, has a term of 10 years and for so long thereafter as exploration and mining operations continue. TGA is required to make advance royalty payments of $50,000 per year, which are credited to production royalties. Production royalties vary from 2% to 3%, depending upon the price of gold. TGA has the option to buy down 1% of the royalty for $1 million. The 20 claims under this lease are unpatented federal lode mining claims that have no expiry but require a claim maintenance fee of $140/claim/year to keep them in good standing.

The Tucker mining lease of two unpatented federal lode mining and four federal unpatented placer claims has an initial term of ten years, commencing on March 28, 2007 and for so long thereafter as mining related activities are carried out. The lease requires payment of advance royalties of $5,000 on or before March 28, 2009, $10,000 on or before March 28, 2010 and an additional $15,000 on or before each subsequent March 28 thereafter during the initial term (all of which minimum royalties are recoverable from production royalties). ITH is required to pay

the lessor the sum of $250,000 upon making a positive production decision. An NSR production royalty of 2% is payable to the lessor. ITH may purchase all interest of the lessor in the lease property (including the production royalty) for $1million. The 6 leased claims are federal claims without expiry. A fee of $140/claim/year or $140 worth of work/claim/year is required to maintain the claims in good standing.

The Griffin lease of three patented federal claims is for an initial term of ten years (commencing January 18, 2007), and for so long thereafter as the Company pays the lessors the minimum

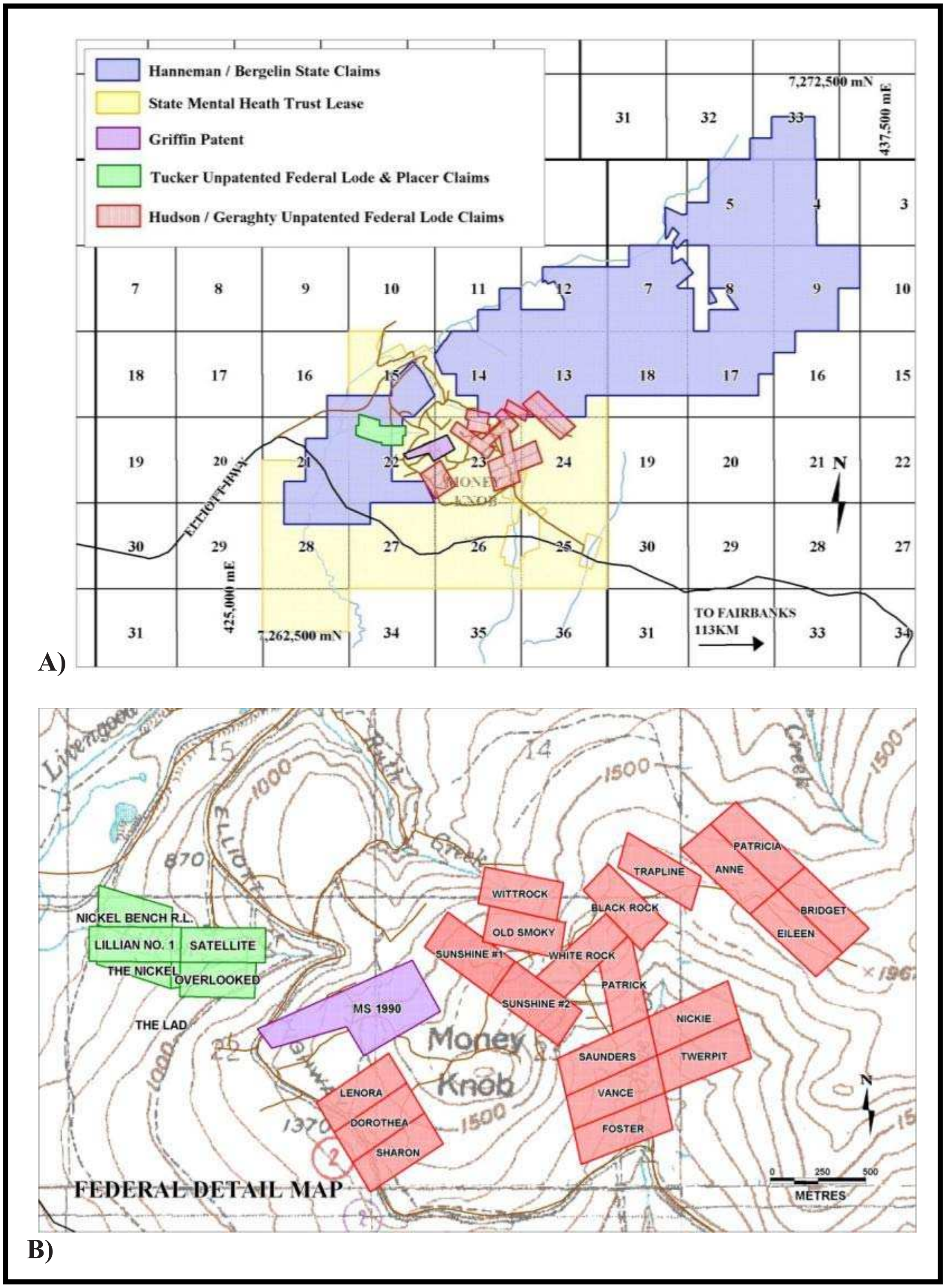

Figure 4.2. Claim map showing the Livengood land position. **A)** The AMHL Lease is shown in yellow and holdings belonging to other parties shown in respective colors. **B)** Detailed map of the individual claims within the AMHL Lease.

TABLE 4.1
SUMMARY OF CLAIM HOLDINGS AND ANNUAL OBLIGATIONS

Holder	Type of Holding	Current Year	2009 Holding Obligation
AMHLT	State Mining Lease	6	$91K advance royalty+$110K work expenditure
Hudson and Geraghty,	20 Fed. unpatented lode claims	7	$50K advance royalty payment
Ron Tucker (estate)	2 Fed. unpatented lode claims	3	$5K
	4 Fed. unpatented placer claims	3	
Griffin heirs	3 patented Fed. claims	3	$15K
Karl Hanneman and the Bergelin Family Trust	169 Alaska State mining claims	4	$50K + $200k work expenditure
Alaska State Lands	169 Alaska State mining claims	current	Annual work up to date to 2013

royalties required under the lease. The lease requires minimum royalty payment of $10,000 on or before January 18, 2009, $15,000 on or before January 18, 2010, an additional $20,000 on or before each of January 18, 2011 through January 18 2016 and an additional $25,000 on each subsequent January 18 thereafter during the term (all of which minimum royalties are recoverable from production royalties). An NSR production royalty of 3% is payable to the lessors. ITH may purchase all interest of the lessors in the leased property (including production royalty) for $1 million (less all minimum and production royalties paid to the date of purchase), of which $500,000 is payable in cash over 4 years following the closing of the purchase and the balance of the $500,000 is payable by way of the 3% NSR production royalty.

The Hanneman/Bergelin Family Trust ground is held via a binding letter of intent with an effective date of September 1, 2006. The lease of 169 Alaska State mining claims is for an initial term of ten years, commencing on September 11, 2006. The lease requires payments of $50,000 in each of years 2-5 and $100,000 in each of years 6-10 and work expenditures of $100,000 in year 1, $200,000 in each of years 2-5, and $300,000 in each of years 6-10. An NSR production royalty of 2% and 5% is payable to the lessors (depending upon the price of gold). ITH may buy all interest in the property subject to the lease (including the retained royalty) for $10 million.

On Alaska State lands, the state holds both the surface and the subsurface rights. State of Alaska 40-acre mining claims require an annual rental payment of $35/claim to be paid to the state (by November 20), for the first five years, $70 per year for the second five years, and $170 per year thereafter. As a consequence, all Alaska State Mining Claims have an expiry date of November 30 each year. In addition, there is a minimum annual work expenditure requirement of $100 per 40 acre claim (due on or before noon on September 1 in each year) or cash-in-lieu, and an affidavit evidencing that such work has been performed is required to be filed on or before November 30 in each year. Excess work can be carried forward for up to four years. If such requirements are met, the claims can be held indefinitely. The work completed by ITH during the 2008 field season was filed as assessment work, and the value of that work was sufficient to meet the assessment work requirements through September 1, 2012 on all unpatented Alaska State mining claims held under lease. Work completed in 2009 has been filed and the expenditure is sufficient to carry forward through 2013.

Holders of Alaska State mining locations are required to pay a production royalty on all revenue received from minerals produced on state land. The production royalty requirement applies to all revenues received from minerals produced from a state mining claim or mining lease during each calendar year. Payment of royalty is in exchange for and to preserve the right to extract and process the minerals produced. The current rate is three (3%) percent of net income, as determined under the *Mining License Tax Law* (Alaska).

All of the foregoing agreements and the claims under them are in good standing and are transferable. Except for the patented claims, none of the properties have been surveyed.

Holders of Federal and Alaska State unpatented mining claims have the right to use the land or water included within mining claims only when necessary for mineral prospecting, development, extraction, or basic processing, or for storage of mining equipment. However, the exercise of such rights is subject to the appropriate permits being obtained.

4.3 Environmental Requirements

Project activities are required to operate within all normal Federal, State, and local environmental rules and regulations. This includes proper and environmentally conscientious protection of operational areas against spills, capture and disposal of any hazardous materials including fuel, drill fluids, and other materials used by equipment that are part of the drilling and exploration process. Reclamation of disturbed ground and removal of all refuse is part of normal operations.

With over 90 years of placer mining activity and sporadic prospecting and exploration in the region, there is moderate to considerable historic disturbance. Some of the historic placer workings are now overgrown with willow and alder. The old mining town of Livengood is now abandoned except for more modern road maintenance buildings at the town site. ITH does not anticipate any obligations for recovery and reclamation of historic disturbance.

ITH commissioned Northern Land Use Research, Inc. (NLUR) to complete a cultural resource survey in 2008 (**Figure 4.3**). An initial report was submitted to ITH in January, 2009 (Northern Land Use Research, Inc., 2009). This Level 1 or Identification Phase survey was commissioned by ITH to locate and document historic sites, cultural features, or artifacts in the project area. Twelve previously undocumented historic sites or artifacts were identified in 2008. No prehistoric artifacts and no previously unknown prehistoric cultural resources were located in the 2008 exploration area.

A second cultural resource survey was conducted by NLUR during the summer of 2009 to cover a larger, expanded exploration area. The survey documented historic (i.e. archeological) mining equipment, buildings and linear ditch features, and relocated a previously known prehistoric site

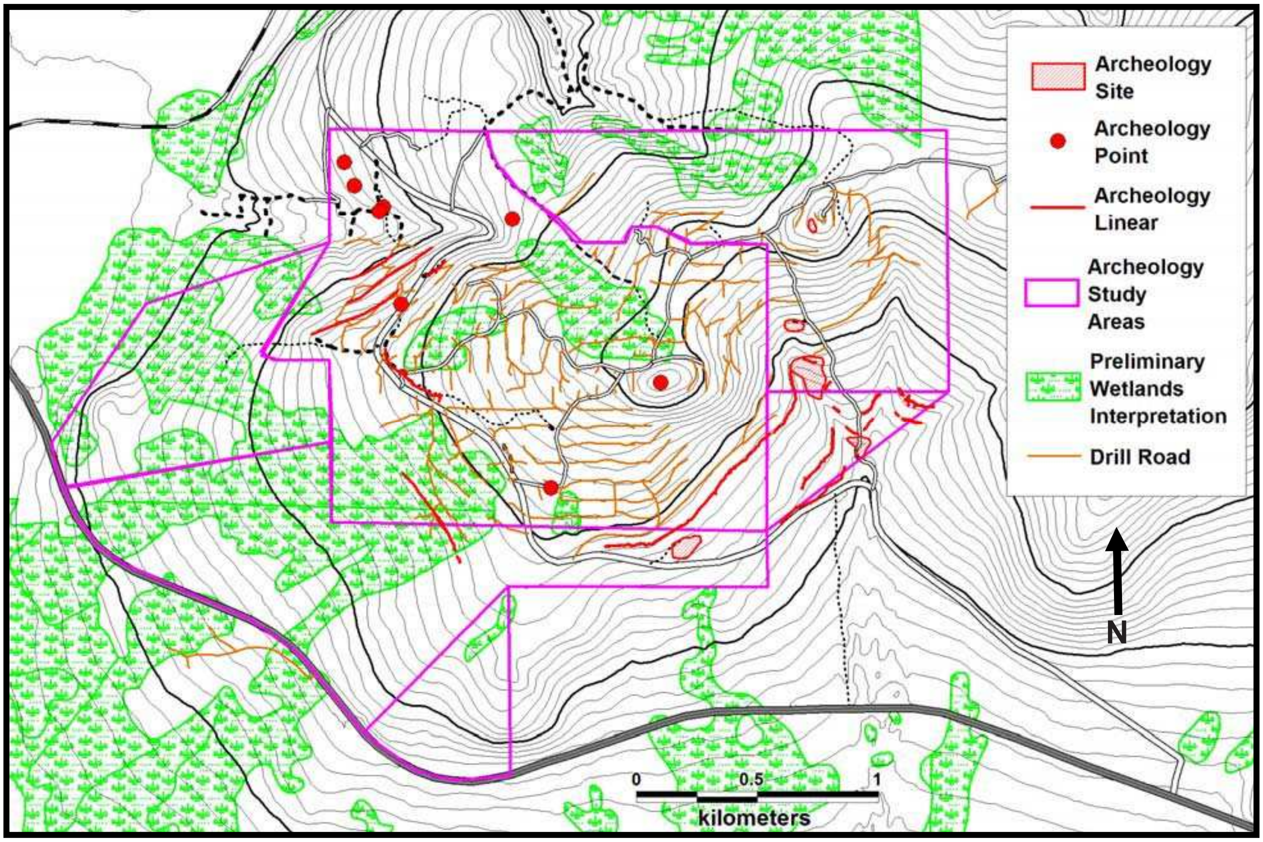

Figure 4.3 Map of the Money Knob area showing "wetlands" in green pattern, initial and expanded archaeological study area, and the location of cultural features identified in the survey. The Elliott Highway runs across the southern portion of the map area.

within the expanded coverage area (**Figure 4.3**). The assessment continues at an identification phase so that their historic significance can be evaluated at a later date. The ADNR will determine if any identified cultural resources require further action or isolation from disturbance. Historic features were flagged and noted to ITH staff for avoidance so they could continue exploration at Livengood.

Total disturbance associated with ITH's exploration consists of drill pad access roads and drill pads. However, as the number of drill holes increases, the local impact does as well. An ongoing program of reclamation of pads and roads reduces the impacted area to the minimum possible at any given time. For much of the exploration area, disturbance involves areas covered by secondary growth of alder, willow, and spruce and consequently, the impact is largely not visible from the Elliott Highway or the road into the Livengood town-site. Visual impact is minimal. The highest ground is naturally bare broken rock or sparsely covered in small shrubs and mosses.

The USACE permit requires that all wetland sites be drilled in the winter to minimize disturbance and it requires that all roads and pads in wetlands be fully reclaimed prior to April

15th of the year in which they are disturbed. The winter 2009 program operated according to these guidelines and is in compliance with the USACE permit.

Three Parameters Plus, Inc. of Fairbanks, AK, a natural resource consulting firm, has been retained by ITH to: 1) conduct an initial baseline surface water sampling program to evaluate metal and organic content of streams that drain the project area as well as regional streams up-gradient from the project area; and 2) complete a wetlands inventory extending beyond ITH's land position.

Water samples are being collected from 13 sites on a near monthly basis during the summer months. This survey is currently in progress and analyses are pending. One well has been established to monitor the static water table fluctuations on Money Knob and water table measurements are taken on each hole upon completion.

ABR Inc. of Anchorage, AK is conducting a survey to assess quality and biodiversity of fish, benthic invertebrate, and periphyton populations in the streams that drain and are adjacent to the project area. Surveys of this type are conducted at this early stage of a project to determine the current conditions against which environmental quality metrics can be established should a mine be constructed. Two separate attempts to identify fish populations that might be suitable for environmental monitoring, including both minnow traps and electrofishing, encountered only grayling, which are unsuitable for monitoring because of their migratory habits. No other species were identified.

Wildlife in the area consists of moose, bear, and various small mammals. None were observed in the course of the site visits although moose and bear have been seen in the vicinity. Hunters can be active in the region and local trap lines may be present. There are no known wildlife issues.

There are no known existing environmental liabilities.

4.4 Permits

Operations which cause surface disturbance such as drilling are subject to approval and receipt of a permit from the ADNR and the BLM. The ADNR permit for ground controlled by the State of Alaska was issued on January 26, 2009 and covers calendar years 2009 and 2010. Exploration on Federal ground is permitted by the BLM under a Plan of Operations covered by EA-AK-024-08-010 (File FF095365) and is effective, without time limit, up until commencement of mining.

A permit is required from the USACE for exploration activities that may affect wetlands areas. This permit was granted on November 13, 2008 and enables ITH to drill in areas of shrub and tundra on and around Money Knob according to a USACE Preliminary Jurisdictional Determination. In support of this permit, the Alaska State Department of Environmental conservation has issued, on November 4, 2008, their Certificate of Reasonable Assurance for mineral exploration by ITH near Livengood. These permits require ITH to comply with all Federal and State regulations that apply to these areas.

There are no known issues at this time that would hinder ongoing renewal of any permits.

There are no known issues concerning water beyond normal operational obligations. These fall under operating permits issued by the state as outlined above.

There are no known native rights issues concerning the project area.

5.0 Access, Climate, Local Resources, Infrastructure and Physiography

5.1 Access

The Livengood Project area is located approximately 115 km northwest of Fairbanks on the Elliott Highway, which provides paved year-round access to the area. At present there are no full time residents in the former mining town of Livengood. A number of unpaved roads have been developed in the area providing excellent access.

A 1400 foot runway is located 6 km to the southwest near the former Alyeska Pipeline Company Livengood Camp. Also, a small airstrip (currently out of service) is in Livengood Creek north of the project area.

5.2 Climate

The climate in this part of Alaska is continental with temperate and mild conditions in summer with average lows and highs in the range of 7 to 22°C. Winter is cold with average lows and highs for December through March in the range of -27 to -5°C. Annual precipitation is on the order of 23 cm which arrives mostly in the summer. Winter snow accumulation ranges up to 66 cm (http://www.wrcc.dri.edu/cgi-bin/cliMAIN.pl?ak5534).

5.3 Local Resources

The project is serviced from Fairbanks, population 87,000. As central Alaska's principal center of commerce it is home to many government offices including the Alaska Division of Geological and Geophysical Surveys and the U.S. Geological Survey, as well as the University of Alaska Fairbanks. The town is serviced by major airlines with numerous daily flights to and from Anchorage and other locations. Helicopters and fixed wing aircraft are readily available. Virtually all supplies necessary for the project can be obtained in Fairbanks.

On-site operations are conducted from a refurbished portion of the former Livengood Camp which was installed for the Alaska Pipeline construction. Current camp facilities can accommodate up to 100 people, sufficient to meet the needs of the on-going exploration program.

5.4 Infrastructure and Physiography

The project is situated in forested hilly countryside with mature, subdued topography partly owing to widespread deposition of Pleistocene loess and gravel in valleys (**Figure 5.1**). Elevation ranges from about 150m (~500') in valley bottoms to 700m (2317') at Amy Dome along the east side of the property. Streams meander through wide, flat-bottomed, alluvial-filled valleys. Ridge lines are generally barren with sparse vegetation. Hillsides host mixed spruce-birch forest with abundant alder.

The area is drained by Livengood Creek which flows to the southwest into the Tolovana River which then joins the Tanana River and ultimately the Yukon River approximately 190 km to the west.

Existing infrastructure includes a paved highway (the Elliott Highway) which passes through the property and within ~ 1.6 km of Money Knob. Lesser unpaved roads are developed throughout the property. A repeater tower has been built on Radio Knob approximately 1.6 km east of Money Knob.

Self generated power currently exists at the Livengood camp. The nearest Alaskan grid power is approximately 67 km (40 miles) away at its closet point to the Livengood property. A power line will need to be constructed for power supply to the proposed Livengood facility for operational demands.

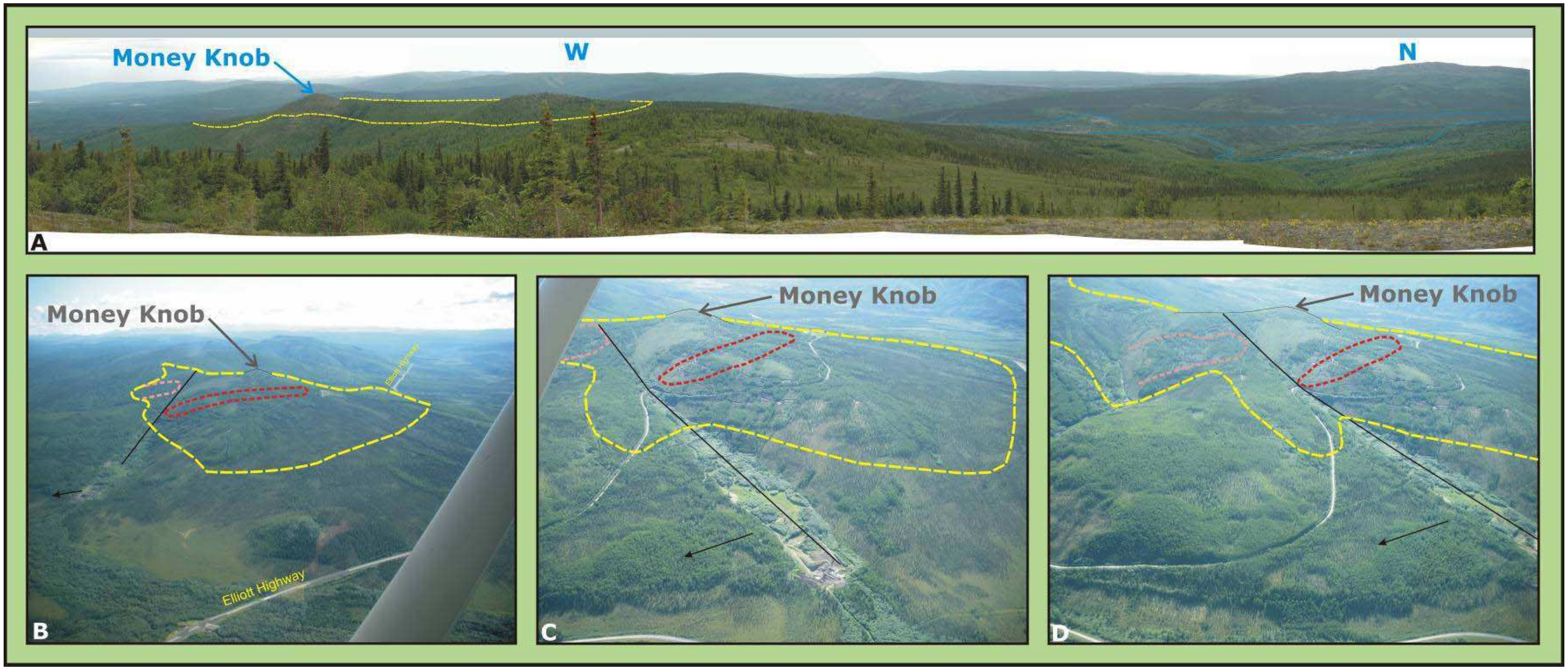

Figure 5.1. Photos of Money Knob and the project area. **A)** Panoramic view looking west and north toward Money Knob. Dashed yellow line outlines the perimeter of the area under investigation by drilling. Blue lines to the right outline placer workings to the north in Livengood Creek. **B - D)** Aerial view of Money Knob from the west and northwest showing the Lillian Fault (black line), and area under investigation by drilling (yellow dashed line). The "Core Zone" is outlined with a dotted red line. The Sundshine Zone is outlined with a pink dotted line. Arrow indicates north.

6.0 History

Gold was first discovered in the gravels of Livengood Creek in 1914 (Brooks, 1916). Subsequently, over 500,000 ounces of placer gold were produced and the small town of Livengood was established. Since then, the primary focus of prospecting activity has been with the placer deposits. Historically, prospectors have considered Money Knob and the associated ridgeline to be the source of the placer gold. Prospecting in the form of dozer trenches was carried out for lode type mineralization in the vicinity of Money Knob primarily in the 1950's. However, to date no significant production has been derived from lode gold sources.

The geology and mineral potential of the Livengood District has been investigated by state and federal agencies as well as explored by several companies over the past 40+ years. Modern mapping and sampling investigations were initially carried out by the U.S. Geological Survey in 1967 as part of a heavy metal assessment program (Foster, 1968). Mapping completed in the course of this program recognized the essential rock relations, thrust faulting, and mineralization associated with Devonian clastic rocks, the thrust system and intrusive rocks. These relations are summarized in the following insightful comment from the report summary.

> *"The small lode deposits in the upper plate rocks may represent leakage anomalies above economically significant metal deposits in rocks in or below the thrust fault zones."*

Since then, the Livengood placer deposits and the surrounding geology have featured in numerous investigations and mapping programs at various scales by the U.S. Geological Survey and the Alaska State Division of Geological and Geophysical Surveys. Principal among these are: Chapman, Weber, and Taylor, 1971; Chapman and Weber, 1972; Cobb, 1972; Albanese, 1983; Robinson, 1983; Smith, 1983; Waythomas, and others, 1984; Arbogast, 1991; Athey and Craw, 2004; and Athey and others, 2004.

In 2003, as part of a larger state-wide program, the Alaska Division of Geological and Geophysical Surveys undertook a district-scale program of mapping and whole rock geochemical sampling in support of the mapping. They report "one highly anomalous sample that yielded slightly over one ounce per ton gold" (Athey and Craw, 2004).

In addition to individuals prospecting the area, corporate explorers have investigated the potential for lode gold mineralization beneath the Livengood placers and on the adjacent hillsides including at Money Knob. A summary of these programs is shown in **Table 6.1**. Placer Dome's work appears to have been the most extensive, but it was focused largely on the northern flank of Money Knob and the valley of Livengood Creek.

The most recent exploration history of Money Knob began when AGA acquired the property in 2003 and undertook an 8-hole RC program on the Hudson-Geraghty lease. The results from this program were encouraging and were followed up with an expanded soil geochemical survey which identified anomalous zones over Money Knob and to the east. Based on the results of this and prior (Cambior) soil surveys, 4 diamond core holes were drilled in late 2004. Results from these two AGA drill

TABLE 6.1
EXPLORATION HISTORY

Company / Year	Major Activity	Results	Comment
Homestake / 1976	Geochemistry & 6 boreholes	Significant soil anomaly, low grade gold in drill holes and auger samples	Management decided on other priorities.
Occidental Petroleum / 1981	6 boreholes	Low-grade gold encountered in several holes	Other priorities.
Alaska Placer Development 1981 - 1984	Extensive soil and rock sampling together with mapping, mag, EM, trenching and auger drilling.	Defined soil and rock anomalies; other data not available.	Mostly on flanks of Money Knob. Changed focus to placer deposits.
Amax / 1991	3 RC holes; surface geochemistry and auger testing	Good geological mapping, lots of rock sampling, low grade gold in drill holes.	Other priorities.
Placer Dome / 1995 - 97	Surface exploration; / geophysics & 9 diamond core holes	Intersected some moderate grade mineralization.	Work focused to north of Money Knob. Limited land position.
Cambior 1999	Geochemistry	First to identify the extent of gold on Money Knob.	Corporate restructuring – no follow-up.
AGA / 2003-2005	Geochemistry, trenching, geophysics, drill testing;	Geochemical anomaly, numerous drill intersections	Intersected gold-bearing intervals.
ITH 2006-2007	Surface geochemical sampling; drilling 23 holes	First intersection of extensive zones of > 1g/t Au.	Intersected more gold-bearing intervals; initial resource estimates
ITH 2008	108 reverse circulation, 7 diamond core holes, and 4 trenches through September 27.	Infill and step-out grid drilling of mineralization	Expanded resource estimates.
ITH 2009	133 reverse circulation holes; 6 diamond core holes	Infill drilling in wetland areas; discovery of Sunshine Zone and other areas of mineralization; expanded resource estimate	Results discussed in this report

programs were deemed favourable but no further work was executed due to financial constraints and a shift in corporate strategy.

In 2006, Livengood and other properties now part of the ITH portfolio were sold to ITH by AGA. In the same year, ITH drilled a 1227 m, 7-hole program. The success of this program led to the drilling of an additional 4400 m in 15 diamond core holes in 2007 to test surface anomalies, expand the area of previously intersected mineralization, and advance geologic and structural understanding of subsurface architecture.

Geophysical work in the vicinity includes an airborne magnetic survey by Placer Dome in 1995. This data has not been recovered. They also conducted VLF surveys in the northern part of the district in 1996 with only limited success because of the mixed frozen and thawed ground. This data is only partially preserved. The state of Alaska flew a 400 meter line spaced DIGHEM survey (an aerial, multi-channel electromagnetic technique) over the Livengood District in 1998 (Burns and Liss, 1999; Rudd, 1999). AGA ran a series of CSAMT (Controlled-Source Audio-frequency Magneto-Telluric) lines across Money Knob in 2004. This survey was designed to look for resistive intrusive bodies in the subsurface. The survey appeared to map the main thrust zone but did not appear to delineate hidden intrusive bodies.

7.0 Geological Setting

7.1 Regional Geology

The Livengood 'district' is a portion of the broader Tolovana Mining District. It is situated in a complex assemblage of rocks known as the Livengood Terrane (**Figure 7.1**). This Terrane is an east–west-trending belt, approximately 240 kilometres long, bounded on the north by splays of the dextral Tintina-Kaltag strike-slip fault system and other terranes to the south (Silberling and others, 1994; Goldfarb, 1997). It is composed of a complex sequence of rocks which do not match assemblages of the adjacent Yukon – Tanana Terrane. Throughout the Livengood Terrane, individual assemblages of various ages are tectonically interleaved. Each assemblage, and perhaps the stratigraphy within each assemblage, is bounded by both low to moderate (?) angle thrust faults and steep faults, of which at least some of the latter type are interpreted to be splays of the Tintina Fault system. Rocks of the Livengood Terrane are generally highly deformed, but weakly metamorphosed Neoproterozoic to Paleozoic marine sedimentary rocks along with Cambrian ophiolitic sequences, Ordovician Livengood Dome chert, overlying dolomite, volcanic rocks, terrigenous clastic rocks, and minor Devonian limestone (Silberling, et al., 1994; Athey et al., 2004).

The Livengood Terrane is overprinted by later Mesozoic intrusions believed to have originated in the back-arc position above subducting oceanic crust. These intrusions are quartz monzonite to diorite to syenite in composition and some of them are believed to be responsible for gold mineralization of the Tintina Gold Belt (McCoy, et al., 1997; Goldfarb, et al., 2000). The Livengood district occurs within the Tintina Gold Belt, an arcuate belt of gold mineralization that extends from the Yukon to south-western Alaska and hosts numerous gold deposits, including Fort Knox and other deposits of the Fairbanks District and the Donlin Creek deposit in the Kuskokwim region (Smith, 2000).

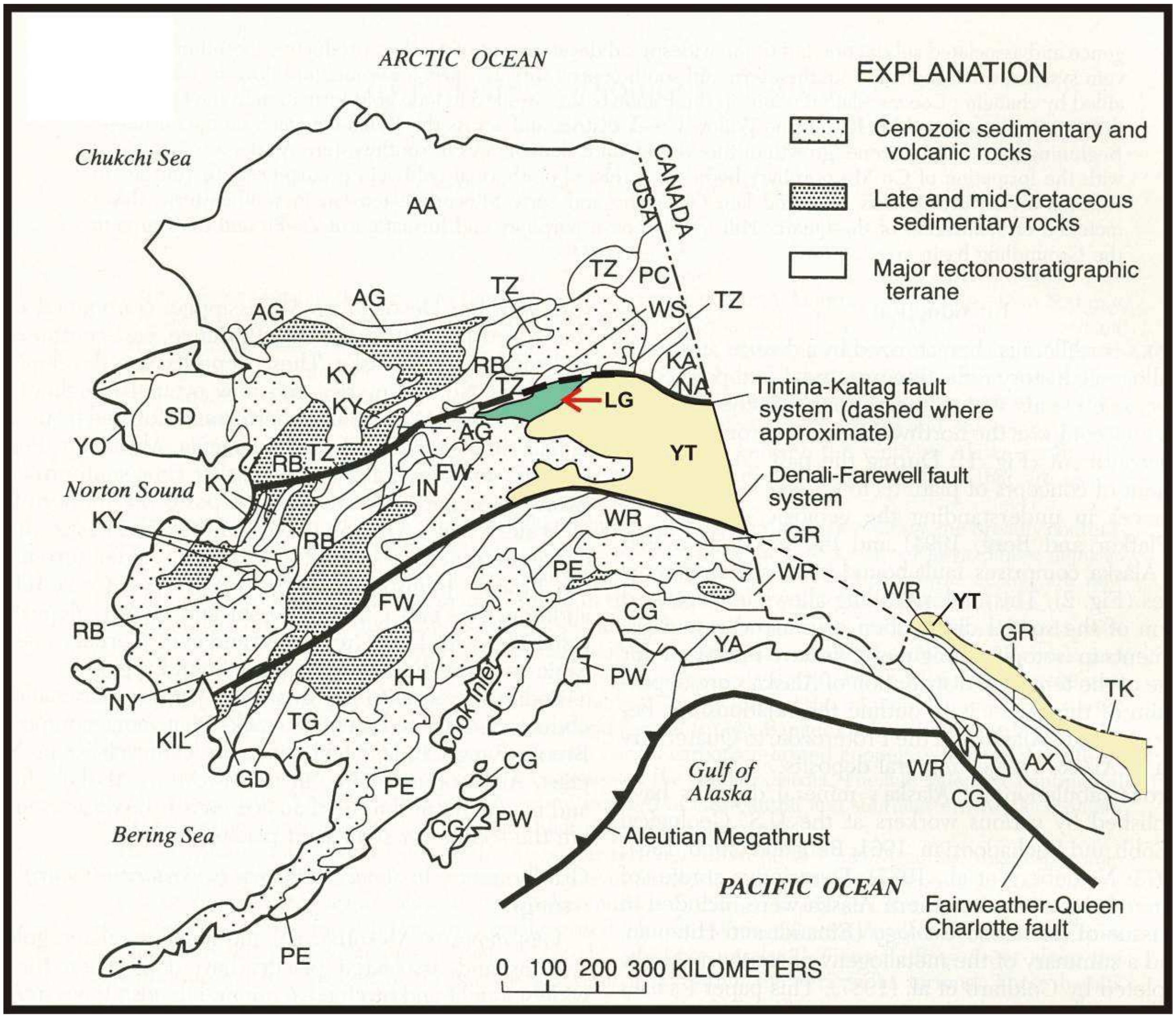

Figure 7.1. Terrane map of Alaska showing the location of the Livengood Terrane (LG; red arrow). The heavy black line north of the Livengood Terrane is the Tintina Fault. The heavy black line to the south of the Livengood and Yukon – Tanana Terrane (YT) is the Denali Fault. The Tintina Gold Belt lies between these two faults. After Goldfarb, 1997.

7.2 Local Geology

In the vicinity of the Livengood project, the oldest rocks are Neoproterozoic to early Paleozoic basalt, mudstone, chert, dolomite, and limestone of the Amy Creek Assemblage (lPzZ units on Livengood geology map; Athey et al., 2004) (**Figures 7.2 and 7.3**). These units are believed to represent incipient ocean floor basalt in a continental rift system and overlying sediments. The origin and age are poorly constrained but fossil evidence suggests a depositional age between Neoproterozoic and Silurian time.

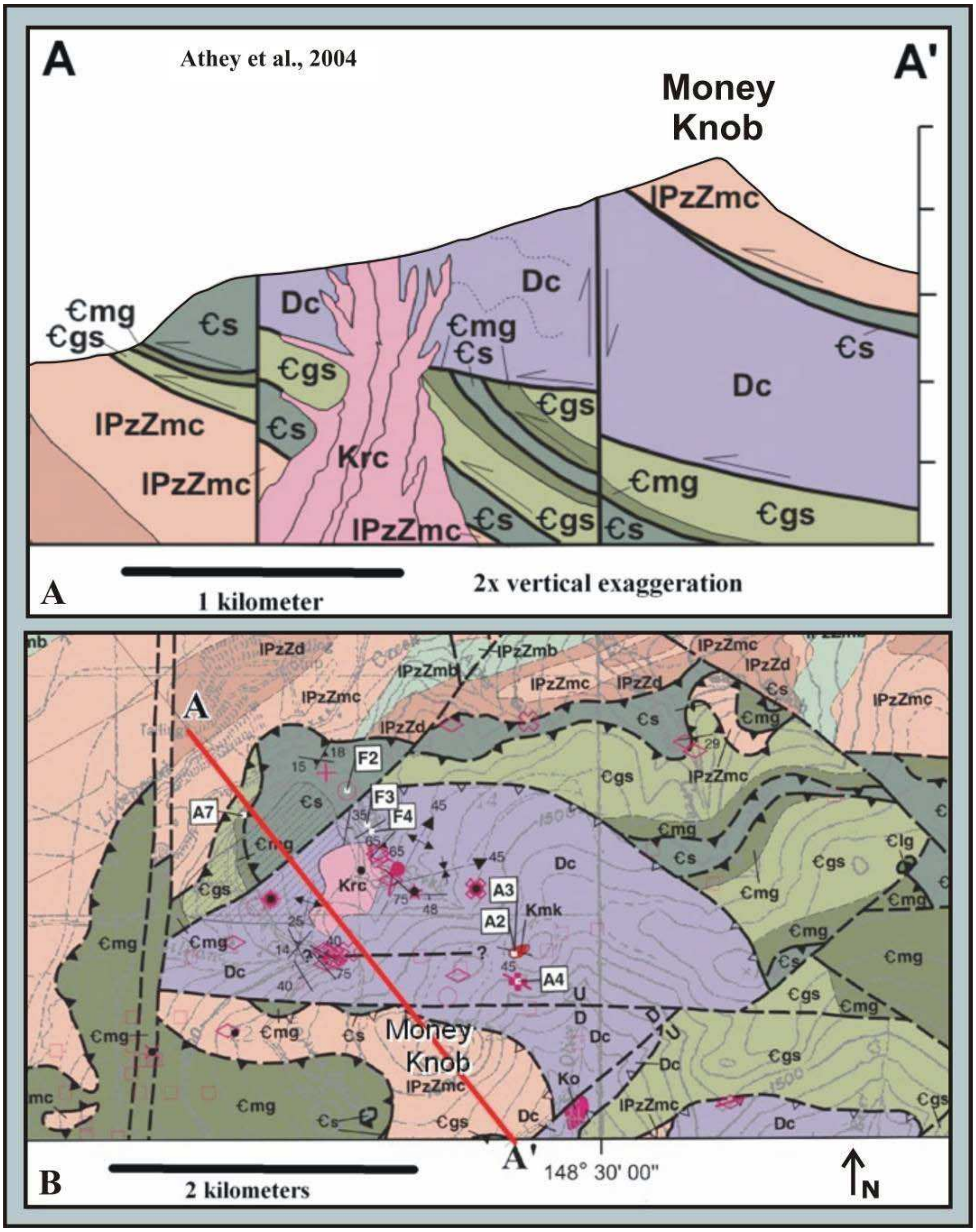

Figure 7.2. Geologic cross section and map of the Livengood project area (Athey, et al., 2004). **A)** Cross section through Money Knob illustrating the geological components of the Livengood District. lPZZmc are older siliceous shelf metasediments. Cs, Cgs and Cmg are Cambrian mafic and ultramafic volcanics and intrusive rocks of oceanic ophiolitic affinity. Dc represents Devonian siliciclastic sediments. The thrust imbrication may reflect two deformation events, one in the Permian and one in the Middle Cretaceous. The thrust package has been intruded by a number of Cretaceous felsic intrusions. **B)** Geologic map showing the location of the cross section 'A-A'. Pink symbols identify rocks mapped as intrusive and mostly known now to be Devonian volcanics.

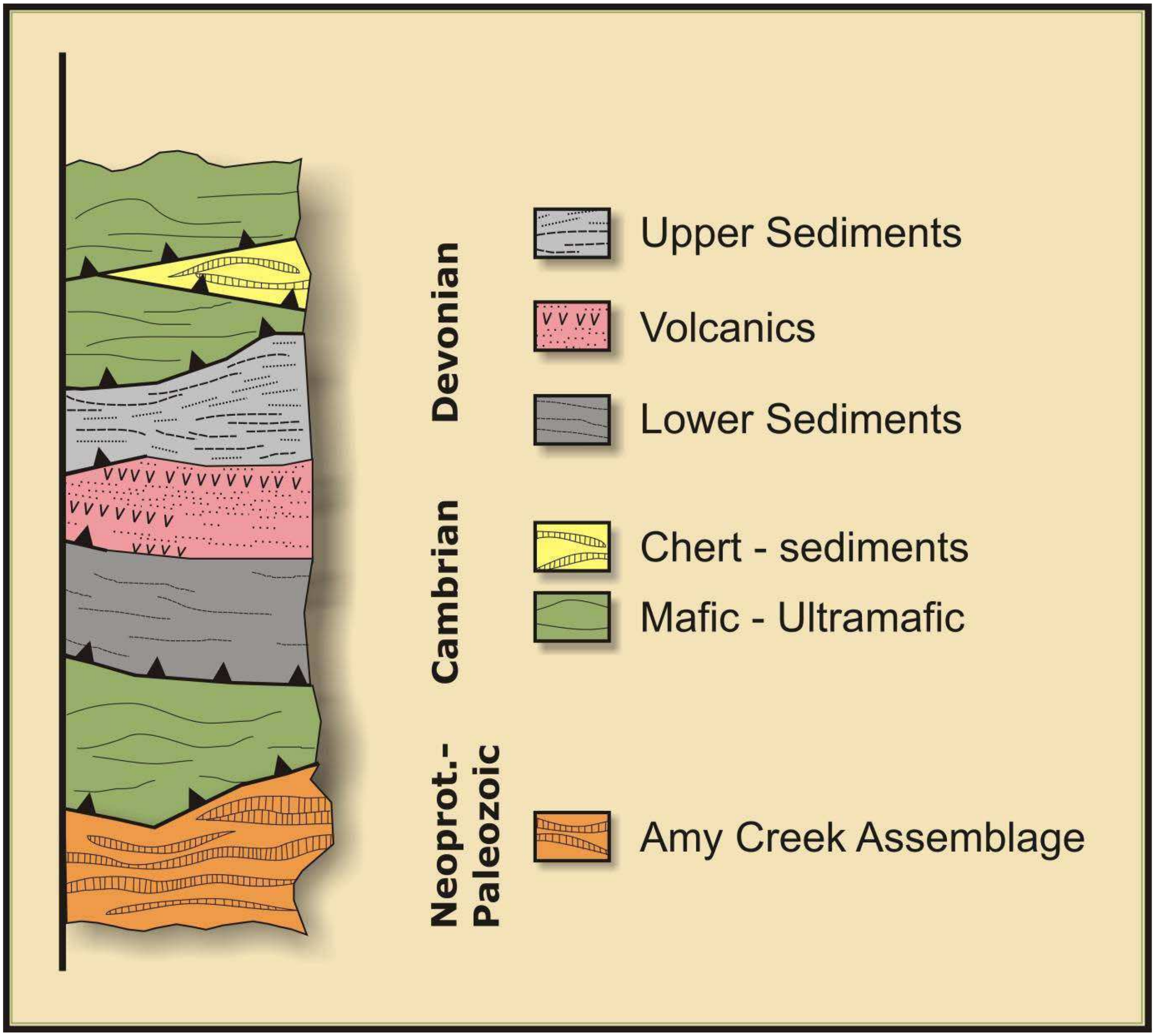

Figure 7.3. Diagrammatic lithologic column shows the tectonic stacking of rock groups in the Livengood area.

Above the Amy Creek Assemblage lies an early Cambrian ophiolite sequence (Plafker and Berg, 1994). This assemblage consists of structurally interleaved greenstone, pyroxenite, metagabbro, layered metagabbro, ultramafic rocks and serpentinite derived from them (**Figures 7.2** and **7.3**). Metamorphic ages suggest that this assemblage was tectonically emplaced over the Amy Creek Assemblage by north-directed thrusting during Permian time.

The Cambrian ophiolite sequence is, in turn, overlain by Devonian rocks which include shale, siltstone, conglomerate, volcanic, and volcaniclastic rocks (**Figures 7.3 - 7.6**). This assemblage is the principal host for gold mineralization. These rocks have been subdivided into “Upper” and “Lower” sedimentary units with volcanic rocks separating them (**Figure 7.3**). The Upper Sediments consist of siltstone, sandstone, conglomerate, shale, and minor limestone and dolomite. The Lower Sediments unit is dominantly shale in the northern portion of the property but includes sandy siltstones and fine sandstones to the south. Use of trace element ratios has helped discriminate these units from one another. The volcanics consist of flows and pyroclastic rocks. Some of these volcanic rocks were previously mapped as Cretaceous intrusive rocks (Athey et al., 2004). However, geologic observations in drill core and the use of trace element ratios indicate that most of the rocks mapped as the “Ruth Creek” and “Olive Creek” plutons are volcanics and part of the Devonian stratigraphy.

Structurally above the Devonian assemblage is a klippe of the Cambrian ophiolitic mafic and ultramafic rocks with tectonically interleaved wedges of cherty sedimentary rock (**Figures 7.3 and 7.4**). The emplacement of this klippe may have taken place in Cretaceous time during closure of the Manley Basin south of the project area. The thrust contacts between the various rock units indicates that there has been extensive thrust stacking and interleaving of the different assemblages as well as possible local interleaving of some units within the assemblages.

Rocks in each of these assemblages have been folded, but overall, they strike east-west to northwest-southeast and dip shallowly to moderately south, consistent with postulated northward directed thrust transport.

Drill intercept patterns and foliation-bedding relations observed in core (**Figures 7.6 d and e**) indicate that these rocks define a principal recumbent fold and possible parasitic folds segmented by south-dipping thrust and normal faults.. Later Cretaceous dikes and sills intrude the sequence, some of which are believed to intrude along these faults.

The thrust-stacked Paleozoic sequence described above is intruded by back-arc Cretaceous (91.7 – 93.2 m.y.; Athey and Craw, 2004) multiphase monzonite, diorite, and syenite stocks, dikes, and sills with equigranular to porphyritic textures. Athey et al. (2004) concluded that the intrusive rocks were the primary host to the gold mineralization. However, exploration work since then has shown that these rocks are, in part, Devonian volcanics which have undergone extensive alteration along with introduction of mineralization in or associated with quartz and quartz-carbonate veins. Narrow Cretaceous stocks (?) and large dikes are biotite monzonite. Narrower, late (?) stage dikes are composed of feldspar porphyry, and aplitic felsic intrusives without biotite (**Figure 7.6**). Mineralization is, at least partially, associated spatially and probably genetically with these dikes.

The structural architecture of the project area is characterized by fold-thrust patterning, apparently overprinted by local, minor normal offset along primary normal faults or reactivated thrust faults (**Figure 7.7**) and a possible second fold event. Apparent upright open folds have axes that strike NW and plunge gently in that direction. Later faults include the Lillian and the Myrtle Creek.

Thrust faults appear to lie in two principle dip orientations; subhorizontal and low to moderately south-dipping. Undulatory subhorizontal thrust faults appear to define the primary thrust surface separating

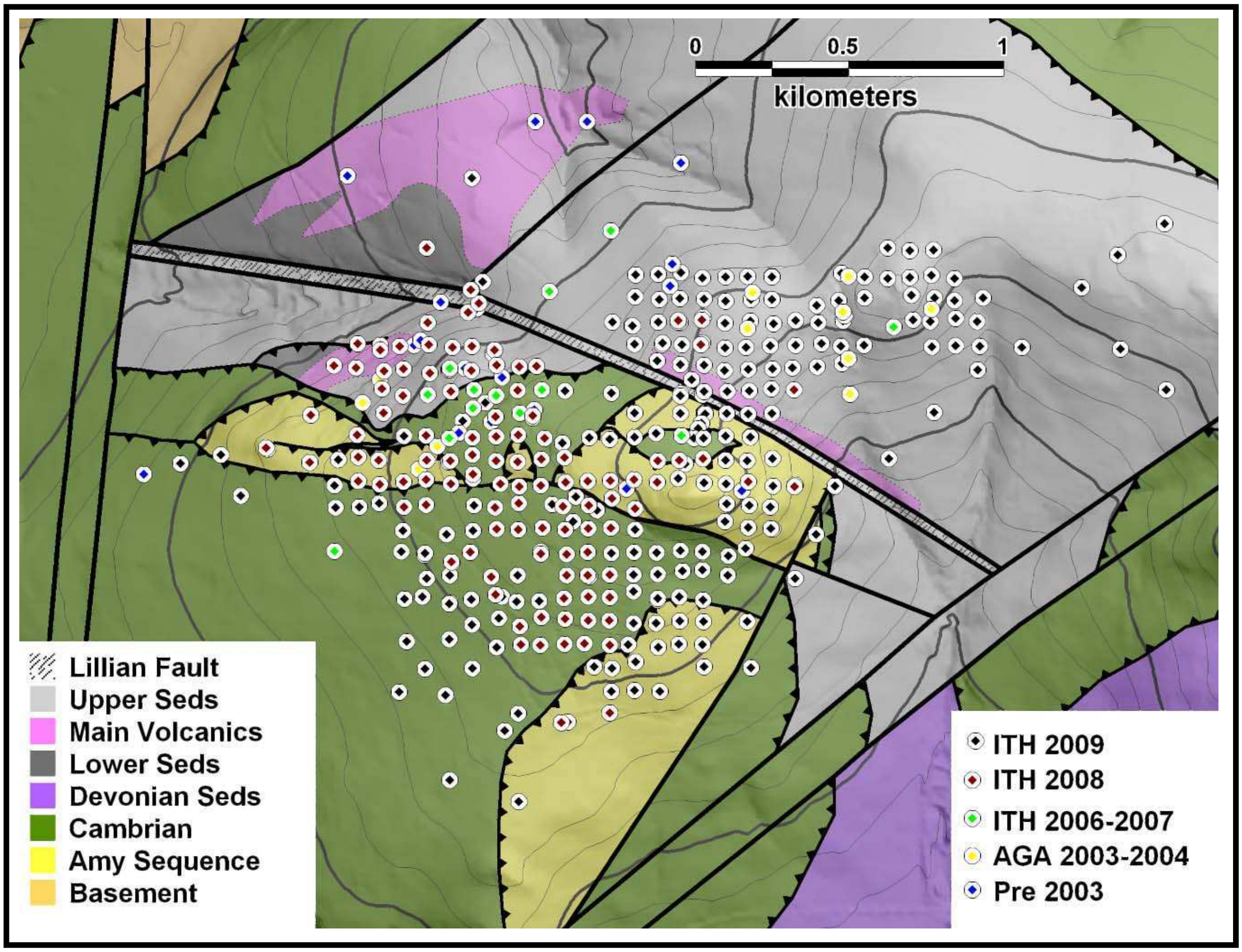

Figure 7.4. Generalized geologic map of the Money Knob area based on geologic work by ITH. Drill hole collars are shown by year drilled.

the Cambrian ophiolite sequence from underlying Devonian sedimentary and volcanic sequence. These rocks and their low angle thrust contact appear to be segmented and offset by low to moderately south-dipping thrust faults. In some instances, these south-dipping structures display apparent normal offset. Details of this patterning are currently being evaluated but possible interpretations include: 1) post-thrusting tectonic relaxation resulting in minor normal offset on reactivated thrust surfaces; 2) the existence of a late-stage extensional tectonic event; or 3) some, as yet, poorly understood complex relation between faults. Correlation of particular faults from one drill hole to another is subject to different possible interpretations. Key points that need to be resolved, if possible, relate to distinguishing low angle and south-dipping structures and the relative timing of these features.

The Lillian Fault is a northwest trending steep, possibly south-dipping fault that is characterized by a wide zone of sheared sedimentary and dike rocks that separates the property into two domains. To the south, the structural and stratigraphic sequence is well-defined consisting of gently south-dipping

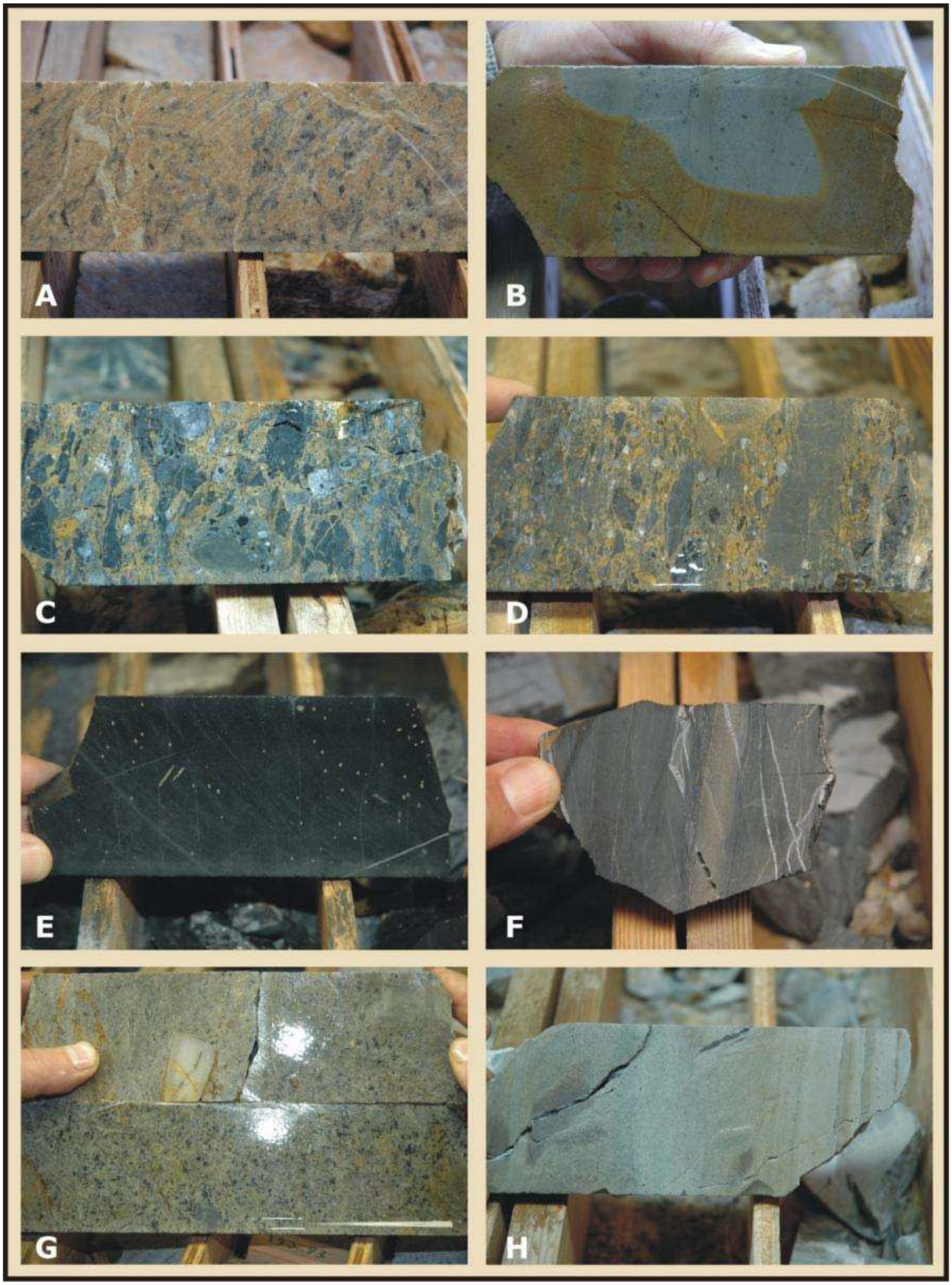

Figure 7.5. Photographs of key rock types at Livengood. **A)** ultramafic rock with carbonate alteration (yellow-brown); MK7-20, 13.5m; **B)** siltstone with carbonate and pyrite knots. Brown color is oxidation front. MK 07-18, 8.5m **C)** sedimentary conglomerate; at least some clasts appear to be rip-

up clasts of similar sedimentary rocks; brown color is after introduced carbonate; MK07-18, 41.2m; **D)** sedimentary conglomerate; at least some clasts appear to be rip-up clasts of similar sedimentary rocks; brown color is after introduced carbonate; MK07-18, 57.7m; **E)** argillite with pyrite; MK07-20, 222m; **F)** argillite with siltstone band; MK07-18, 280 ; **G)** tuff showing lithic fragments; this unit contains MK07-18, 190m 0.23 – 0.75 g/t Au; **H)** fine-grained tuffaceous sediment; MK07-20, 151.5m.

sedimentary and volcanic stratigraphy and thrust faults. These rocks host the Core Zone and surrounding mineralization.

To the north of the Lillian Fault, the upper Cambrian ophiolite sheet is not preserved and the upper sedimentary sequence is much thicker than the sequence preserved south of the Lillian Fault. Immediately to the north of the Lillian fault the stratigraphy dips very steeply to the north and strikes parallel to the Lillian Fault suggesting that movement on the fault was reverse at some time. Steeply dipping rocks north of the Lillian Fault host mineralization of the Sunshine Zone.

Immediately south of the fault, the axis of a north-vergent, major recumbent fold is subparallel to the strike of the Lillian Fault. This implies that, during the early history of the fault, there may have been steep reverse movement followed by later collapse and normal offset with down drop to the south. At present, subhorizontal lineations are common on faults in and around the Lillian Fault suggesting a history with possible late strike-slip movement. Regional Mesozoic to Cenozoic dextral slip on the Tintina-Kaltag Fault system to the north of Livengood may support an interpretation of late dextral motion on the Lillian Fault.

To the west of the deposit, the approximately north-south Myrtle Creek Fault (**Figure 7.2**) is mapped as having strike-slip offset by early workers and west-side-down, normal offset by Athey, et al. (2004). It is believed that offset along this fault influenced the paleo-drainage system of the area. Based on a number of lines of evidence, it is proposed that Livengood Creek used to flow to the northeast. Capture of the stream by the Tolovana River, and reversal of flow could have been related, in part, to movement along the Myrtle Creek Fault (Karl, et al., 1987; Athey and Craw, 2004). The origin and relationship of this fault to other structural elements in the area is not understood. It lies in an anomalous direction, but also extends for several 10s of kilometres to the south and a lesser distance to the north. This fault is not known to affect mineralization and is peripheral to the area of interest at Money Knob.

Immediately to the south of Livengood, the early to middle Cretaceous Manley Basin is preserved as a fold thrust sequence. Asymmetric overturned folds indicate a northern vergence direction to this deformation event. The precise age of the deformation is not well constrained but the youngest fossils in the basin are Aptian (125 – 112 m.y.) and the sequence was folded and thrusted prior to the emplacement of the 90Ma monzonitic intrusions in the thrusted sediments (Reifenstuhl et al., 1997). Because rocks of the Livengood Terrane at Livengood lack structural markers, it is not possible to determine if the fold-thrust deformation and closure of the Manley Basin impacted the older Livengood sequence. However, given the close spatial proximity of the two sequences and the fact that they are in thrust contact elsewhere, it seems likely that the Cretaceous deformation event affected the Livengood area. The extent to which thrust deformation at Livengood is Cretaceous or earlier

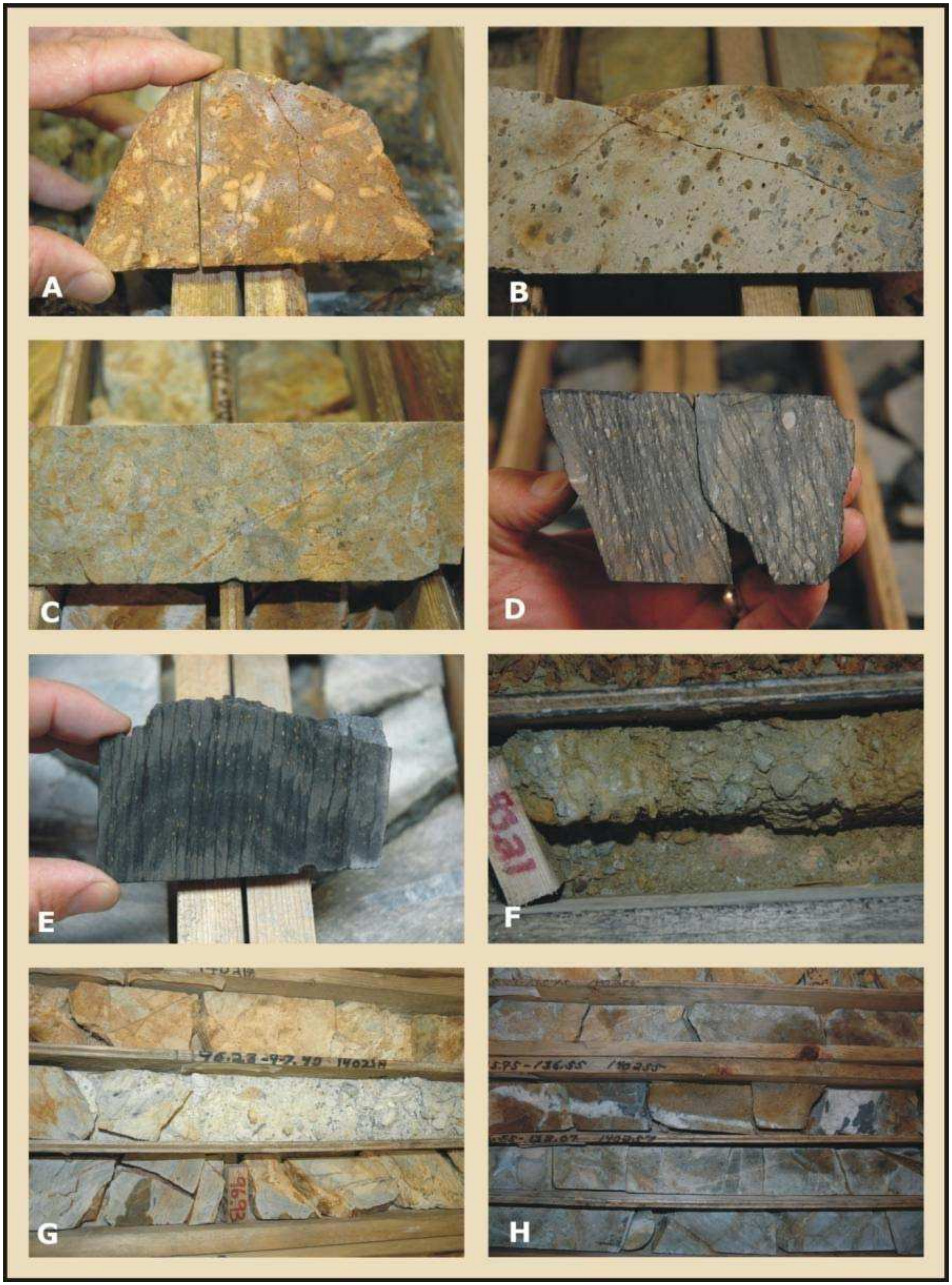

Figure 7.6. Photographs of key rock types and mineralization features. **A)** porphyry dike; MK07-18, 41.2 m; 1.01 g/t Au **B)** amygdaloidal volcanic, presumably a flow, with possible Na alteration; MK07-18, 152-189 **C)** silicified volcanic breccia; MK07-18 **D)** argillite with more silty band and coral hash;

note the shearing which is approximately 30^o to bedding; MK07-18, 288.4m **E)** axial planar cleavage on fold nose in interlayered argillite – silty argillite; MK07-18, 296.11m. This type of feature supports the fold-thrust interpretations of the cross section shown in Figure 10. **F)** fault; broken siltstone fragments in clay gouge/shear zone; this is part of an ~8m interval which contains 2 – 22.4 g/t Au; MK07-18, 77.9 – 86.08m; **G)** broken rock in shear zone within mineralized interval. The material in the photo includes portions of sample intervals that contain 15-16.2 g/t Au; MK 07-18, 96.93m **H)** narrow mineralized quartz vein in silicified volcanic contains 13 g/t Au and 35,900ppm As from arsenopyrite; MK07-18, 136.5m.

(Permian), and which rocks were affected at which time is currently being evaluated by ITH geologic staff. In addition, there is the possibility that multiple thrust events are overprinted by one or more (?) extensional events. As the Livengood project advances, structural interpretations will continue to mature and some structural interpretations may change as more information becomes available.

Key to understanding the structural architecture is collection of oriented structural data from drill core, which ITH does. In addition, understanding lithologic relations, and thereby the structural relations, is also key. This is done visually by drill core and chip loggers, but also through use of a portable XRF device (Thermo Fisher Scientific Niton XLT3) for initial assessment of the RC chips. Multielement ICP analyses provide additional data for geochemical evaluation of the rocks by principle component analysis. This technique utilizes the relative abundance and ratios of various immobile elements and has enabled discrimination of Devonian volcanics from Cretaceous intrusive and dike rocks as well as the upper and lower sedimentary assemblages. Procedures used by ITH for rock type discrimination rely on consistency between visual and chemical assessment of rock type. These procedures are described more fully in section 13.2.

7.3 Geological Interpretation

At the district scale, thrust stacking of rock assemblages (Amy Creek, Cambrian ophiolite, Devonian sedimentary and volcanic rocks) is reasonably well understood. Drilling reveals that there are numerous local fold and thrust complications which are only partially understood at this stage. It is likely that faults and fractures produced during fold-thrust deformation, along with possible overprinting extensional deformation, produced architecture that enabled localization of dikes and auriferous hydrothermal fluid. Gold mineralization largely appears to be controlled by and is spatially related to the fault architecture. The gold mineralization envelope encloses and lies parallel to axial planes of thrust-related recumbent folds. It appears as if mineralization occupies a broad ‘damage zone’ related to the fold-thrust architecture. Patterning in the resource block model is consistent with this interpretation.

The location and density of veins and diffuse mineralization appears to be controlled by lithology and alteration. Mineralization spatially associated with dikes appears to occur within ‘damage zones’ related to the south-dipping faults. However, the exact relationship and relative orientations of

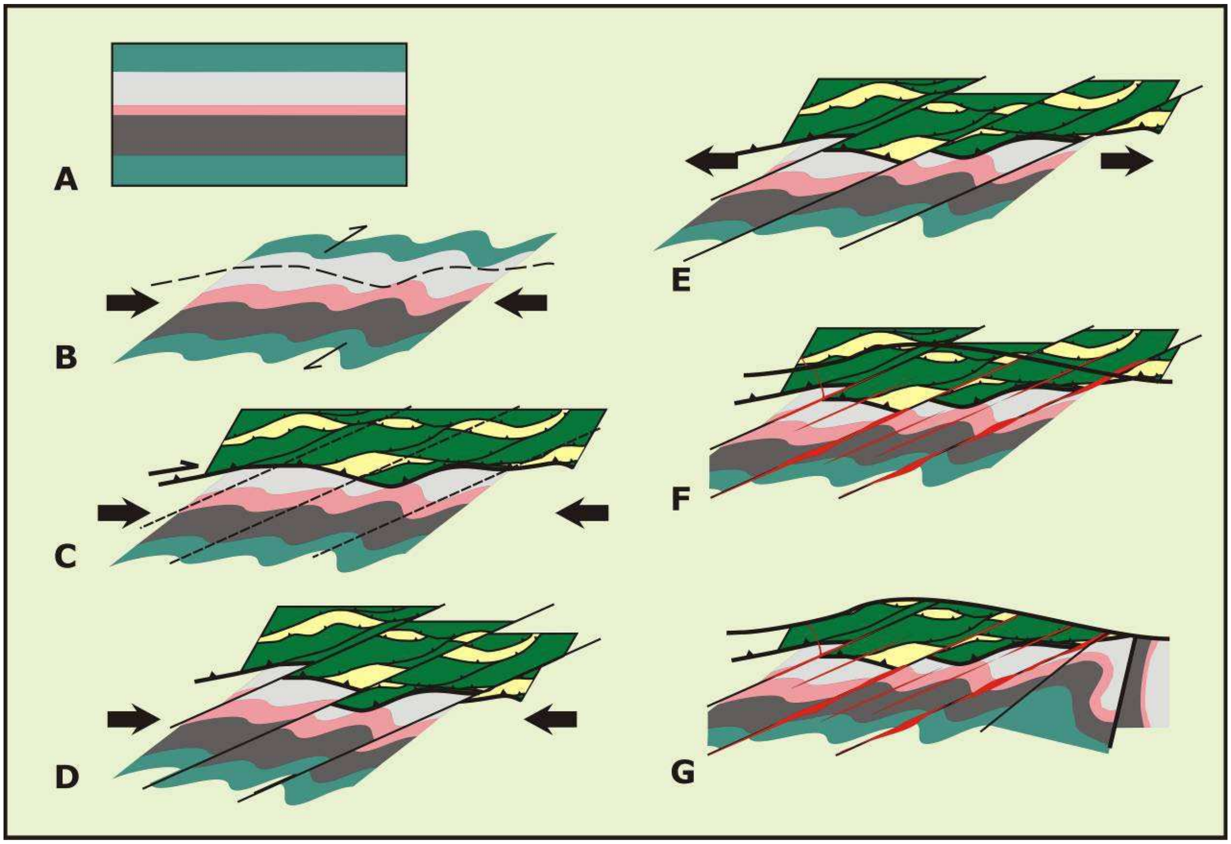

Figure 7.7. This cartoon shows an interpretive sequence of north-south sections and events to explain the structural relations observed at the surface and in drill core. The details and sequence of the events shown here are interpretations of Dr. Klipfel. ITH staff geologists are currently developing new hypotheses concerning the relative sequence and suggest that normal faulting has played a role in development of the structural architecture. One possibility is that the Cambrian ophiolite sequence was thrust in the Cretaceous, possibly contemporaneous with the closure of the Manley Basin.

A) Devonian volcano-sedimentary sequence is deposited. Pink – volcanics; light gray – upper sediments; dark gray – lower sediments; blue-green – other sediments likely to be present in the Devonian sequence, but not yet identified in outcrop or drill holes.

B) A compressional event (heavy black arrows) causes initial asymmetric folding typical of early stages in the development of a fold-thrust belt. Dashed line shows where incipient thrust truncation will develop.

C) Cambrian ophiolitic basalt, ultramafic rocks (serpentinite), and gabbro (green) along with tectonic thrust wedges of chert (Amy Creek) and other sediments (pale yellow) are thrust over the folded Devonian volcano-sedimentary sequence. The thrust surface is undulatory but overall is subhorizontal in orientation. ITH geologic staff are currently attempting to establish if this event happened in the Cretaceous as part of the deformation event that impacted the Manley Basin to the south or if it is the product of an earlier, possibly Permian deformation event. Dashed lines show where the next stage of faulting occurs.

D) Continued (?) thrusting causes thrust stacking along structures that dip 30-45 degrees. Earlier folds and the Cambrian-Devonian thrust surface are segmented with reverse offset.

E) Tectonic relaxation after thrusting or a tectonic extensional event following fold-thrust compression allows for normal offset, particularly along some pre-existing faults, particularly the most recent thrust faults shown in D.
F) Cretaceous dikes (red) of various composition and crystalline character infiltrate the region, particularly along pre-existing faults that dip 30-45 degrees. Dikes intrude all rock types and generally do not occur along the earliest thrust surface that separates the Cambrian ophiolite sequence from the Devonian volcano-sedimentary sequence.
G) Erosion to the current topography removes much of the over-thrust Cambrian ophiolitic sequence. Also, other faults such as the Lillian Fault (steep fault at far right) may have formed during or after extensional tectonism. This fault separates like rocks but with different orientations.

these features is not fully understood. Structural measurements in drill core indicate that the dominant dike orientation is east-west with dips 30-50 degrees to the south.

Many of the dikes are in faults or are bounded by faults suggesting that they, at least partially, follow thrust faults. Measured fault orientations in core reveals a broad scatter of attitudes but with clustering generally coincident with dike orientations. This pattern of partial coincidence between dikes, faults, and mineralization envelopes reinforces the interpretation that the dikes and faults are key controls for mineralization.

Despite these apparent relations, mineralization in sections 428625, 428850, 428925, and 429675 appears to follow, in particular, the Devonian volcanic unit as well as lie oblique to the thrust fault contact between rocks of the Cambrian ophiolite and the Devonian assemblage (**Figures 7.8 – 7.11**). Although it is not possible to reliably correlate individual dikes between the drill holes on these sections, it is clear that the 30-50 degree dip of the dikes and associated structures is compatible with the southerly dipping zones of mineralization. These relationships need further evaluation. Improved understanding ought to offer predictive information for the location of more mineralization.

8.0 Deposit Types

Gold occurs in vein, veinlet, and disseminated styles of mineralization. It occurs in and adjacent to narrow (≤10 cm) multistage quartz veins dominantly in volcanic rocks, but also in intrusive, sedimentary, and ophiolite rocks, generally in or near intrusive dikes and sills. Gold also occurs as diffuse mineralization through the same rocks without a clear association with quartz veins. Many of the dikes appear to fill thrust-related structures and some of the diffuse mineralization occurs in envelopes around these zones.

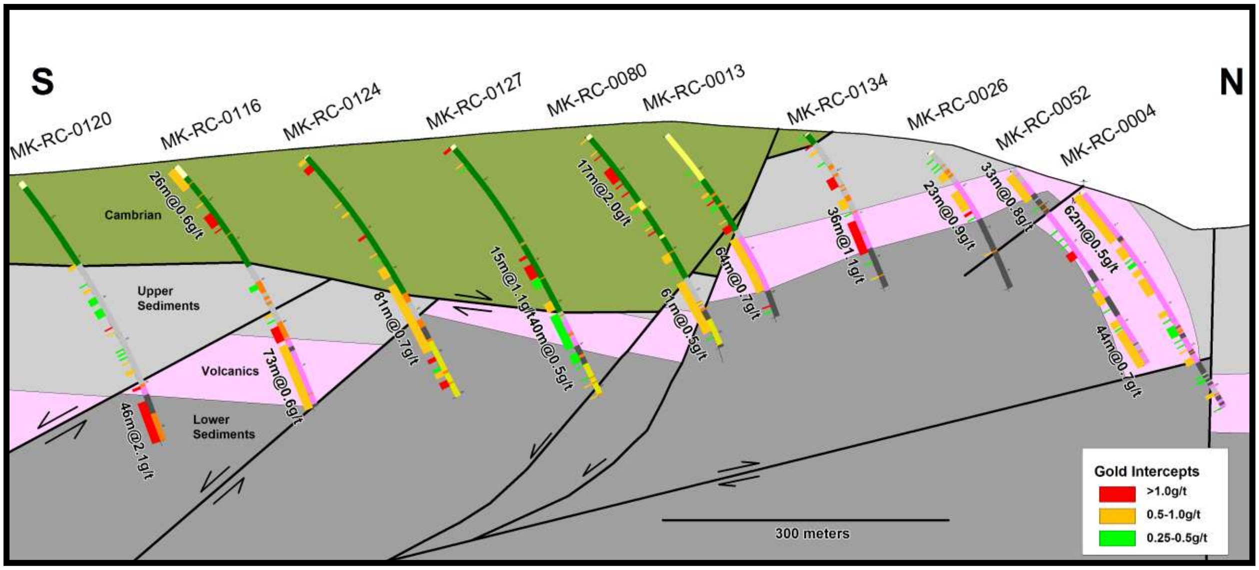

Figure 7.8. N-S Section 428625 E illustrates the complexities of thrust and normal fault interpretation and shows the southerly dip of high grade zones (red).

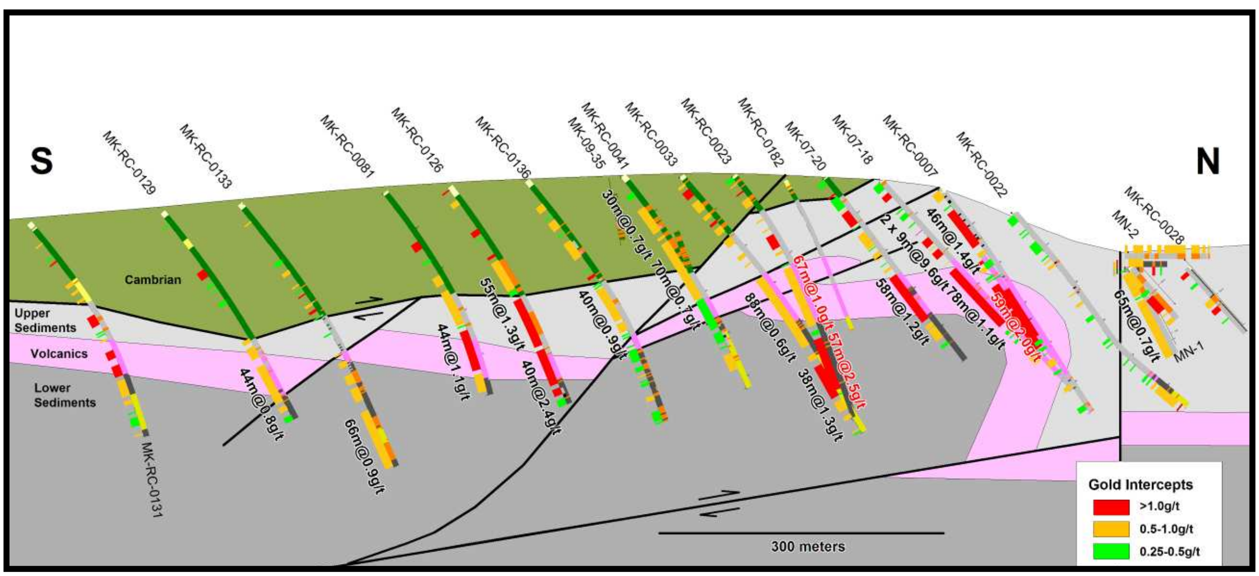

Figure 7.9. N-S Section 428850 illustrates the southerly dip of high grade zone (red) along the general stratigraphic pattern.

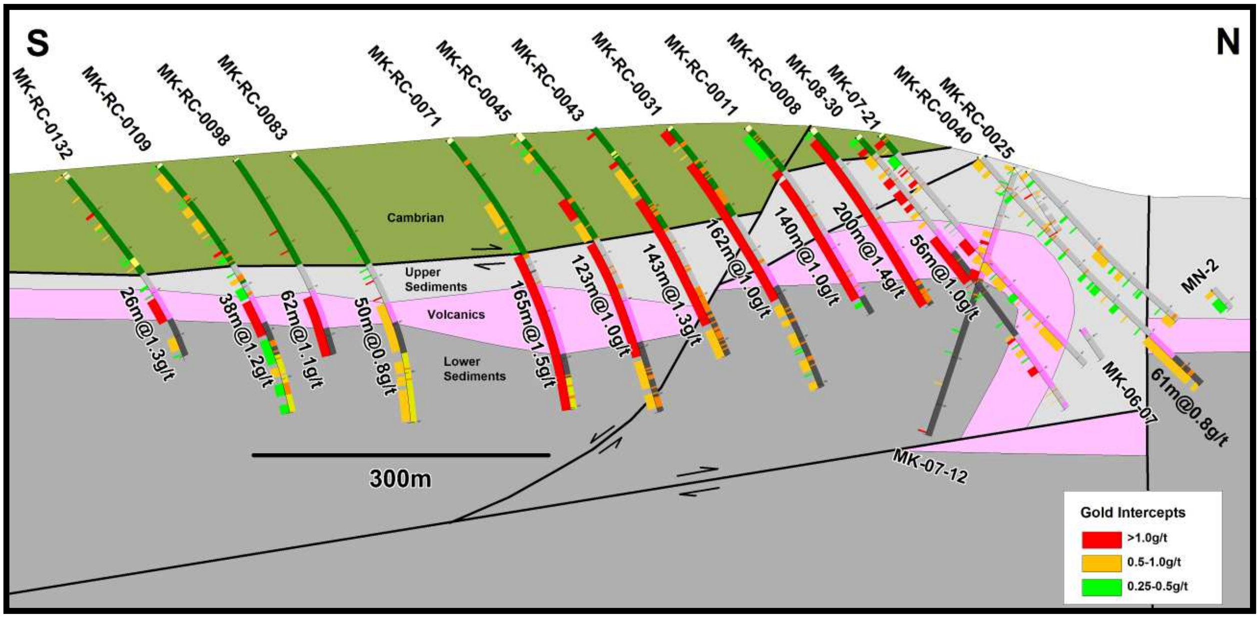

Figure 7.10. N-S Section 428925 illustrates the general southerly dip of mineralization and how it lies along the stratigraphic and structural grain.

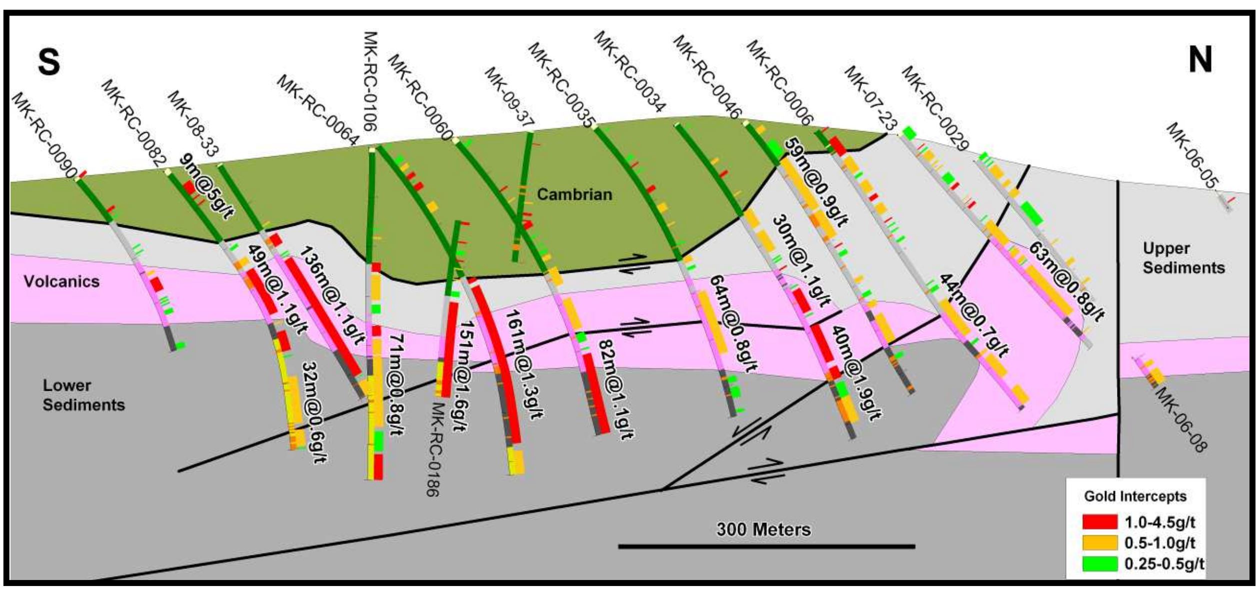

Figure 7.11. N-S Section 429075 illustrates the pattern of mineralization reflecting structural and stratigraphic controls.

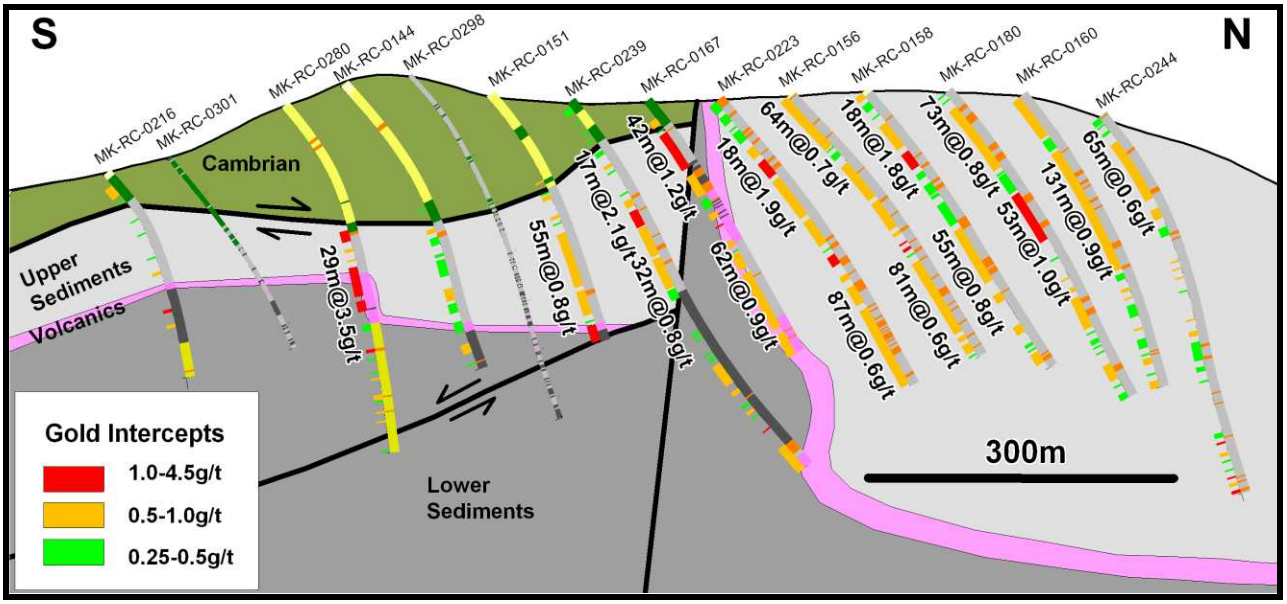

Figure 7.12. N-S Section 429675 illustrates the pattern of mineralization reflecting structural and stratigraphic controls.

The structural architecture, host lithologies, styles of alteration, inferred fluid chemistry, and metallogenic association of As, Sb, ±W, Bi, and very minor Cu and Zn at Livengood show similarities to several styles of gold mineralization and deposit types. Principal among these is the occurrence of Livengood in the Tintina Gold Belt where gold mineralization is hosted in or genetically associated with mid- to late-Cretaceous reduced I-type intrusions (Newberry and others, 1995; McCoy and others, 1997). Mineralization at Livengood appears to be associated genetically with 91.7 – 93.2 m.y. back-arc Cretaceous dikes (Athey and Craw, 2004). For this reason, Livengood should be considered most closely aligned with intrusion-related gold system (IRGS) type deposits.

Among deposits of the Tintina Gold Belt, Livengood mineralization appears to be most similar to the dike and sill-hosted mineralization at Donlin Creek deposit where gold occurs in fine quartz veins associated with dikes and sills of similar composition (Ebert, et al., 2000). However, unlike Donlin Creek, the gold at Money Knob is not tied up in the lattice of arsenopyrite. Instead, it occurs as native gold grains in and around the pyrite and arsenopyrite grains.

The gold-arsenopyrite-stibnite metal association hosted, in part, by sedimentary rocks with dikes associated with a thrust fault system is also reminiscent of sediment-hosted disseminated deposits (SHD) of the Great Basin (aka Carlin type deposits). Foster (1968) initially proposed this potential similarity of mineralization types and Poulsen (1996) speculates on the potential of this type of deposit in the Canadian Cordillera which overlaps in its northern portion with the Tintina Gold Belt. While there are similarities, Livengood lacks prolific decalcification, jasperoid, and a moderate to strong Hg association which are important characteristics of SHD-type deposits. The association of mineralization with intrusions and possible similar structural preparation for both deposit types may be important.

Vein and diffuse gold mineralization along with the metallogenic association and alteration types are most consistent with IRGS type deposits. The mineralogy, alteration types, and geochemical association of As-Sb suggests mineralization formed at a crustal level higher than mesothermal depths (~5-10 km) and deeper than shallow epithermal systems (≤3 km).

9.0 Mineralization

9.1 Mineralization

Historically, the Livengood district has been known for its >500,000 ounce placer gold production. The source of this gold is unknown, but the principal drainages which fed the placer gravels are sourced from Money Knob and the associated ridgeline. Prospecting in this area has revealed numerous gold-bearing quartz veins, generally associated with dikes, sills and stocks of monzonite, diorite, and syenite composition. The reduced magma type and porphyritic to brecciated textures as well as local zones rich with arsenopyrite, are characteristics common to many deposits of the Tintina Gold Belt (e.g. Brewery Creek, Donlin Creek) (McCoy, et al., 1997; Smith, 2000).

No lode production has taken place at Money Knob. Exploration of the area by various companies, including soil surveys by Alaska Placer Development, Cambior, AGA and ITH, reveals a 6 x 2 km northeast-trending anomalous area in which a 2.2 x 1.5 km area (~25% of the anomaly area) forms the locus of current exploration interest (**Figure 9.1**). Despite drilling of 311 holes to September 25, 2009, this area has been only partially drill tested. At this time, mineralization shows local fault and contact boundaries such as the Lillian Fault, but overall is open in all directions.

Drilling since 2003 by AGA and ITH has resulted in identification of an indicated and inferred gold resource interpreted to be part of a large IRGS deposit, the details of which are discussed further in section 17.

Mineralization consists of gold in multi-stage quartz, quartz-carbonate, and quartz-carbonate-sulfide veins and veinlets as well as disseminated throughout altered rock with arsenopyrite and Fe-sulfides. Gold mineralization preferentially occurs in Devonian volcanics and Cretaceous dikes but also occurs in Upper and Lower Sediments as well as locally in the overthrust ultramafic rocks primarily where dike rocks are present. Mineralization associated with Cretaceous dikes may also be spatially associated south dipping faults. Overall, the mineralization envelope appears to dip south along with the dikes and faults.

Better gold values (>1 g/t) tend to be associated with the Devonian volcanics, Cretaceous dikes, dike margins and in broad zones within adjacent volcanic and sedimentary or mafic-ultramafic rocks. Visible gold occurs locally, particularly in quartz veins and with isolated coarse blebs of arsenopyrite and/or stibnite. Where gold occurs in sedimentary host rocks, veins are most common in brittle

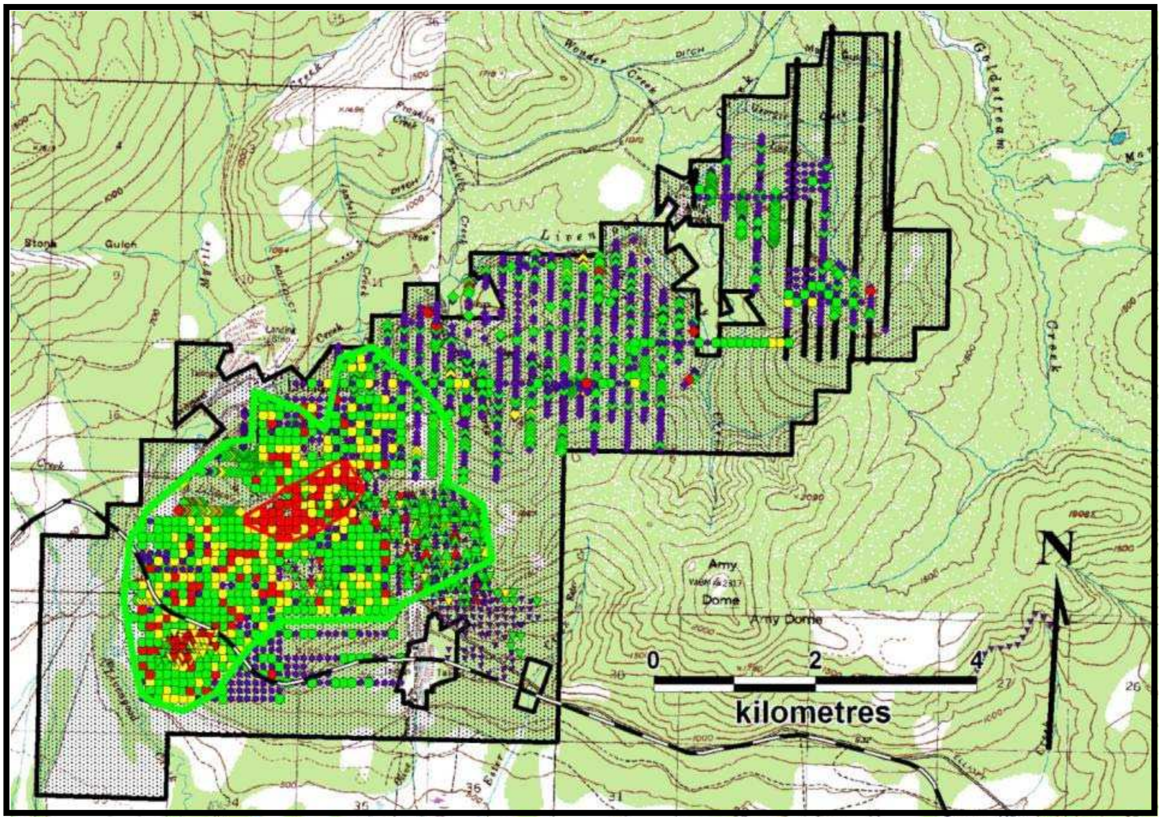

Figure 9.1. Plot of soil samples. Color coding shows relative gold content with red indicating gold ≥0.100 g/t Au. The green line encloses the area containing anomalous gold samples.

siltstone, sandstone, and pebble conglomerate as opposed to shale. The diffuse style of mineralization is spatially associated with areas containing vein mineralization, but gold can be present where there is no discernable quartz veining to explain it.

Gold is strongly associated with arsenopyrite and locally with stibnite although stibnite is relatively rare. Other metallic minerals include pyrite, pyrrhotite, and marcasite. Some pyrite may be arsenian. Small amounts of chalcopyrite and sphalerite are observed in thin section and locally in core. Small amounts of molybdenite have been reported by previous workers.

Mineralization appears to be contiguous over a map area approximately 2.5 km square and ranges up to 200m thick. On the south side of the Lillian Fault, individual mineralized envelopes are tabular and follow lithologic units, particularly the volcanics, or lie in envelopes that dip up to 45 degrees to the south and follow the structural architecture and dikes. On the north side of the Lillian fault mineralization is similar in style, but more widespread in steeply dipping Upper Sediments. Interestingly, visible gold has been noted more often in Sunshine Zone mineralization north of the Lillian Fault.

9.2 Alteration

Rocks of Livengood have undergone multiple stages and styles of alteration. As increased drilling reveals a wider range of subsurface material, complex overprinting and spatial relations for different stages of alteration are becoming apparent. Four principle alteration styles are currently observed. These are identified by each stage's principal alteration mineral; biotite, albite, sericite, and carbonate. Two other lesser styles of alteration also may be present. Local zones of smectite-illite alteration and local presence of possible minor pyrophyllite is curious and may be important, but absolute identification has not been made and it is unclear at this time where and how that mineral fits into the sequence.

Biotite alteration consists of fine-grained remnant patches of secondary biotite in sedimentary, volcanic, and dike rocks or as phlogopite (phlogopitic biotite?) in mafic and ultramafic rocks (**Figure 9.2 and 9.3**). Pyrrhotite and quartz accompany the biotite. Arsenopyrite may be in rocks with this type of alteration, but timing relations are not clear. Macroscopically, the secondary biotite renders a weak to dark brown hue to the rock or margin to some veinlets. All rock types have been affected by this stage of alteration, however, secondary biotite and accompanying pyrrhotite are observed only as remnant patches in local intervals in some drill holes where subsequent alteration stages have not obliterated it.

Albite alteration occurs as extensive replacement of volcanic and dike rocks and overprints biotite alteration. Secondary albite occurs as intergrown radiating plumose to acicular sheaves and rosettes that locally replace all previous rock textures (**Figure 9.2 and 9.3**). Albite is accompanied by intergrown fine-grained dark gray to black patches and grains of quartz. This quartz is cryptocrystalline with an almost cherty character. The dark color may be from included carbonaceous material (Sillitoe, 2009). Albite alteration appears to be accompanied by disseminated arsenopyrite and pyrite mineralization.

Sericite alteration consists of pervasive sericitization, sericite veins, and quartz-sericite envelopes around quartz±sulfide veins in all rock types. Sericite cross-cuts and/or replaces all previous alteration minerals, and may, in some part, be developed from destruction of secondary biotite. Pyrite and arsenopyrite accompany this stage, some of which may result from pyritization of biotite-stage pyrrhotite. In mafic and ultramafic rocks, tremolite and local fuchsite are the dominant sericite-stage phyllosilicates. In addition to the previously described black silica that accompanies albite alteration, fine-grained introduced quartz is widespread in many thin sections and replaces primary mineralogy. However, this form of silica is rarely observed macroscopically due to other alteration minerals which are more readily apparent. Sericite-stage silica also occurs as the inner zone of centimetre-scale alteration selvages around narrow fractures.

Smectite-illite alteration has been observed in a number of locations, generally in and around brittle fault zones, but is not as widespread as the albite and sericite alterations. It is characterized by bleaching of the affected rocks and strong swelling and consequent disentegration of core samples from these zones. The alteration has been observed most commonly in sediments and dikes. Pyrite and arsenopyrite are disseminated through the alteration and gold grades of several hundreds of ppb are common.

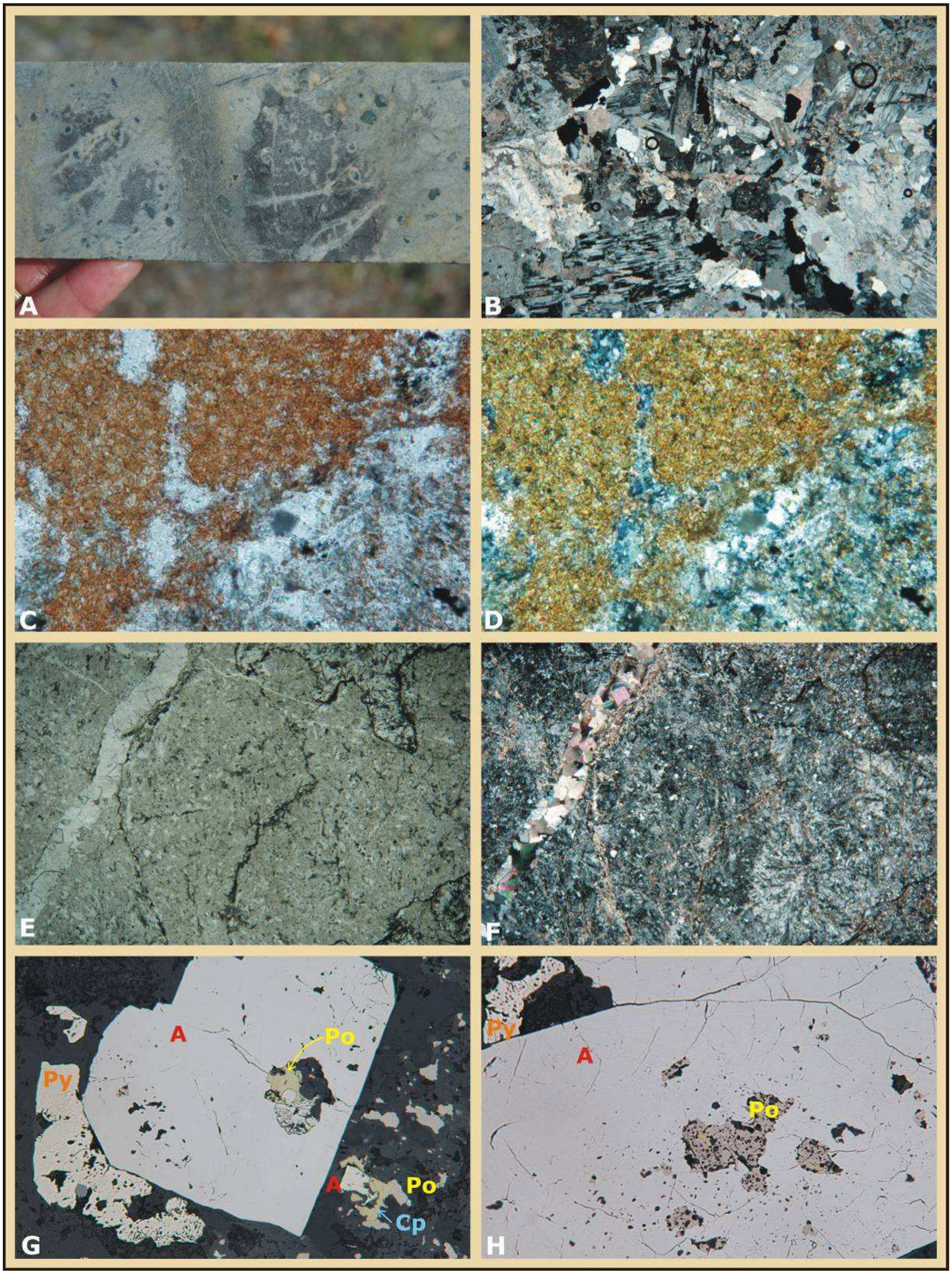

Figure 9.2. Photomicrographs of characteristic alteration among rocks at Money Knob. **A**) View of core showing relict patches of secondary biotite (dark color) cut by and overprinted by albite and

sericite alteration. 08-33, 190.25 **B)** rare, relatively weakly altered Cretaceous intrusive dike with abundant interlocking plaigioclase laths and blocks; Weak sericite and carbonate alteration are present. Some of the plagioclase may be in the early stages of being altered to secondary albite. 09-34, 252.76. **C and D**) plane and polarized light examples of a patch of secondary biotite in Devonian volcanics; sericite and carbonate are also present in the lower right portion of the photo; 200x; 8-33; 190.25. **E and F)** A quartz-carbonate veinlet crosscuts albitized volcanic rock (MK07-18, 247.5m). **G**) Large arsenopyrite grain (A) with an inclusion of pyrrhotite (po), and adjacent to pyrite (py). Minor chalcopyrite (cp) occurs in the lower right. 200x, 08-33, 230.55.

Carbonate alteration consists of at least three styles of introduced carbonate: 1) clear but fine-grained scaly patches and flakes throughout the rocks; 2) fine-grained cloudy carbonate patches; and 3) clean large euhedral rhombs and clusters of rhombs in and adjacent to carbonate-quartz-sulfide veins. Some very fine carbonate is brown in color. It is not clear whether this is a natural color or a product of oxidation or overgrowth and incorporation of very fine secondary biotite. Macroscopically, some brown carbonate has been mistaken for secondary biotite. A fourth style of carbonate consists of very late calcite veinlets which crosscut all features. These could be the product of late-stage cool hydrothermal alteration or supergene. The vast majority of carbonate appears to overprint previous alteration stages, however, some may accompany earlier alteration stages. Carbonate abundance ranges from scattered flakes to complete replacement, particularly in the mafic and ultramafic rocks. In the sedimentary rocks, it is difficult to determine if some carbonate is redistributed primary carbonate or introduced hydrothermal carbonate. Local marl and limey beds occur in the Devonian sediments. Carbonate apparently consists of dolomite and other Fe- Mg species of carbonate such as sideriteand ankerite. Arsenopyrite and pyrite are common in carbonate-quartz veins and veinlets.

9.3 Synthesis of Mineralization and Alteration

The types of alteration stages and their sequence are consistent with other IRGS deposits and prospects of the Tintina Gold Belt (Newberry and others, 1995; McCoy and others, 1997). This is important as it strongly supports the interpretation that mineralization at Livengood is part of an intrusion-related mineralizing system. Although it is possible that each alteration stage is the product of independent hydrothermal events, the mineralogy of each alteration type suggests that the various stages formed as part of an evolving, cooling system with initial biotite and pyrrhotite being the highest temperature and subsequent lower temperature assemblages following (**Figure 9.4**). This patterning can also be interpreted as consistent with the chemical evolution of hydrothermal fluids emanating from an intrusive source.

Gold shows a strong correlation with arsenopyrite. However, arsenopyrite has been introduced at least at the biotite alteration stage and significantly at the carbonate stage. Some amount of arsenopyrite also may have been introduced at the albite and sericite alteration stages. It is unclear, though, whether gold has been introduced during all of these stages or mostly during a particular stage. Understanding these relationships is part of ITH's current exploration program.

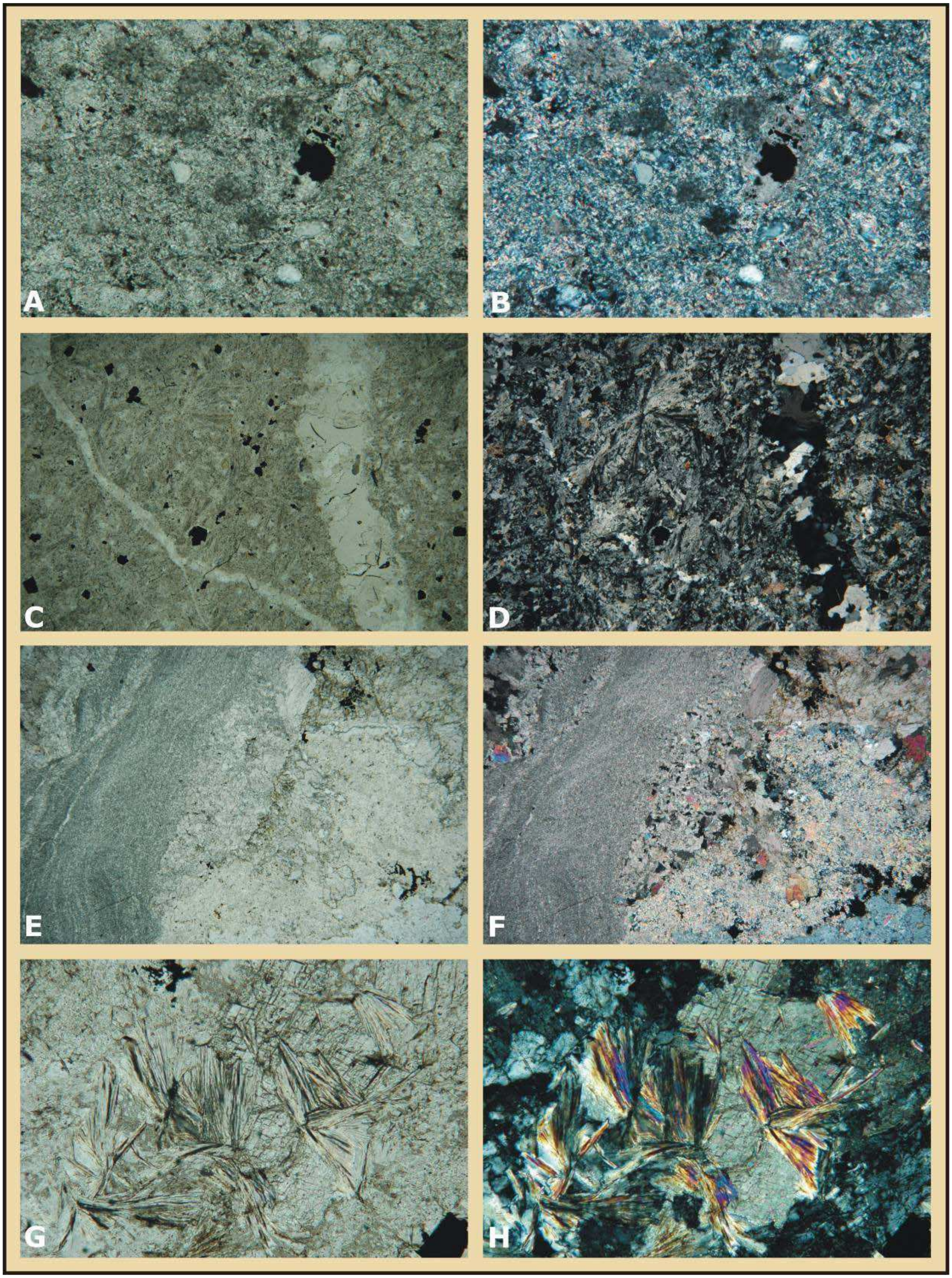
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Figure 9.3. Photomicrographs of characteristic alteration among rocks at Money Knob. Plane light on the left; crossed polarized light on the right. **A and B**) Sericite and carbonate replace a silty phyllite (MK07-18, 76.0m). **C and D**) A quartz-carbonate veinlet crosscuts albitized volcanic rock (MK07-18, 247.5m). **E and F**) Carbonate (upper left 2/3rds of section) and tremolite (lower right 1/3 of section) replace mafic rock. 25x; 02-21, 19.35. **G and H)** Possible pyrophyllite sprays appear to be after carbonate. Other examples appear to be before carbonate. 200x; 07-20, 61.2.

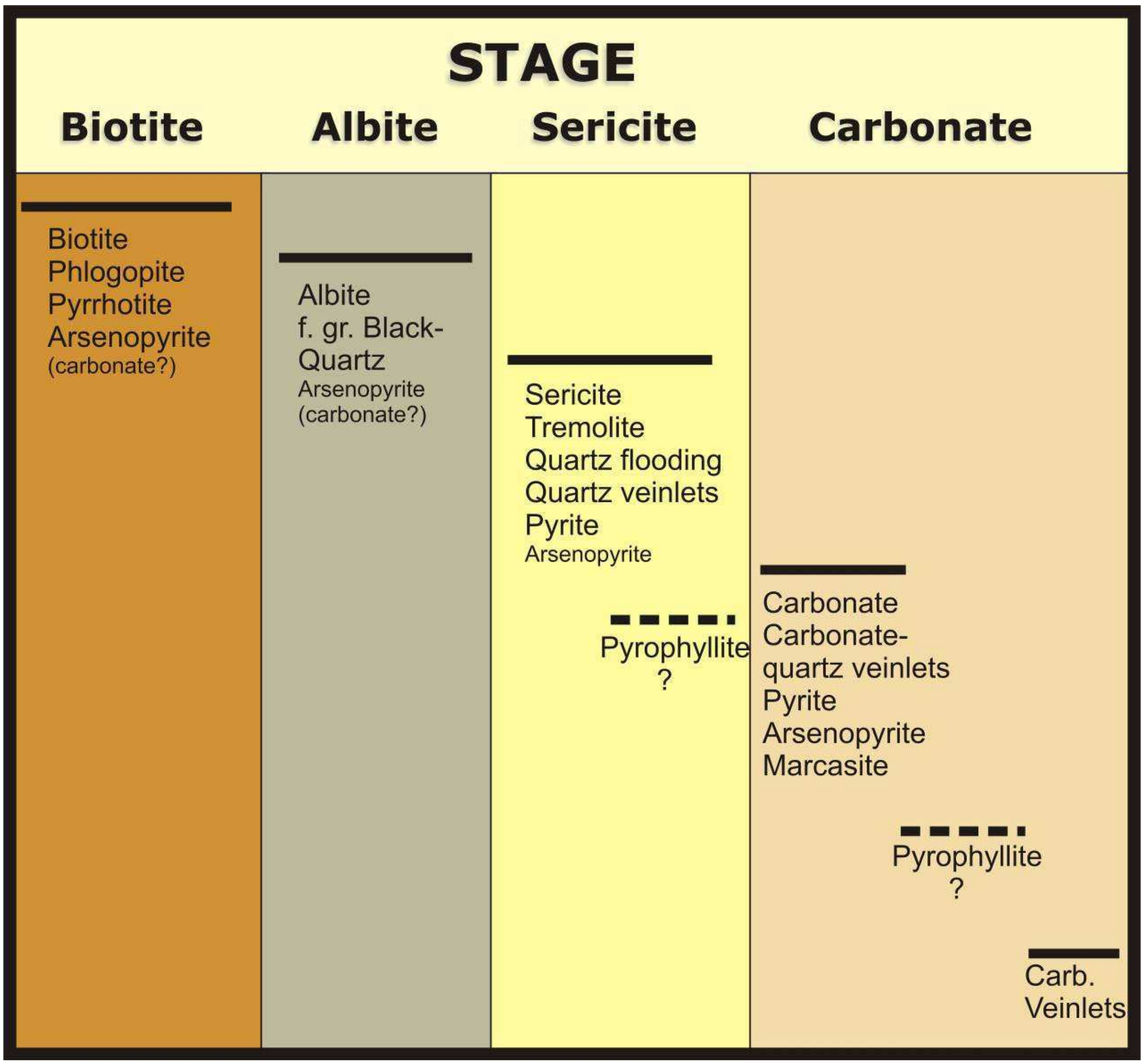

Figure 9.4 Interpreted paragenetic sequence of key alteration and mineralization stages. Gold occurs with arsenopyrite and may have been introduced during all stages or dominantly during a particular stage(s). It is not clear where smectite-illite alteration might fit into this sequence.

10.0 Exploration

10.1 Past Exploration

Several companies have explored the Livengood area as outlined in Section 6 (History). That work identified a sizeable area of anomalous gold in soil samples and intervals of anomalous gold mineralization in drill holes (described in previous sections).

ITH advanced the soil sampling coverage in 2006 and 2007 by collecting an additional 361 samples. These samples helped improve definition of anomalous gold in soil on the southwest side of Money Knob and between Money Knob and Radio Knob.

ITH undertook drilling of the surface geochemical anomalies in 2006 with favourable results. In 2007, the area was drilled sufficiently to produce a resource evaluation (Giroux, 2007; Klipfel and Giroux, 2008a) and a program for 2008 was planned that would further that evaluation. Drill results through September 27, 2008 were used as part of a revised resource evaluation in October, 2008 (Giroux, 2008; Klipfel and Giroux, 2008b). Geochemical results received and drilling completed after that date were used for a subsequent resource update (Giroux, 2009; Klipfel and Giroux, 2009). Results from 34 reverse circulation holes drilled in the winter of 2009 were primarily infill holes. Data from these holes were applied to a new resource estimate which also incorporated advancements in modeling the deposit (indicator kriging) and resulted in upgrading and enlarging the resource estimate to 4.04Moz and 3.6Moz in the indicated and inferred categories respectively (Klipfel, Barnes, and Pennstrom, 2009). In addition, this midyear 2009 report included initial metallurgical findings and included initial cost estimates which were incorporated into the resource evaluation.

10.2 Current Exploration

ITH has continued to conduct step-out and infill drilling throughout 2009. This report includes the results from drilling as received through September 25, 2009. This data includes results from 103 new holes (98 RC and 5 diamond core) as well as all past data (142 RC holes and 29 diamond core holes). Results have been used in a new resource estimate reported in Section 17, and include further advances in metallurgical understanding and improved cost estimates which have been incorporated into the estimation process. Results from an additional 66 holes completed by October 23 will be incorporated into the next round of resource evaluation.

11.0 Drilling

11.1 Past Drilling

All of the companies that have explored at Livengood in the past, except Cambior, have undertaken drill programs to evaluate the district. In each case, drill holes targeted different geologic concepts such as veins in bedrock beneath the alluvial gold. AGA initially, and ITH later, focussed drilling on possible mineralization beneath and down dip from the surface soil anomaly area (**Figure 11.1**).

Drilling since 2003 by AGA and ITH is summarized in **Table 11.1**. Drilling in 2003 by AGA consisted of 1,514 m of vertical and angled reverse circulation (RC) drilling in eight holes. It identified broad zones of gold mineralization (BAF-7; **Table 9.1**). Drilling in 2004 by AGA consisted of 654m of HQ coring in 4 diamond drill holes designed to test for gold beneath the thrust fault at the base of the Cambrian rocks. These holes were up to 1.7 km to the west of 2003 drill holes. They identified thick zones of gold mineralization in Devonian rocks beneath relatively barren, thrust-emplaced Cambrian rocks (MK-04-03; 96m@>0.5 g/t in 2 intersections). These results highlighted the fact that significant mineralization could exist beyond the limits of the main soil anomaly, particularly in blind locations beneath thrust faults.

No drilling took place in 2005.

In 2006, ITH drilled 1,230m of core (HQ) in 8 holes and continued to demonstrate the presence of mineralization over a broader area. The 2007 campaign consisted of 14 diamond drill holes for a total of 4,400m. These holes focused on extending and defining the geologic setting of mineralization first recognized in MK-04-03. This mineralization was originally thought to be hosted primarily in the Devonian volcanic rocks. However, as drilling has progressed, it has become clear that mineralization is strongest in the volcanic rocks, but occurs in all rock types at Money Knob (**Figure 11.2**).

Based on favourable results in 2007, the 2008 program consisted of 30,653m of RC and core in 108 RC and 7 core holes. These holes were designed to improve definition and expand the resource calculated early in 2008 based on 2007 drill data. The 2008 drill program did not identify limits to mineralization in any direction. Instead, a thicker mineralized zone was identified (up to 200m; **Table 9.1**). In addition, this campaign highlighted the fact that mineralization occurs in all rock types, not just in Devonian volcanic rocks. This was important as it indicated that there was potential for broader extent of mineralization than envisioned prior to the 2008 drill program.

The winter 2009 program helped fill in gaps in the drilling grid and enabled increased continuity of information for improved resource estimation. In addition, more rigorous estimation procedures using indicator kriging, improved modeling of the oxidation profile, recoveries of various lithologic types, and cost estimates based on comparable pit mining techniques in this environment.

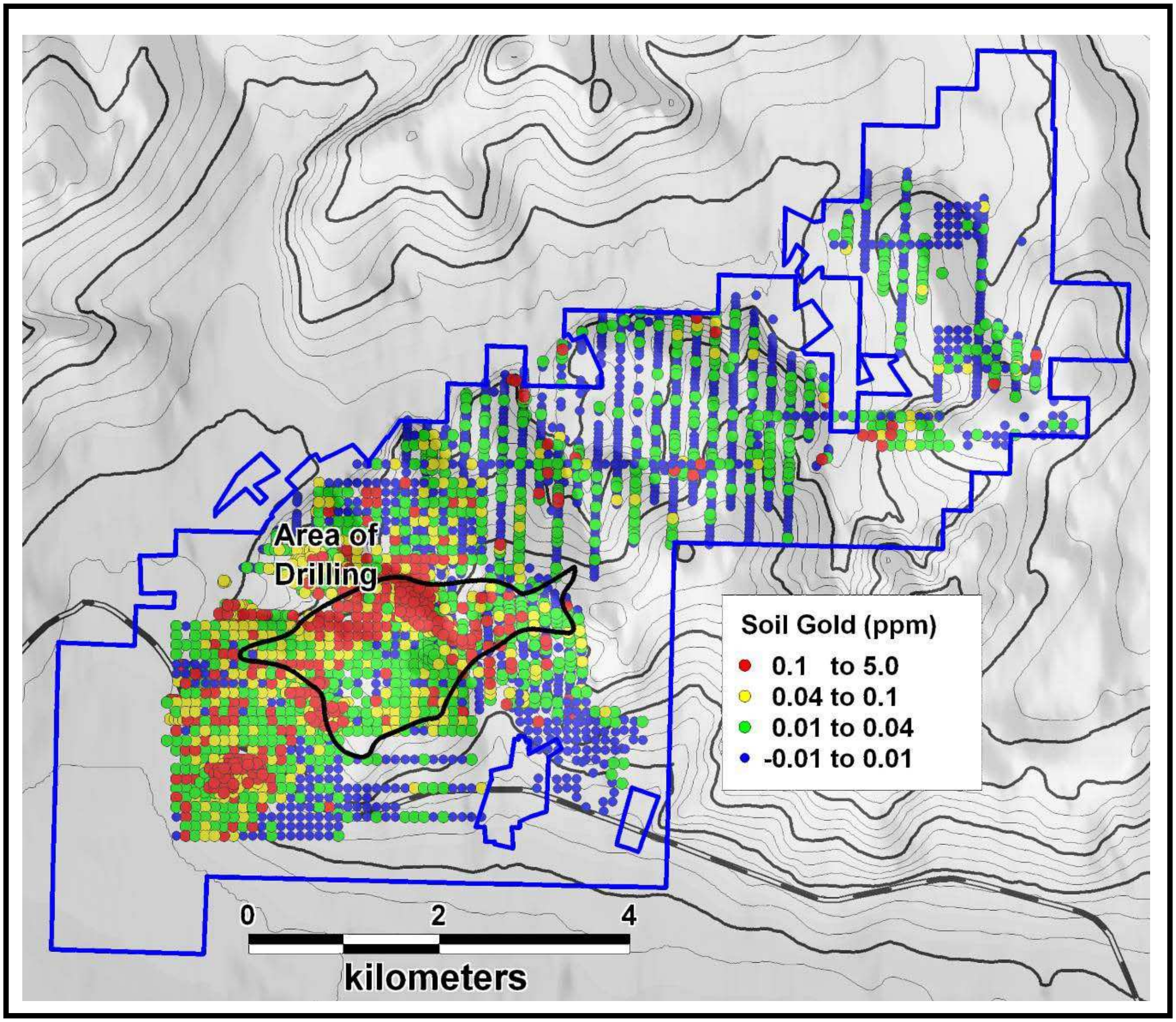

Figure 11.1 Distribution of drilling in the Money Knob area with respect to anomalous soil samples. The majority of the soil geochemical target remains untested.

TABLE 11.1
SUMMARY OF AGA AND ITH DRILLING AT LIVENGOOD

Year	DDH	m	RC	m	Results
2003	-	-	8	1,514	Broad zones of Au mineralization
2004	4	654	-	-	Discovered Devonian volcanics as preferential host rock
2005	-	-	-	-	No drilling
2006	8	1230	-	-	Drilled first >100gram meter intersection in Devonian volcanics
2007	14	4,400	-	-	Defined continuity of volcanics and mineralization. Discovered first sediment-hosted mineralization
2008	7	2,040	108	29,040	Discovered core zone where sericite alteration mineralizes all rock types. Delineated 6.8M oz indicated and inferred resource
Spring 2009	-	-	34	9,650	Continued definition of core zone. Discovery of SW extension.
Summer 2009	6	2323	99	30,614	Expanded the scope of the mineralization and provided input for the resource evaluation reported in this report

11. 2 Current Drilling

ITH has been actively drilling from June through October in the current program. The resource estimate along with other information presented in this report is based on all previous data as well as data received up to September 25, 2009. This includes the addition of data from 99 new RC drill holes (30,614m) and 6 diamond drill holes (2323m). Drill holes with incomplete assays and drill holes completed after September 25, 2009 are not included in this work. These holes were drilled in locations to offer additional fill-in data, but more importantly, they expanded the area of known mineralization through identification and assessment of the new Sunshine Zone north of the Lillian Fault and about 600m east of the Core Zone. Mineralization between these two zones is contiguous and open in all directions with the exception of a partial local margin formed by the Lillian Fault.

During the summer of 2009, 6diamond drill holes were drilled across the NNW-trending Core Zone in order to better understand the structural controls and to test the depth continuity of the mineralization. Three holes were drilled to the northeast and three were drilled to the southwest. This drilling confirmed that the Core Zone is the locus of a swarm of 0.2 - 1.0m thick southerly dipping dikes. In addition, a number of larger (≤10m thick) steeply dipping NNW-trending dikes were observed suggesting that ENE extension may have occurred at about the time of dike magmatism. Quartz-arsenopyrite and quartz-stibnite veins were encountered and display SW dip and N-S, subvertical orientations respectively. In general it appears that the broad zones of mineralization in the core zone

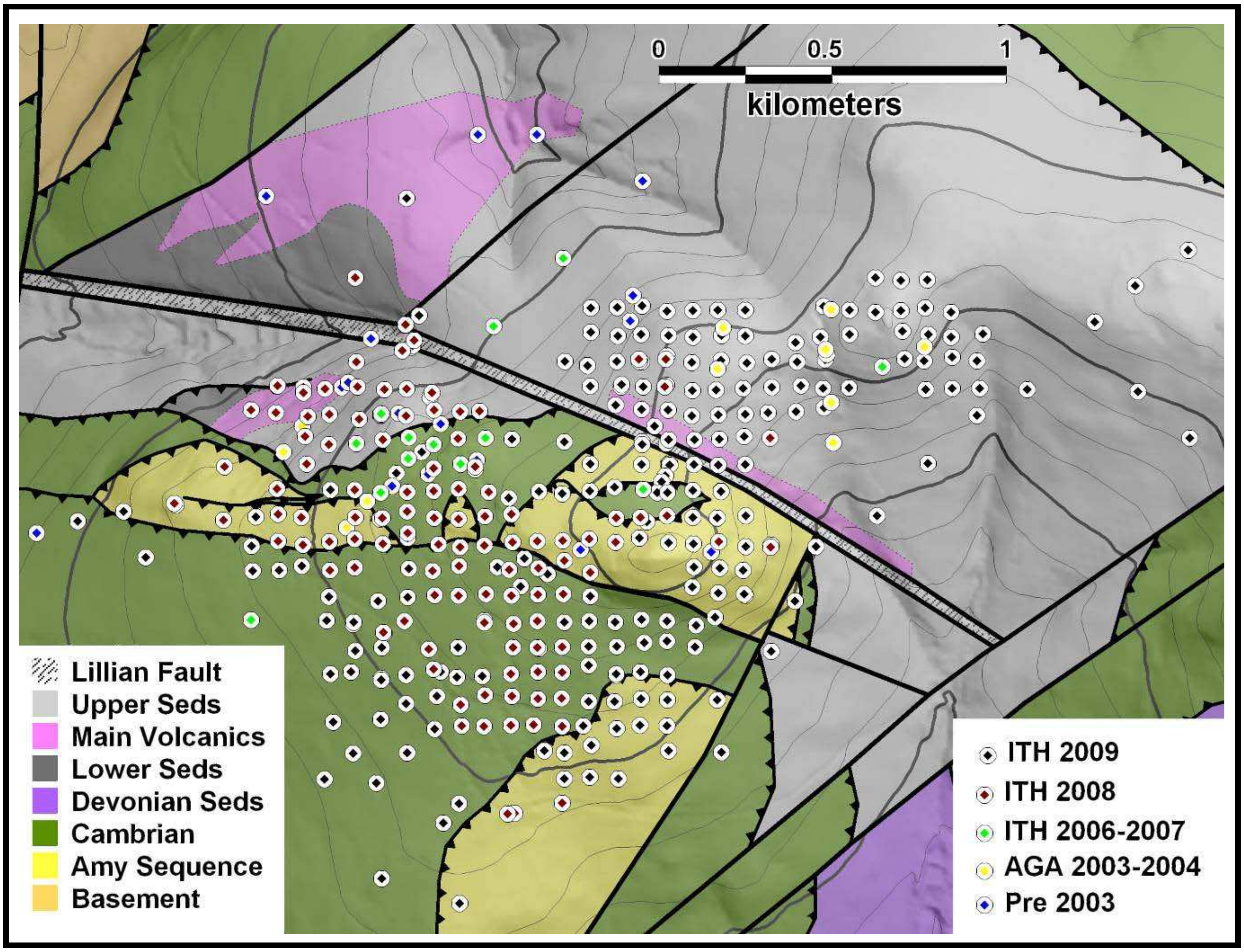

Figure 11.2 Distribution of drilling in the Money Knob area according to year and company.

do not continue to depth, but narrower very high grade structures were encountered below the Core Zone and represent a potential future target.

As part of an overall step-out program, a NE-trending fence of 11 holes spaced at approximately 150 meters trending N45E and dipping 55 degrees was drilled. The holes confirmed the stratigraphic continuity of the area but failed to delineate major zones of mineralization. However, the farthest NE holes, representing a step out of 500m from the nearest grid drilling encountered multi-gram mineralization associated with the faulted contact between the Devonian and the Cambrian mafic rocks.

In the winter of 2009-2010, ITH will be undertaking a drill program within areas that are sensitive to disturbance during the summer season. This drilling will allow the infill of information in key areas not accessible for drilling in the summer. The goal is to test areas further to the southwest. If successful, this drilling will expand known mineralization further to the southwest.

11.3 Drill Procedures

To date, virtually all drill holes at Money Knob have been drilled in a northerly direction at an inclination of 50 degrees in order to best intercept the south dipping structures and mineralized zones as close to perpendicular as possible. A few holes have been drilled in other directions as described above. Most holes have been spaced at 75m along lines 75m apart. A few holes are more closely spaced. Surveys of the holes show that with depth, holes steepen 10-20 degrees depending upon the length. Most holes have been drilled to depths of 250-300m.

Diamond drill core is recovered using triple tube techniques to ensure good recovery and confidence in core orientation. Recovery is excellent being greater than 95% over the course of the entire program. The core is oriented using the ACT system and/or the EZ Mark tool. Core is marked so that a continuous line is located along the base of the core as long as core pieces can be matched continuously from the marked top of the run. Subsequent runs are matched also. Oriented core is important for recovery of structural, vein, and contact orientation information and is essential for interpreting fault and dike orientations on sections.

After marking the core for its orientation, the drill core is placed as an entire run in a case of prepared PVC pipe and sealed until opened by core loggers at ITH's core shed. This custom procedure has been implemented to assure minimal breakage or crumbling of core between retrieval from the hole and transfer to boxes by the logging geologist. Core is cleaned, measured, marked, labelled, and logged by contract geologists from Northern Associates, Inc.

Reverse circulation holes are bored and cased for the upper 0-30m to prevent downhole contamination and to help keep the hole open for ease of drilling at greater depths. Recovery of sample material from RC holes is done via a cyclone and dry or wet splitter according to conditions. Sample chips are split into 3 recovery points (**Figure 11.3**): one is the interval sample, the second is an equivalent split "met" sample, and the third smaller split is used to collect chips for logging purposes. These chips are placed in standard chip trays. Samples are collected in porous polybags that allow retention of sample material and evaporative seepage of water from the sample.

Drill hole locations are determined by sub-meter differential GPS surveys at the drill collar. Initial azimuth of drill hole collars are measured using a tripod mounted transit compass in conjunction with a laser alignment device mounted on the hole collar (**Figure 11.3**).

Down hole surveys of core and reverse circulation drill holes are completed using the Gyro-Shot survey instrument manufactured by Icefield Tools Corporation. Precision and accuracy of this method was assessed in 2008 through a series of duplicate surveys using this instrument and by comparison in holes surveyed by the EZ-Shot (magnetic) borehole surveying device. Results of surveys and duplicate tests show normal minor deviation in azimuth and inclination with reproducibility within a close margin of error. In 2009, a duplicate survey performed by the Gyro-Shot instrument measuring the same hole twice (MK-RC-0195 to 985 feet) and a tandem survey performed by running two Gyro-

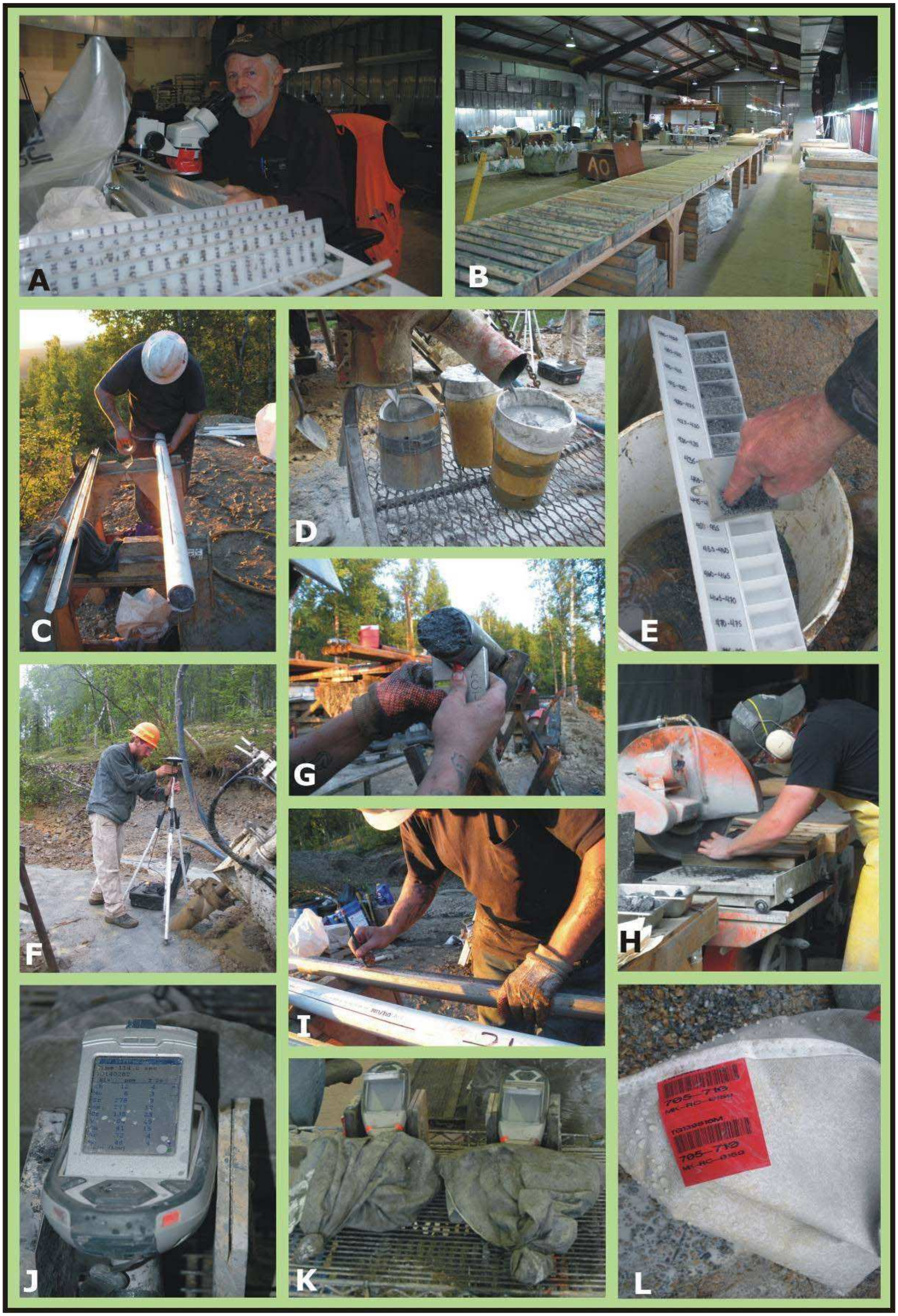
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Figure 11.3 Photos of various exploration functions. A) ITH geologist logging RC chips with a binocular microscope. B) View of ITH'S core shed and core boxes in the foreground. C) Driller taping core securely in PVC holder/carrier. Core barrel parts are on the left. D) RC drilling chips are split into 3 collection points, the sample (foreground bucket), the met sample (background bucket), and the visual chip sieve for logging purposes (left). E) A representative sample of RC chips is retained in chip trays with individual compartments for each 5' interval. F) Drill hole collars are surveyed with a differential GPS instrument. G) The driller marks the core to indicate its oriented position with respect to the core barrel. H) Drill core is sawed in half with a diamond saw at the core shed. I) The driller marks a line along the base of the core to indicate its oriented position. J) Niton portable XRF instrument records trace-element abundances prior to shipment of samples to the lab. K) Trace elements are measured by two NITON portable XRF instruments for all RC samples prior to shipment to the lab for assay and multi-element ICP analyses. L) Example of porous polybag which allows the escape of water, but not sample material. Pre-printed labels indicate drill hole, depth interval, sample number, and bar-coded sample ID information.

Shot instruments simultaneously on the same probe assembly (MK-RC-0178 to 900 feet), demonstrated close replication and agreement between the surveys. The 3-D coordinates at the maximum depth of the paired surveys plot to within 1% of the coordinates in the corresponding survey relative to length of hole surveyed. Drill hole surveys were completed by Northern Associates, Inc. and were observed in the field by Dr. Klipfel.

The RC drilling in 2003 was conducted by Layne Christiansen Company using an MPD 1500 Track RC drill. Drilling in 2004 was also by Layne using a CS1000 core drill. No drilling took place in 2005. In 2006, 2007, 2008, and 2009, diamond core drilling was conducted by AK Drilling Inc, and Layne Christensen. RC drilling was by AK Drilling, Inc., and T and J Enterprises.

12.0 Sampling Method and Approach

12.1 Past Sampling

The sampling procedures of previous companies are not known but the major companies that did the work are known for their conscientious QA/QC protocols. Sample data from past programs are consistent with more recent data generated by AGA and ITH. On this basis, there is no reason to doubt the validity or credibility of samples from Occidental, AMAX, Homestake, or Placer Dome. The similarity of results for each program suggests that sample collection and analytical procedures are sufficiently similar to allow use of their data by ITH in current exploration efforts.

For samples collected by AGA, all soil, stream sediment, rock, and drill sampling was done according to AGA in-house protocols for geochemical sampling. These protocols specified technical procedures for collection and documentation of samples. In general, -80 and -200 mesh material was analyzed for soils and stream sediment respectively. Dr. Klipfel reviewed these protocols in 2006 as well as AGA's security procedures and verified that they met or exceeded standard industry practices. Sampling procedures remained the same through the course of the 2003 and 2004 exploration programs.

All AGA geochemical samples were secured and shipped to Fairbanks according to AGA protocols for sample preparation (drying, crushing, sieving, and pulverizing) at ALS-Chemex in 2003 and Alaska Assay in 2004. Sample splits (300-500g for rock material; -80 mesh for soil samples) were sent to ALS Chemex in Vancouver for analysis. Analytical methods used were standard 50g fire assay with AA finish and four-acid digestion, multi-element ICP-MS. These are standard analytical packages for the exploration industry and are performed to a high standard. Analytical accuracy and precision were monitored by the analysis of reagent blanks, reference material and replicate samples. Quality control was further assured by the use of international and in-house standards. ALS Chemex is accredited by the Standards Council of Canada, NATA (Australia) and also has ISO 17025 and 9001 accreditation.

AGA reverse circulation drill samples were collected at five foot intervals as measured by the driller. Pulverized material from the hole was passed though a cyclone to separate the solids from the drilling fluid and then over a spinning conical splitter. The splitter was set to collect two identical splits each of which weighed 2-5 kg. Representative material was also collected and saved in chip trays for later visual inspection. The split material was put into pre-numbered bags by the drillers' helpers on site. One of the splits was sent for analysis while the other was retained for future reference. Samples were secured and transported to the sample preparation facility of ALS Chemex in Fairbanks for drying, crushing, pulverization, and splitting. 120 gram splits were sent to Vancouver for analysis by standard 50 gm fire assay with AA finish and multi-element ICP-MS. The RC chips were logged by project geologists by recording basic information on the lithology, alteration, and mineralization for each interval.

AGA's core material was collected at the drill site and placed in core boxes under the supervision of an experienced geologist and Qualified Person for the purposes of NI 43-101. It was logged for rock type, alteration, structure, and with detailed descriptions. Dr. Klipfel examined the core logs and core from the four 2004 holes and can verify the reliability of the logging. Sample intervals were determined on the basis of the distribution of veining and alteration with a minimum sample width of 30 cm and the maximum width of 1.5m. Samples were collected to isolate different components of the alteration and mineralization to characterize them.

After the samples were marked, the core was sawed in half, and one half sent for analysis. The other half was either kept on site or at AGA's core storage facility in Fairbanks. The average recovery in the core program was in excess of 90% and there is no indication that poor recovery is an issue in the interpretation of the assay data. Sampling was selective but barren samples were always collected to bracket zones of mineralization so that reliable boundaries could be defined in the intercepts. Dr. Klipfel examined this core in the course of the site visits.

12.2 Current Sampling

ITH has adopted and continued the sampling protocols used by AGA and described in the previous section, with the exception that all drill holes are sampled from surface to total depth. In addition, ITH has implemented a number of customized steps in their procedures to minimize errors and assure the integrity of sample material. This assures a high level of reliability in the sample data set and assures continuity of methodology, laboratory standards and conventions as well as confidence in the data generated. All core samples are weighed prior to shipping to the ALS-Chemex facility in Fairbanks. These weights are compared to the laboratory received weights to confirm that the samples were logged in correctly. RC samples are collected in pre- numbered, bar-coded bags (**Figure 11.3**). They are logged in on-site by ITH using the barcodes to prepare the shipments and ALS Chemex uses the same barcodes to log the samples into their system. The sample weights are recorded at various stages in the preparation process. These procedures minimize labelling and other potential errors and add an extra level of assurance that the sample is tracked correctly and matched with the data generated by that sample.

Core samples are no longer placed in a core box by the drillers. Instead, core is slid from the core barrel into a half-section of PVC pipe, covered with the other half of PVC pipe, and sealed for transport to the logging shed at ITH's camp (**Figure 11.3**). This procedure is effective and minimizes disturbance to the core, prevents unnecessary breakage, and minimizes crumbling of core prior to logging by a geologist.

13.0 Sample Preparation, Analyses and Security

13.1 Past Procedures

Soil and drill samples obtained in 2003 and 2004 exploration programs were subject to AGA's in-house methodology and Quality Assurance/Quality Control (QA/QC) protocols. Samples were analyzed by various methods by different laboratories.

The QA/QC program implemented by AGA met or exceeded industry standards. The program involved analysis of blanks, standards and duplicates. Blanks help assess the presence of any contamination that might be introduced by analytical equipment. Standards are used to assess the accuracy of the analyses, and duplicates help assess the reproducibility or precision of the analytical methods and equipment used.

All sampling campaigns were subject to insertion of blanks and standards at a rate of 1 blank and 1 standard for every 23 samples (total = 2QA/QC samples per 25 submitted samples). Blank samples consist of material known to contain below detection amounts of the metal for which the sample is being tested. Standards consist of sealed sachets of material with a certified abundance of the metal for which the sample is being tested. Standards were purchased from RockLabs and GeoStats.

Duplicate core and rock samples were run from pulp and coarse reject splits along with sample repeats approximately every 20 samples. Duplicate samples were also collected at the drill rig for 2003 RC drilling. Results of AGA's QA/QC program were reviewed by Dr. Klipfel in 2006 and in subsequent visits and reports. Overall, the QA/QC samples indicate that sampling and analytical work is accurate and reliable. In 2004, there were two instances of issues with blanks and standards out of compliance with AGA protocols, but these were satisfactorily resolved by AGA. The sample database did not appear to be compromised.

13.2 Current Procedures

ITH has continued with the QA/QC protocol of AGA as described above and increased the number of control samples (blanks and standards) to 1 in 10. Duplicate splits of drill samples are prepared for every 20 samples. ITH has undertaken rigorous protocols to assure accurate and precise results. Among other efforts, weights are tracked throughout the various steps performed in the laboratory to assure accurate assignment of results to the appropriate sample. ITH weighs all core samples before shipping. They are then reweighed by the laboratory when received and logged in. RC samples are dried and then weighed at the laboratory. Sample reject material is weighed again by the laboratory after the sample aliquot has been removed for pulverization. This tracking of sample weights enables constant verification of quality throughout the preparation process. Key results of this protocol include minimization of sample switches and transcription errors.

Samples are analyzed by standard 50g fire assay for the gold determinations. All core samples and select RC drilling samples are also submitted for multi-element ICP-MS analyses using a 4 acid digestion technique. All RC samples are analyzed on site for trace elements using a Thermo Fisher Scientific NITON portable XRF before shipment to the laboratory (**Figure 11.3**).

ITH geologic staff have developed a set of decision criteria that compare the abundance of Cr, Ni, Th, Zr, Mo, and V for determination of ultramafic, volcanic, Cretaceous intrusive (dikes), Upper Sediment, and Lower Sediment rocks. These results are cross checked with visual logging and ICP data before a final lithologic determination is entered in the database. The advantage of this type of procedure is that rock types can be more readily and more consistently identified in spite of significant alteration and replacement of original rock textures and minerals. Also, because arsenic correlates strongly with gold, an XRF determination of arsenic abundance has helped ITH anticipate gold-bearing zones before assays are returned. This information has proved constructive for drill planning and execution.

13.3 Data Handling

A project database is maintained by ITH with all drill hole location, survey, logging, sample, and assay information contained therein. As drill holes are completed, data is entered either manually, or through data downloads directly from instruments to the database. Assay information is received electronically from the laboratory and downloaded into the database. Subroutines check for errors and data format consistency.

The creation of sample data for RC drilling begins with pre-numbered sample bags that have drill hole number, sample interval, and sample number printed and bar-coded on a label attached to the bag (**Figure 11.3**). These bags are used at the drill rig for collection of RC chips into a primary sample, a secondary duplicate sample, and a chip sample for logging purposes (**Figure 11.3**). Drill core is sawed in half with a diamond saw with half the core going in a sample bag together with a tear off sample ticket preprinted with the sample number, and the other half retained in core boxes and stored on site.

NITON data collected by the instrument is keyed to the sample number so that data transferred from the NITON "gun" to the database remains matched with the sample number. Chip loggers similarly enter information into the logging database while reviewing chips under a binocular microscope with all intervals keyed to the sample interval and sample number (**Figure 11.3**). These are checked regularly by loggers and rechecked by the senior geologist. Database check and validation tools are also used to detect errors. Core logs are created manually and then the information is entered into a digital format for the database.

Dr. Klipfel has reviewed these procedures and watched the data entry process at various steps at different times on each of the visits. He is satisfied that ITH is diligent in their data management procedures and have check procedures in place that should identify any issues. He has not completed a thorough check or validation of the database but is not aware of any issues.

13.4 Quality Assurance and Quality Control

The QA/QC data from ITH sampling program has been reviewed by Dr. Klipfel. Analyses of blanks and standards that fall outside of an acceptable range, such as 3x detection limits for blanks or 10% for standards, are flagged for investigation. Unless a suitable explanation, such as a sample switch, can be found, the error is reported to the laboratory and the sample intervals around the questionable sample are rerun. A new certificate is issued by the lab for the reanalysis if the correct values for the standards and blanks are determined. Errors are generally attributable to sample switches, weighing errors and contamination of the first sample in a batch. Multi-element QA/QC is monitored using the compositions of the blank and standard materials.

Duplicate samples are used to assess reproducibility of the laboratory procedures and to ensure that the sampling procedure is representative. Field duplicates (28 in 2009) represent equivalent samples collected at the drill rig during the original sampling process and confirm that the sampling process is representative (Fig. 13.1a). Prep duplicates (1456 in 2009) are prepared by splitting the whole sample in half at the laboratory and subjecting each half to the full sample preparation routine and subsequent analysis. These duplicates are designed to confirm that no bias is created during the sample preparation process (Fig. 13.1b and c). Pulp duplicates (352 in 2009), representing multiple assays of the same pulverized material show that the laboratory procedures are precise and that the pulp material is uniform with errors of mostly less than 10% (**Figure 13.1**). Errors greater than 10% are believed to be due to normal nugget effect typical of gold deposits.

As the number of samples increases with each drilling campaign, it appears that there are local variations in the scale of nugget effect. The result is that some duplicates at higher values of gold (e.g. >3 g/t Au) show higher variance in reproducibility. Dr. Klipfel has evaluated this issue carefully and believes it is the result of normal nugget effect where a grain of relatively coarse gold ends up in one

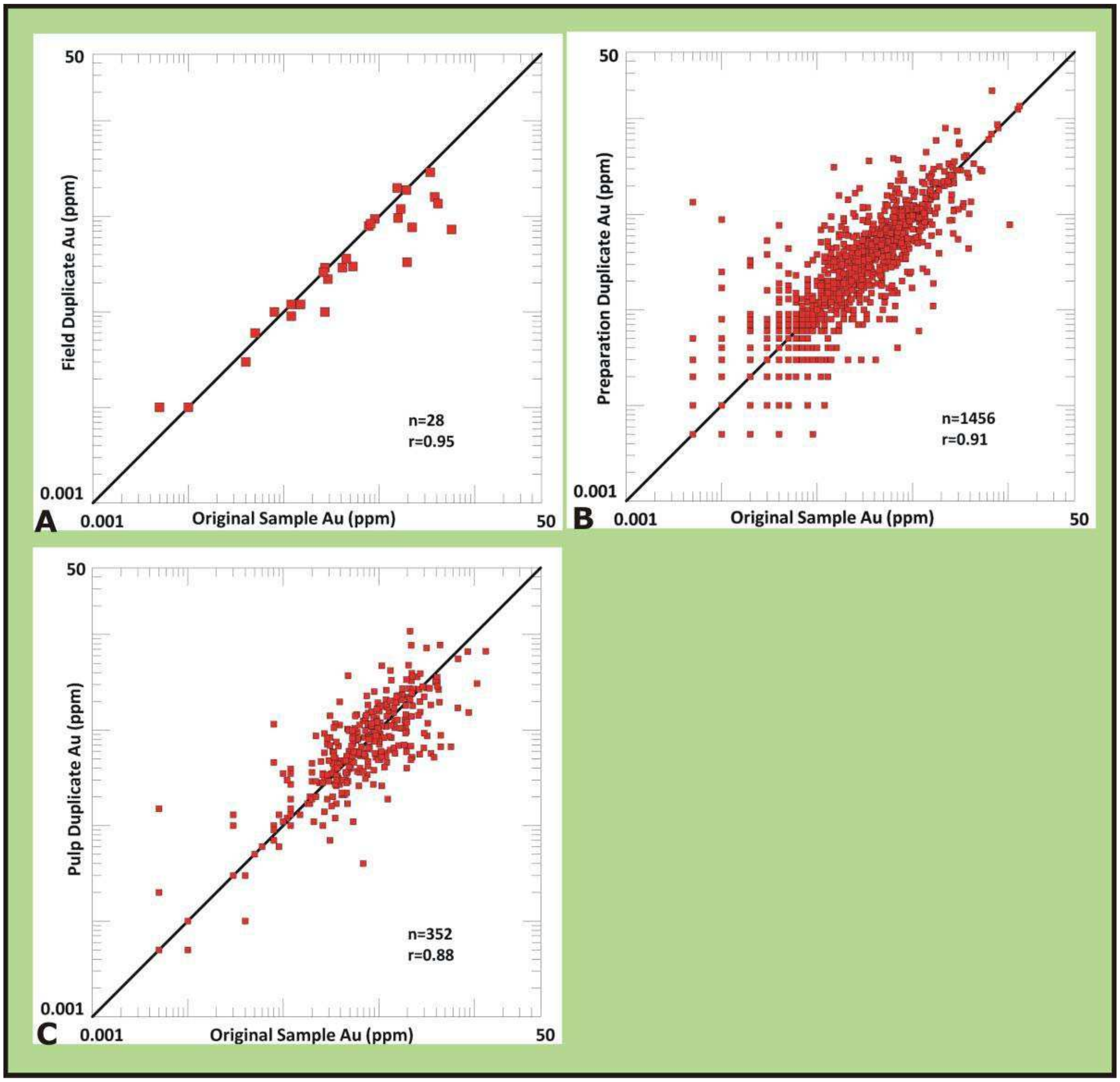


Figure 13.1. These scattergram plots show how different categories of sample duplicates compare with original sample results. The diagonal line has a slope of 1. Perfect duplication of results would plot on this line. Variation and scatter is interpreted to be the product of normal nugget effect. **A)** 2009 field duplicate vs. original samples; n= 27. The envelope of points flares with increasing grade. This is typical of nugget effect which becomes more pronounced at higher grades. **B)** 2009 prep duplicates compared to original sample values. The scatter indicates no particular bias with a good overall correlation between the two sets. The scatter is believed to reflect normal nugget effect in thes samples.. **C)** 2009 pulp duplicates vs. original sample. Scatter is similar to that in B.

split and not the other, thus producing a high value in one run and a lower value in another. This can be tested by comparing the blanks and standards for that range of samples and verify that these values are accurate and precise (**Figure 13.2**). Also, reproducibility tends to improve as gold values decrease except as the detection limit is approached (e.g. 0.005 vs 0.01 g/t = 100% error, but is at the detection

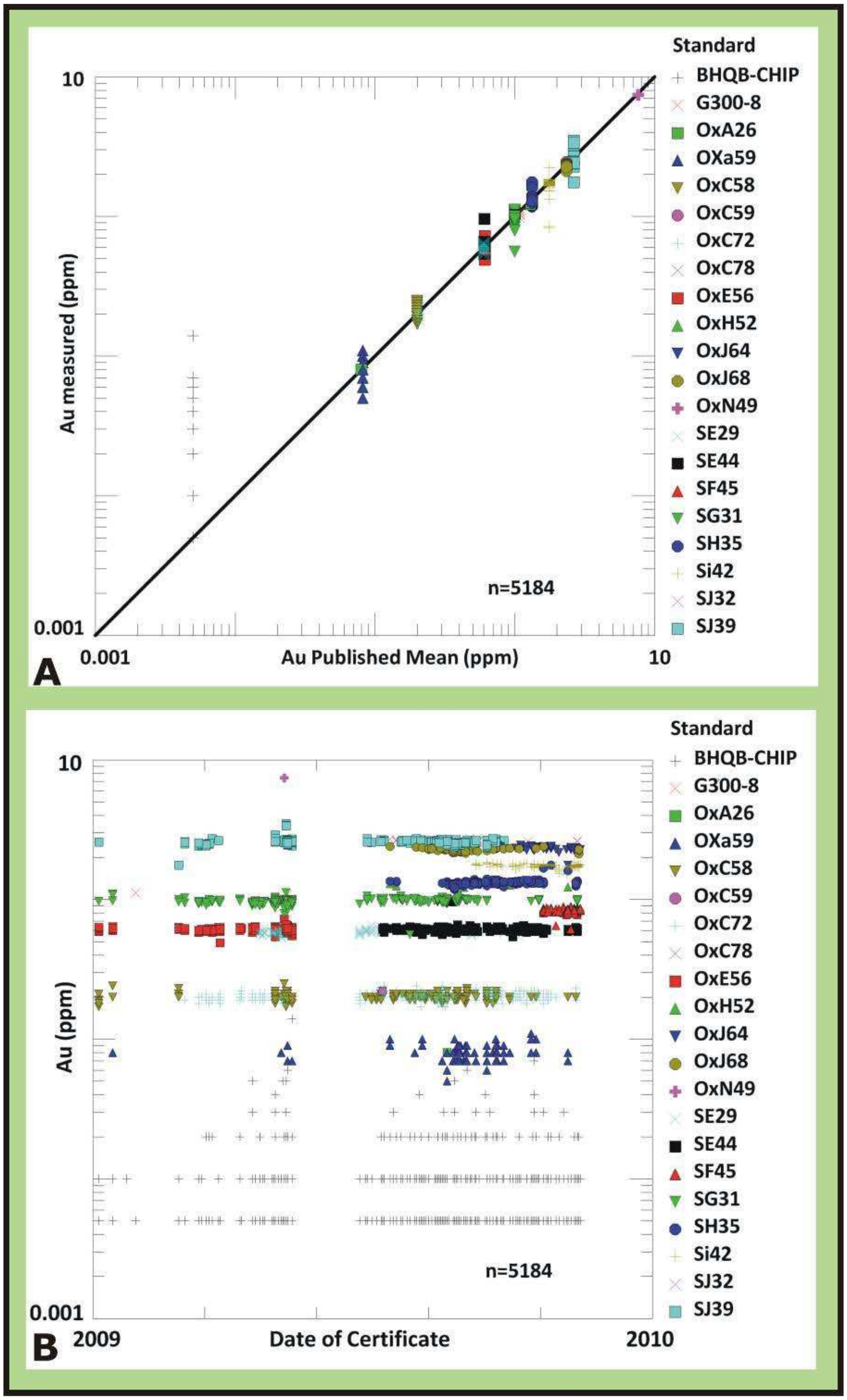


Figure 13.2. X-Y scattergrams for 2009 showing the stated value of standards vs. the measured value by the lab. **A)** values are plotted according to measured value vs. stated values of a standard placed in the sample stream. **B)** values plotted as a function of time to check for drift in results over time. The horizontal nature of the points for each value indicates that drift is minimal to non-existent.

limit and normal error envelope). This is most likely due to more even distribution of smaller gold grains so that an equal number of fine grains end up in each sample split. This level of variation due to nugget effect is deemed unlikely to impact the data set or the resource evaluation, because for each instance of a value in one sample being higher than in its paired duplicate, there should be an equal number of lower values recorded which missed the higher value split.

Prep duplicates (1456 in 2009, 736 in 2008, 187 in 2007), created by splitting either core samples after coarse crushing or splitting raw RC chips, show a somewhat higher degree of variability but demonstrate no bias to either high or low grade (r=0.91, Mean original samples = 0.43g/t, Mean of duplicates=0.45g/t). The reproducibility of most pulp duplicates also indicates that most of the gold is not so coarse that it causes major nugget effects. The variability in the coarse duplicates indicates that gold grains are not uniformly distributed within the sample material. This is consistent with the interpretation that gold is, at least partially, hosted in narrow veins and veinlets, which when crushed produce a small number of gold-bearing fragments in the overall sample, thereby causing nugget effect during the coarse sample splitting. In recognition of this effect sample preparation procedures were modified so that 1kg of sample material is now pulverized rather than 350g aliquot previously used. Dr. Klipfel considers these results to be appropriate for Livengood mineralization and indicative of sound QA/QC procedures.

14.0 Data Verification

Field and drill core observations made by Dr. Klipfel during site visits are consistent with the style of mineralization and alteration interpreted and reported in ITH documents. Outcrop exposures in drainages, trench faces, road cuts, and along the ridge lines were examined and found to be consistent with existing geological maps.

Drill logs, sections and maps were reviewed and are to a high quality. Provided information is consistent with observations of core and surface exposures.

In 2006, Dr. Klipfel collected a single sample along 3 m of a trench face where intrusive material with quartz veins is exposed. This sample was crushed, split, pulverized and assayed with a 50 g fire-assay AA finish method by ALS Chemex in Reno, Nevada. The sample contains 1.31 g/t Au, a value consistent with results from AGA sampling and expectations for material of that type and location.

In 2007, Dr. Klipfel collected seven samples from portions of two different drill holes, MK-07-18 and MK-07-20, from the remaining half of drill core previously sampled by ITH. Samples were selected for a range of gold content and rock type. The range of gold content in these samples is from below detection to 16.8 g/t Au. The core was quartered for the same sample interval as previously collected by ITH. Core material was bagged, labelled and information recorded by Dr. Klipfel and by ITH staff. Sample bags were sealed and transported to the ALS-Chemex laboratory in Fairbanks for sample preparation. Pulverized material was split into 300 gram master pulps and 120 gram analytical pulps before being sent to ALS Chemex in Vancouver for analysis. All samples except one returned results reasonably consistent with results from the ITH original sampling. The single sample that is different contains 0.61 g/t Au compared to 6.92 g/t Au in the original ITH analysis. This discrepancy is similar to the few discrepancies that occur in ITH's QA/QC sample duplication procedures. For this reason, the discrepancy is interpreted to reflect normal variation attributable to nugget effect as described in section 13.2. To the extent that this type of error is throughout the database, it is equally likely that a corresponding number of samples report low when the other half of core might report higher.

In 2008, 31 samples (26 RC and 5 core) were collected by Dr. Klipfel for verification analyses. These samples came from 5 different RC holes and 1 core hole. Samples were selected at random and specifically for a range of gold content from near detection limits (0.005 g/t Au) to high grade (20.9 g/t Au). Half-core that remains after a first sample was quartered and analyzed. Two standard and two duplicate samples demonstrated good reproducibility. RC samples demonstrated reasonable reproducibility, and core samples showed a range. No systematic bias was observed. Dr. Klipfel interprets these results to show normal scatter and nugget effect typical of mineralization at Livengood and for gold in general.

As a check of the data generated during the 2009 winter program, and the source of updated information in this report, Dr. Klipfel selected 28 samples from the duplicates collected by ITH from the winter program. These samples were selected to be representative of a range of rock type and gold values from different holes. Results show very good accuracy and precision for the standard and blank samples included with the sample set. The duplicate sample shows variation (2.13 vs. 2.89) of about 25%. Five other samples within this batch show significant variation between the original and duplicate analysis. For this reason, both the original and duplicate samples were re-analyzed. The values from these four runs show consistent variation among samples with higher gold values (e.g. 1 or more runs with higher values) for at least one run out of the four runs (**Figure 14.1**). It also shows minimal variation among samples with very low gold content. Importantly, samples with minimal or no gold (≤0.1 g/t Au) show consistency and repeatability. When plotted in log-log format, the envelope of variation becomes smooth, again suggesting a natural nugget effect. This assumes that the gold at Money Knob is consistent with the concept that natural systems follow logarithmic abundance patterns (Levinson, 1974; Rose and others, 1979).

In June, 2009, Dr. Klipfel selected 13 duplicate RC chip samples randomly at each of the three RC drill rigs and examined the chips for these samples. A comparison of results with the original sample is shown in **Figure 14.2**. Sample results correlate well and do not display any discernible bias. Deviation from an ideal 1:1 correlation is consistent with past sampling and the degree of nugget effect observed throughout the course of ITH's drilling program.

In addition to five rounds of sample verification, Dr. Klipfel witnessed the sluicing and panning of concentrated "clean up" material shovelled from a trench face in 2006. The material contained a significant amount of fine colors as seen in the panning dish verifying the presence of free gold at a range of sizes in that part of the trench face.

Data from duplicates for drilling have been reviewed by Dr. Klipfel and conform to previous QA/QC assessments.

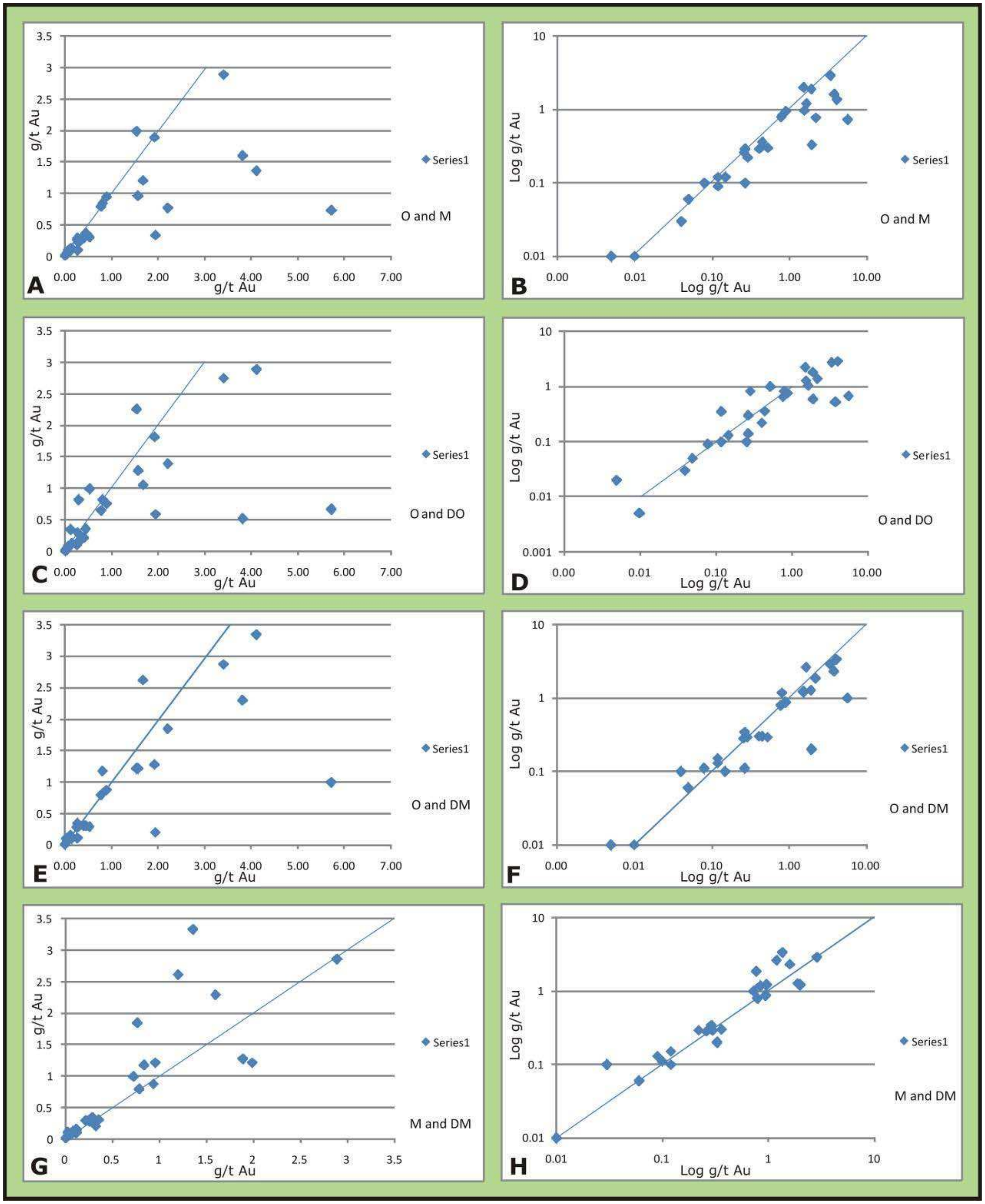


Figure 14.1 X-Y scatter plots of 2008 and early 2009 original and duplicate sample data for check samples collected by Dr. Klipfel as part of data validation procedures. The diagrams on the left are plotted with numeric scales. The diagrams on the right are plotted with log-log scales. The scatter increases with grade on diagrams with numeric scales while the envelope of points remains approximately parallel to the "unity" line. This is consistent with data following lognormal abundance

pattern typical of natural elemental abundance patterns. **A and B)** original vs. "met" splits. **C and D)** original vs. duplicate original splits. **E and F)** original vs. duplicate "met" sample. **G and H)** met and duplicate met samples. These diagrams collectively indicate a lack of consistent bias and show that different splits show variation consistent with nugget effect at all grades, but more pronounced at higher grades.

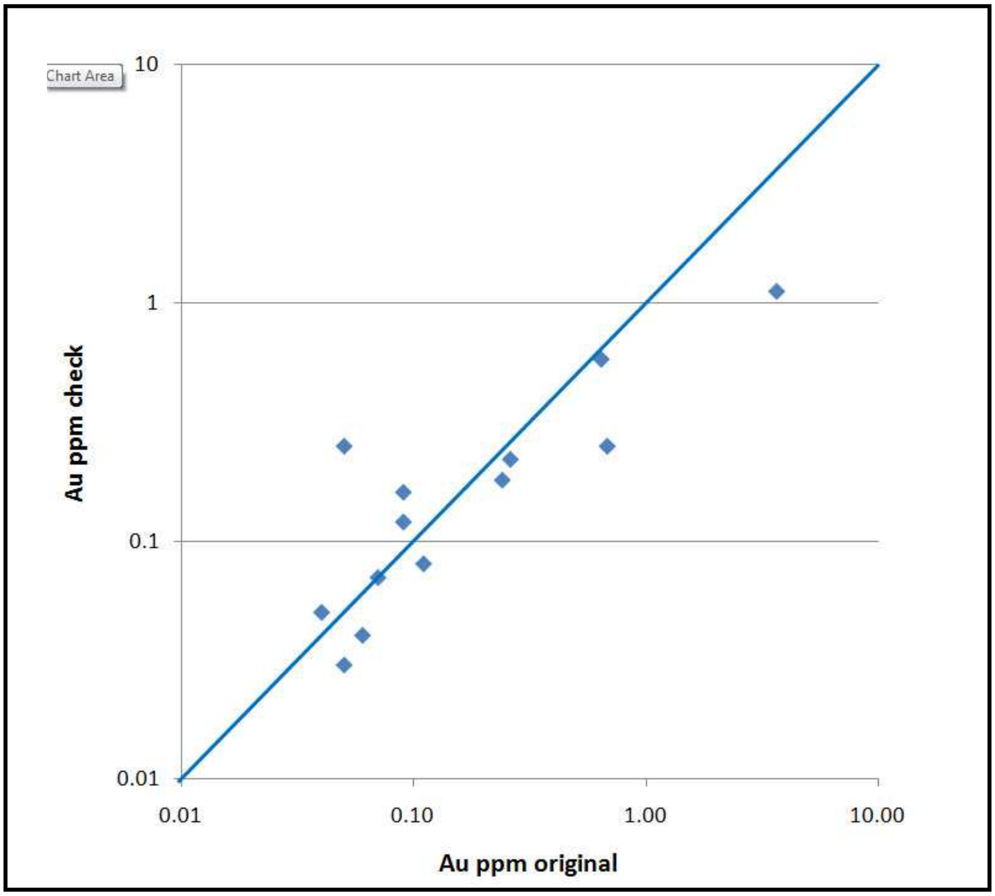


Figure 14.2 X-Y scatter plot of original and check samples for June 2009 RC drilling. The correlation line shows a slope of 1. Samples with identical results will plot on the line. Deviation of results from the line is interpreted to be the result of normal variation and nugget effect.

Dr. Klipfel has not verified all sample types or material reported. To the best of Dr. Klipfel's knowledge, ITH has been diligent in their sampling procedures and efforts to maintain accurate and reliable results.

15.0 Adjacent Properties

Another claim block called the Shorty Creek claims is controlled by Select Resources and is located approximately 10 km to the SW of the Livengood project area. This area is actively being explored for gold mineralization by Select Resources.

The Alaska Pipeline, the main means of transporting crude oil from Alaska's North Slope to the south coast of Alaska, runs northwest-southeast about 6 km to the west. This feature is not expected to have any impact on the project.

16.0 Mineral Processing and Metallurgical Testing

16.1 Introduction

ITH has undertaken metallurgical and processing test work to determine optimal recoveries using heap leach, mill with Carbon in Leach (CIL) and gravity or flotation techniques. Current test work focuses on determining the best means of optimizing these combined recovery methods. This work involves studies that will evaluate how the ore is characterized and how the ores vary in their physical and metallurgical response to process treatment parameters by ore type according to the various lithologic units that host ore. The characteristics under review include grindability, abrasiveness, optimal particle size for downstream treatment, and response to leach or gravity process parameters as a function of oxidation and lithology. In addition, the right combination of these techniques for different ore types is being evaluated. At this stage, however, only information on the heap leach processing is sufficiently advanced to allow inclusion as a recovery technique at this stage. ITH anticipates that other techniques will play a role in recovery of gold, but is unable to adequately assess or include these techniques in the plans at this stage of understanding.

Tests so far reveal that the metallurgy for the Livengood deposits is extremely variable depending on the ore type. The host rocks are folded and have been subjected to numerous geologic events during formation. The result is a deposit that has ore types that are highly amenable to simple cyanide leaching process techniques like heap leaching and carbon in pulp processes, while having other ore types that are more difficult and will require carbon in leach processing, gravity or flotation concentration steps, and finer grinding.

The information presented here derives from studies focused on utilizing the simpler, less expensive heap leaching process scenario and only includes those ore types that are amenable to this process. A significant portion of the ores at Livengood should respond positively to this process scenario. Ores that will not be economical to treat in a heap leach scenario have been excluded and handled as waste in this report.

Overall recoveries for the project base case heap leach study average approximately 60%. These recoveries were determined using 10 mesh bottle roll data as well as numerous finer grind data to develop projected scale-up extraction curves and reagent consumptions for each ore type. These ore types are modeled within the deposit using all of the key determinate geologic and analytical data to produce individual block gold recovery values for use in the mining analysis.

Metallurgical test work is currently underway on processes that may be economically viable for the more difficult to process Livengood ore types. Mineral processing and metallurgical work to date indicates that a combined heap leach – gravity or flotation concentrate – CIL recovery system will result in optimal recovery of gold.

16.2 Metallurgy Summary

Metallurgical test work programs on the Livengood ores began in 2004 and continue as of the writing of this report. The ore types at Livengood are variable in their chemistry, in their physical properties, and in their metallurgical characteristics. The following statements best describe the observed results of the test work performed to date:

- Most Livengood ores could be considered moderately soft to medium hard in hardness with an average Bond Ball Work index of 15.8. The ores varied significantly in hardness, with Bond Ball Work indices varying from a minimum of 11.1 to a maximum of 19.1.
- The majority of the ores would be considered non-abrasive, with an average Abrasion index of 0.0809. The ore types abrasion characteristics varied significantly from 0.0023 to 0.2872.
- All of the Livengood ore types respond to cyanide leaching to some degree.
- The effect of leach times on gold recovery and gravity concentration results, indicates some of the ores contain coarse gold.
- Gold recovery at 10 mesh particle sizes on some of the ore types exceeded 90 percent
- Gold recovery on some of the ore types, but not all, is improved with finer grinding. A grind size where 80 percent of the particles are smaller than (p80) 200 mesh (74 microns) has been tested to date.
- The degree of oxidation of the ore, as observed by the geologist, has a marginal impact on the gold recovery.
- Differences in gold recovery between cyanide shake leach tests, bottle roll leach tests, and Carbon-in-Leach tests suggest organic carbon in the ores is active to varying degrees in some of the ore types, particularly the un-oxidized version of those ore types.
- The gold in the ore is often associated with sulfides, but the ore would not be classified as a sulfide refractory ore.

These results indicate that the majority some of the ores are amenable to conventional heap leaching and gravity separation recovery processes, while others present more challenging metallurgical issues.

It became evident early on in the test work that the oxidized to partially oxidized ores responded well to cyanide leaching while other un-oxidized ore types performed moderately to poorly depending on the method used to perform the analysis, i.e cyanide shake leach tests versus bottle roll tests. However, it was found that all of the ores do respond to cyanidation to some degree.

The most significant metallurgical parameter observed to date with the Livengood un-oxidized ores is the presence of organic carbon and the indication that some, but not all, of the organic carbon is "active" or "preg robbing" in nature. Metallurgical test work began to focus on process methods that could be used to counter the preg robbing effects of the ores. The simplest of these methods, the Carbon-in-Leach (CIL) process, is currently being used in test work performed in the latest round of tests. The CIL test work is showing very positive results in counteracting the effects of preg robbers, providing an average increase of gold recovery for all ore types of approximately 18 percent and as high as a 49.5 percent increase in gold recovery for the more difficult un-oxidized ores.

In addition, initial gravity concentration testing of the Livengood ores has produced encouraging results with 59% of the gold reporting to the gravity concentrates. The results show a 69-1 concentration ratio (gravity concentrate weight percent of 0.0145%) provides an average concentrate grade of 42.9 g/t Au. Test work is ongoing on the gravity concentrate to establish the viability of ultra-fine grinding and high intensity cyanide leaching of the concentrate.

From the data currently at hand the oxide and partially oxidized ores will respond well to heap leaching. Ongoing test work indicates higher gold recoveries can be obtained from all ore types and particularly the weakly to un-oxidized ores with the use of standard milling that utilizes an initial gravity circuit followed by a Carbon-In-Leach (CIL) process for the gravity circuit tails. The ability to increase recoveries from the higher grade mineralized zones as well as effectively process the weak to un-oxidized ores has the potential to significantly improve the Livengood project in both its size and economic performance.

Metallurgical test work currently planned and scheduled for the future will continue to focus on utilizing CIL as a primary metallurgical process. Enhancing the CIL test work with tests that attempt to render preg robbing organic carbon inactive will also be performed.

Flotation test work will be performed to get an understanding of the ability to concentrate the gold and depress gold preg robbers prior to downstream cyanidation.

Column leach test work will be performed on ores that do not show preg robbing tendencies to establish the effectiveness of heap leaching as a process option.

Other future test work will include enhancing the ability of utilizing the gravity susceptible component of the ores for improving overall gold recovery.

This test work will be performed in conjunction with enhancing the ability to identify the ores that are subject to having preg robbing issues. Understanding the geology of the ore types with respect to preg robbing organic carbon will be an important task moving forward in the Livengood project.

16.3 Gold Characterization

Hazen Research, Inc. performed gold characterization work on products they prepared from a heavy liquid separation test program performed on Livengood samples during late 2006 and early 2007 (Hazen Research Inc. 2007),

The samples were ground to minus 35 mesh for gravity separation. The minus 35-mesh material was first wet-screened at 500 mesh (25 µm). The minus 35- plus 500-mesh product was split in half, and each half was separated with heavy liquid at a density of 2.96 to upgrade the heavy minerals plus the gold to enhance detection of the gold. The float (tailings), sink (concentrate), and the unseparated minus 500-mesh slimes from one set of heavy-liquid separation were fire assayed for gold and silver. The products from the other set were used for the mineralogical examination. To concentrate the gold even further, the sink product and the minus 500-mesh slimes were panned.

The test showed 4% to 10% of the sample mass reported to the heavy mineral concentrate, which contained between 44% and 77% of the gold. Another 13% to 33% of the gold reported to the minus 500 mesh slime fraction with the balance reporting to the +35 mesh float fraction. Silver values in the ore were essentially negligible and the silver did not report to the heavy mineral concentrate with the gold. Microprobe analysis of one gold grain indicated that the silver content was 7.4%. The balance of the silver was probably held in other sulphide phases.

The main sulphide minerals in the heavy mineral concentrates were pyrite and arsenopyrite in ratios ranging from 2:1 to 6:1. Pyrrhotite and chalcopyrite were commonly observed as inclusions in both pyrite and arsenopyrite. Pyrite may be euhedral or anhedral and was frequently porous, enclosing abundant inclusions of gangue and rutile. Sphalerite tended to occur as liberated grains or intergrowths with pyrite and arsenopyrite rather than as inclusions. Trace amounts of several other sulphide minerals and gold were also present. Hematite was observed in the only partially oxidized sample examined. Marcasite was reported in some samples also, and in one of these it occurred as distinct clusters of acicular crystals and was possibly a product of oxidation.

Gold occurrences were scarce. The size of the gold varied between less than 5 and 23 μm. The particles observed were mostly associated with arsenopyrite as small attachments or inclusions, and one liberated particle was found in the minus 500-mesh product of the partially oxidized volcanic-hosted sample.

16.4 Historical Test Work Programs

In 2004, AGA attempted to test the cyanide solubility of gold in drill sample material by analyzing samples containing more than 200 ppb Au. Samples were sent to ALS Chemex for a 30g cold cyanide leach assay (Au-AA24). A total of 198 samples were analyzed in this manner and they showed consistent CN soluble assays, on average about 60% of the fire assay value (AGA in house memorandum to files). The significance of this result was unclear at the time because there were many variables which could affect this outcome. These included small sample size, nugget effect, host rock type, sulphide content, other mineral content, encapsulation, and possible inappropriate testing method. Of these, nugget effect is expected when there is coarse free gold which was witnessed by Dr. Klipfel in the sluice sample of trench face material and has been seen in drill core. Sulphide and organic carbon are present and also could be significant factors. In an effort to determine which minerals might impact the cyanide test, AGA used principle component analysis for four sets of ‘factors’. They concluded that As and Sb had little impact, but that sulphide content and coarse gold were the leading contenders for lowering recovery in the CN leach samples.

The AGA test work was deemed inconclusive due to small sample size and nugget effect. However, it should be an indicator of processing and recovery possibilities and issues. It also showed that gold and sulphide characterization studies are needed for metallurgical and process planning. Any such study should address sample size, coarse free gold content, distribution and location of gold in host rock, material type (shale, volcanic, intrusive), sulphide species, and organic carbon content. At this stage, the results were only considered as a preliminary indicator of potential issues for a cyanide leach process.

In 2006, ITH submitted a single sample of unoxidized vein-related mineralization to Hazen Research for a gold characterization study. The sample showed that the bulk of the gold occurs as micron-scale native gold grains in and adjacent to pyrite and arsenopyrite grains with a smaller number of grains associated with silicate gangue. Cyanide recovery in a bottle roll test was 61% (**Table 16.1,** Sample 1A).

In 2007 six more samples were submitted to Hazen Research for additional gold characterization studies. These samples represented both high and low grade mineralization from oxidized, partially oxidized and unoxidized material. Cyanidation of the samples shows that the cyanide extraction of gold is very high on the oxide and partially oxidized samples (**Table 16.1**) and somewhat less in the sulphide material. Two of the sulphide samples (**Table 16.1**, samples 3 and 1A) were from rock with albitic alteration and they each returned 60% cyanide recovery. The 3rd sulphide sample (**Table 16.1**, sample 5) came from rock with sericite alteration and had only a 42% recovery.

A very important result of this work is the observation that, for all the samples tested in 2007, the bulk of the gold recovered by cyanide extraction is released in the first 16 hours. This implies that the gold is readily available to the cyanide solution. Further studies will address the cyanide extraction on both fine and coarse material as a first step in the determination of the optimal recovery process.

TABLE 16.1
GOLD RECOVERY FROM 2007 CYANIDE EXTRACTION TESTS

Sample #	Ore Type	Average Grade (g/t)	% Cyanide Extraction*
1	Oxide Sediments	1.52	99.9%
2	Oxide Sediments High-grade	10.80	96.9%
3	Un-Oxidized Volcanic	1.52	59.7%
4	Oxide Sediments	1.39	99.9%
5	Un-Oxidized Volcanic	1.38	42.3%
6	Weakly Oxidized Volcanic	1.06	90.2%
1A	Volcanic Un-Oxidized	2.30	60.9%

** Samples were 300 gram bottle rolls with sample material crushed to ~200 mesh and sampled every 8-10 hours for a total of 48 hours.*

In 2008 an additional 24 samples were submitted to Hazen research for bottle roll testing on coarse material from a variety of lithologies and oxidation states (**Table 16.2**). This was undertaken as a separate study from a previous one with Chemex. Results indicate that overall average cyanide extraction was approximately 70% with 15 of the 24 samples showing greater than 70% recovery. Interestingly many of the unoxidized samples showed better recovery than some of the partially oxidized samples. These data also show that the majority of the gold is released to solution within the first 16 hours. The same sample materials have been submitted to Kappes Cassiday in Reno for fine grinding and tests of gravity recovery and cyanide extraction at a -200 mesh grind. The results are presented in **Table 16.3**.

TABLE 16.2
GOLD RECOVERY FROM 2008 HAZEN CYANIDE EXTRACTION TESTS (-10 MESH)

Sample ID	Ore Type	Hazen Head Au g/t	Chemex Head Au g/t	Calculated Head Au g/t	Residue Assay Au g/t	Hazen Gold Extraction	Chemex Gold Extraction	Calculated Head Extraction
100112113	Partial Oxide Um	0.48	1.26	0.81	0.17	64%	87%	79%
100123124	Trace Oxide Um	0.83	0.83	0.81	0.33	60%	60%	59%
100588589	Partial Oxide Um	0.88	1.03	1.13	0.47	47%	54%	58%
100772773	Partial Oxide Intr	0.77	0.74	0.96	0.23	70%	69%	76%
100829830	Unoxidized Lower Seds	1.18	1.04	1.33	0.31	74%	70%	77%
101024026	Unox Volc	1.30	0.85	1.04	0.31	76%	64%	70%
101273274	Unox Volc	1.00	0.92	1.11	0.25	75%	73%	78%
101291292	Partial Oxide Volc	1.24	0.71	1.51	0.21	83%	70%	86%
101437438	Partial Oxide Volc	0.60	1.44	1.12	0.46	23%	68%	59%
101548549	Partial Oxide Volc	2.47	1.17	3.22	0.16	94%	86%	95%
101604605	Partial Oxide Volc	1.70	0.80	1.36	0.35	79%	56%	74%
101618619	Partial Oxide Volc	1.15	0.96	1.14	0.47	59%	51%	59%
101774775	Partial Oxide Volc	1.13	0.82	1.06	0.16	86%	80%	85%
101827829	Partial Oxide Volc	0.72	0.84	0.59	0.12	83%	86%	80%
101847849	Partial Oxide Volc	0.80	0.81	1.05	0.44	45%	46%	58%
101896897	Partial Oxide Volc	3.36	1.16	1.17	0.89	74%	23%	24%
102070071	Trace Oxide Volc	0.44	0.49	0.74	0.06	86%	88%	92%
102096097	Trace Oxide Volc	1.35	1.03	0.94	0.28	79%	73%	70%
102536537	Comp Ox Upper Seds	1.67	1.09	0.69	0.07	96%	94%	90%
102575576	Part Oxide Upper Seds	0.77	1.96	1.16	0.05	94%	97%	96%
102642643	Part Oxide Upper Seds	0.58	0.71	0.81	0.25	57%	65%	69%
102886887	Part Oxide Upper Seds	0.96	0.95	1.05	0.69	28%	27%	34%

Sample ID	Ore Type	Hazen Head Au g/t	Chemex Head Au g/t	Calculated Head Au g/t	Residue Assay Au g/t	Hazen Gold Extraction	Chemex Gold Extraction	Calculated Head Extraction
102925926	Part Oxide Upper Seds	1.46	1.16	1.49	0.77	47%	34%	48%
103110111	Part Oxide Upper Seds	0.63	0.91	0.87	0.22	65%	76%	75%

**Samples were 1400 gram bottle rolls with sample material crushed to -10 mesh and sampled in multiples of 4 hours for a total of 72 hours.*

TABLE 16.3
GOLD RECOVERY RESULTS FROM KAPPES CASSIDAY CYANIDE EXTRACTION TESTS (-200 MESH)

Sample ID	Calculated Head, Au g/t	Extracted, Au g/t	Avg. Tails, Au g/t	Au Extracted, %	Leach Time, days	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
100112113	0.459	0.39	0.073	84.1%	3	1.10	2.75
100123124	0.609	0.47	0.144	76.4%	3	0.45	1.00
100588589	1.686	1.23	0.461	72.7%	3	0.53	2.00
100772773	0.728	0.51	0.221	69.6%	3	2.01	2.75
100829830	1.278	1.06	0.221	82.7%	3	0.55	2.50
101024026	0.620	0.54	0.077	87.6%	3	0.66	2.25
101273274	0.787	0.68	0.105	86.7%	3	0.51	1.50
101291292	1.333	1.21	0.125	90.6%	3	0.81	1.00
101437438	0.819	0.57	0.247	69.8%	3	0.48	1.50
101548549	2.670	2.51	0.162	93.9%	3	0.22	1.50
101604605	0.992	0.83	0.166	83.2%	3	0.37	1.50
101618619	1.434	1.15	0.280	80.5%	3	0.82	2.50
101774775	1.069	1.00	0.068	93.7%	3	0.56	1.50
101827829	2.733	2.67	0.063	97.7%	3	0.66	1.50
101847849	1.279	0.75	0.525	59.0%	3	0.48	1.50
101896897	1.269	0.52	0.747	41.1%	3	0.79	1.50
101925926	1.552	1.00	0.555	64.2%	3	0.12	1.50
102070071	0.594	0.52	0.077	87.0%	3	0.72	2.00
102096097	1.074	0.96	0.117	89.1%	3	0.57	1.50
102536537	0.875	0.84	0.034	96.1%	3	0.69	2.00
102575576	0.927	0.87	0.053	94.3%	3	0.71	1.50
102642643	0.596	0.48	0.120	79.9%	3	2.49	4.00
102886887	0.873	0.36	0.510	41.6%	3	1.28	4.00
103110111	0.711	0.60	0.110	84.6%	3	0.94	2.50
Average	**1.124**	**0.90**	**0.219**	**79.4%**	**--**	**0.77**	**1.99**

Samples were 1000 gram bottle rolls with sample material crushed to **-200 mesh and sampled in multiples of 4 hours for a total of 72 hours.*

Comparing the results of the two test series indications were that finer grinding improved the overall gold recovery, in some cases as much as 18 percent. These results indicated that the gold was not refractory, but is tightly held in the ore matrix. The gold recovery averaged 79.4 percent on an average head grade of 1.12 g/t. Lime and cyanide consumption data were also gathered during this series of tests and are presented in **Table 16.3.**

Additional test work is currently underway on 35 composites made up of 1195 individual samples from the Livengood drilling campaign. The composites are of eight different stratigraphic units further delineated by the degree of oxidation and gold grade. The test work is being performed to further investigate ore chemical and physical characteristics, and the effectiveness of gravity and cyanidation for gold recovery.

Other test work planned for these composites includes flotation, gravity and flotation concentrate fine grinding and high intensity leaching, and aeration and lead nitrate addition.

16.5 Current Test Work Program

A test work program is currently underway at Kappes Cassidy and Associates (KCA) in Reno, Nevada, on Livengood ore samples. KCA has also contracted with ALS Chemex to perform ICP analyses of the composites, and Phillips Enterprises LLC to perform grinding and abrasion studies.

Thirty-five test composites comprised of RC met samples up to 200kg each were sorted and binned on site and shipped directly to the KCA laboratory in Reno, NV for testing. The samples represent eight different stratigraphic units with distinct silicate mineralogies. Samples from each stratigraphic unit were selected to represent variations in grade and degree of surficial oxidation.

When the samples arrived at the lab, they were identified by composite type, logged in, and weighed. The lab blended the samples to insure the composites were thoroughly mixed and homogenous prior to removing any sample splits. Samples were handled and stored in a manner which prevented the possibility of cross contamination with other clients' samples and other Livengood composites.

The primary focus of the test work campaign was to identify the chemistry of each of the composites, identify the potential for utilizing gravity separation and cyanidation as a metallurgical processes for gold extraction, and establishing preliminary grinding parameters for the various Livengood ore types. The lab conducted grind studies to develop laboratory stage ball mill grind times and developed Bond Ball Work indices. Gravity concentration test work has been performed in a stage grinding test that identified the total gravity recoverable gold (GRG). Cyanide shake leach tests and cyanidation bottle roll tests were performed in duplicate and at a target 80% passing 10 mesh, 100 mesh, and 200 mesh grind sizes.

The following diagram, **Figure 16.1**, presents the breakdown of sample requirements by composite for the proposed test work program. A list of proposed tests and a test work outline for Livengood Gold ore follows the diagram.

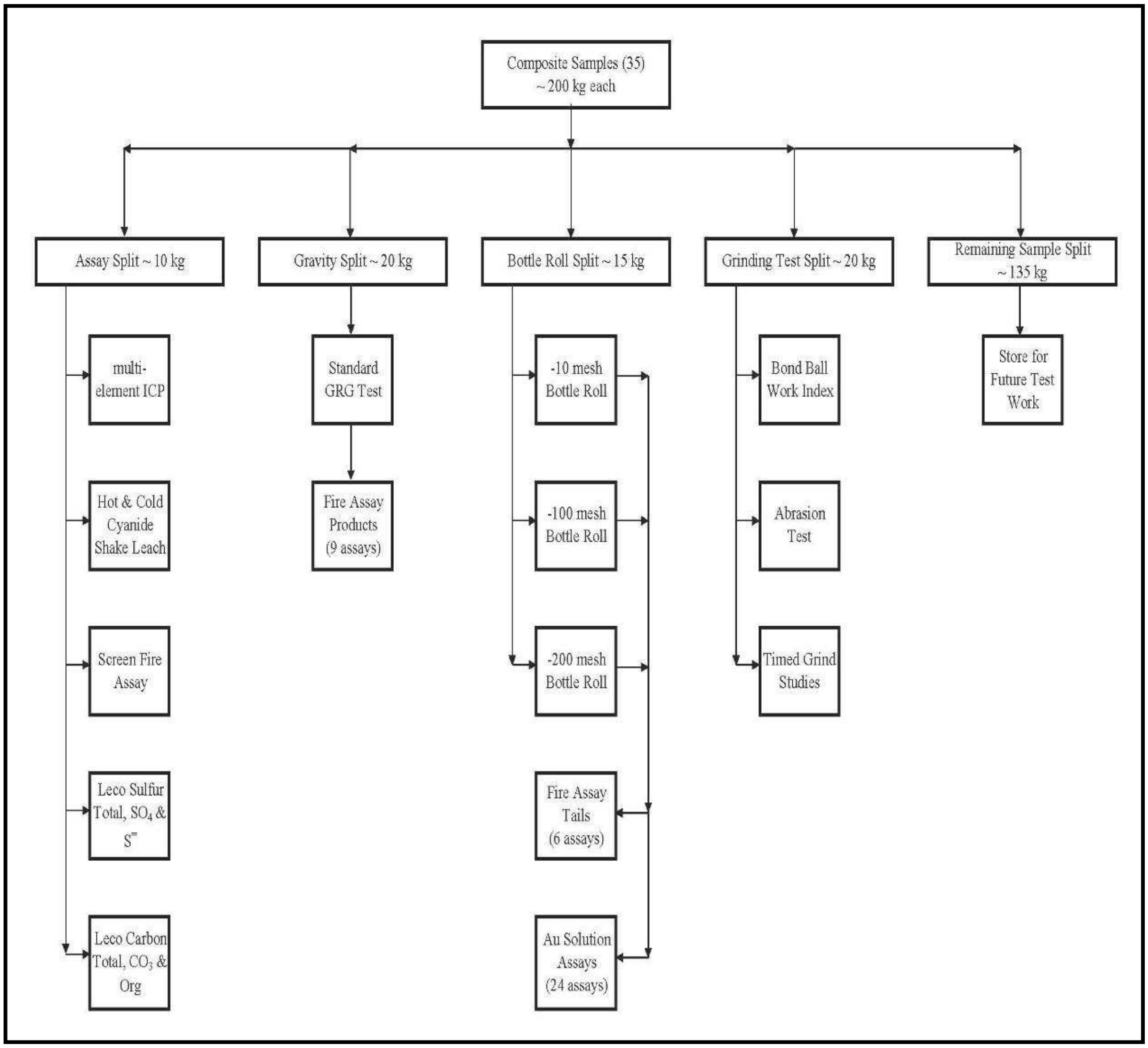


Figure 16.1 Flow chart and breakdown of Livengood composite sample testwork.

The Livengood Samples were initially separated by the following stratigraphic units

- Overburden
- Upper Sediments
- Main Volcanics
- Lower Sediments
- Lower Sands
- Kint
- Cambrian
- Amy Sequence

Each stratigraphic unit was then separated by degree of oxidation

- None
- Trace
- Partial and Complete

Each stratigraphic unit by degree of oxidation was composited by grade

- 0.5 ppm Au to 1.0 ppm
- >1.0 ppm to 5.0 ppm

Using this methodology the total number of composite samples comes to 48. However, some of the composites selected were volumetrically insignificant in the deposit and therefore the total number of composites submitted totaled 35.

The composites were blended in order to ensure each composite was homogeneous prior to removing any sample splits. Most of the composites weighed approximately 200 kg each, with 5 composites weighing about 40 to 50 kg.

Each composite has had a multi-element analysis performed by ALS Chemex (4-acid digest ICP-MS method ME-MS61m). Gold was determined by triplicate 1 kilogram screen fire assays and silver was determined by triplicate fire assays with an AA finish. Composites were also analyzed for sulfate, sulfide and total sulfur, as well as carbonate, organic carbon and total carbon.

All of the composites had a comparative cyanide leach assay using a hot cyanide leach and a cold cyanide leach. The tests were performed under the conditions shown in **Table 16.4**.

After leaching, the samples were centrifuged and the solution removed for Au assay by atomic absorption spectrometry. Assays were performed in triplicate. These results are shown in **Table 16.5.**

TABLE 16.4
CYANIDE SHAKE LEACH TEST PROCEDURE PARAMETERS

Procedure	Sample wt.	Soln. Temp.	Soln. NaCN Conc.	Soln. Amount	Leach Time
Hot Cyanide Leach	30 g	60°C	0.50 %	60 mL	1 hour
Cold Cyanide Leach	30 g	Ambient	0.50 %	60 mL	1 hour

TABLE 16.5
LIVENGOOD PROJECT
SUMMARY OF CYANIDE SHAKE TESTS (5 GPL NACN)

Description	Average Head Assay, Au g/t	Average Met Screen, Au gs/t	Overall Average Head, Au g/t	Average Cyanide Sol. (22 °C), Au g/t	Average Cyanide Sol. (60 °C), Au g/t
Overburden: Partial Ox (L)	0.82	0.59	0.71	0.27	0.30
Cambrian: Partial Ox (L)	0.28	1.21	0.75	0.19	0.21
Cambrian: Partial Ox (H)	2.17	1.78	1.97	0.21	0.21
Cambrian: Trace Ox (L)	0.69	0.66	0.67	0.15	0.15
Cambrian: Trace Ox (H)	1.79	1.79	1.79	0.23	0.36
Kint: Partial Ox (L)	0.80	0.72	0.76	0.21	0.25
Kint: Partial Ox (H)	0.68	2.18	1.43	0.41	0.47
Kint: Trace Ox (L)	0.73	0.76	0.75	0.02	0.02
Kint: Trace Ox (H)	0.68	1.43	1.06	0.01	0.03
Kint: No Ox (L)	0.66	0.89	0.77	0.02	0.01
Kint: No Ox (H)	0.93	0.95	0.94	0.01	0.05
Lower Seds: Trace Ox (L)	0.74	1.00	0.87	0.01	0.01
Lower Seds: Trace Ox (H)	1.81	0.85	1.33	0.01	0.02
Lower Seds: No Ox (L)	0.54	0.73	0.63	0.01	0.02
Lower Seds: No Ox (H)	0.78	1.10	0.94	0.01	0.02
Main Volcanics: Partial Ox (L)	0.53	0.77	0.65	0.16	0.25
Main Volcanics: Partial Ox (H)	1.79	1.75	1.77	0.19	0.39
Main Volcanics: Trace Ox (L)	0.73	0.74	0.73	0.03	0.10
Main Volcanics: Trace Ox (H)	1.12	1.55	1.33	0.05	0.07
Main Volcanics: No Ox (L)	0.96	1.02	0.99	0.05	0.08
Main Volcanics: No Ox (H)	3.01	1.88	2.45	0.03	0.06
Upper Seds: Partial Ox (L)	1.84	0.89	1.36	0.23	0.20
Upper Seds: Partial Ox (H)	1.30	1.40	1.35	0.38	0.41
Upper Seds: Trace Ox (L)	1.25	1.11	1.18	0.06	0.03
Upper Seds: Trace Ox (H)	0.94	1.53	1.24	0.09	0.08
Upper Seds: No Ox (L)	0.77	1.14	0.95	0.05	0.01
Upper Seds: No Ox (H)	2.77	0.99	1.88	0.06	0.03
Lower Sand: Partial Ox (L)	0.80	0.98	0.89	0.01	0.03
Lower Sand: Partial Ox (H)	1.52	2.01	1.76	0.04	0.05
Lower Sand: Trace Ox (L)	1.29	0.70	0.99	0.02	0.01
Lower Sand: Trace Ox (H)	0.82	1.33	1.08	0.03	0.01
Lower Sand: No Ox (L)	1.05	0.59	0.82	0.03	0.06
Lower Sand: No Ox (H)	0.75	1.25	1.00	0.05	0.02
Amy Sequence: Partial Ox (L)	1.34	0.29	0.81	0.09	0.09
Amy Sequence: No Ox (L)	0.49	0.44	0.46	0.03	0.06
			1.12	**0.10**	**0.12**

Descriptions from documentation provided by Talon Gold: (L) - 0.5 ≤ Au g/t ≤ 1.0; (H) - 1.0 ≤ Au g/t ≤ 5.0

As indicated by the test results (**Table 16.5;** right two columns), the response of the Livengood ores to the CN shake leach test procedure for determining gold leachability was poor averaging 9% and 11% of the average head grade for 22°C and 60°C solutions respectively . The poor results were later found to be linked to "active" organic carbon in some of the ores, slow leaching gold ores, and large gold particle sizes.

16.5.1 Grind Studies and Ball Mill Bond Work Indices Tests

Grind studies were performed on each of the composites in order to establish grind time versus grind size relationships. This information was used to prepare samples for future studies at varying grind sizes.

In addition to the above grinding tests, Bond Ball Work Index tests were performed. The results of these tests will be used to obtain preliminary grinding operating costs and to perform preliminary mill sizing calculations.

A total of 35 composites were tested to achieve an index for each of the ore types. **Table16.6** provides the results of the Bond Ball Work Index tests. Since the samples used for performing the tests were finer than typically received for bond testing, a conservative factor of 1.2 has been applied to the test work results.

Core from the Livengood property was obtained for performing abrasion tests. Results of this test are shown in **Table 16.7** and reveal low abrasion indices for most rock types. The albite altered volcanics and the Amy chert shows the highest abrasion indices with unoxidized siltstones and partially oxidized volcanic having moderate abrasion indices.

TABLE 16.6
LIVENGOOD PROJECT
BOND BALL MILL WORK INDEX TEST RESULTS

Description	BWI kW-hr/st	BWI kW-hr/MT	BWI x 1.2 kW-hr/st	BWI x 1.2 kW-hr/MT
Overburden: Partial Ox (L)	9.81	10.82	11.78	12.98
Cambrian: Partial Ox (L)	11.21	12.36	13.45	14.83
Cambrian: Partial Ox (H)	9.76	10.76	11.71	12.91
Cambrian: Trace Ox (L)	12.66	13.96	15.19	16.75
Cambrian: Trace Ox (H)	11.00	12.12	13.20	14.55
Kint: Partial Ox (L)	11.25	12.41	13.50	14.89
Kint: Partial Ox (H)	11.80	13.01	14.16	15.61
Kint: Trace Ox (L)	13.20	14.55	15.83	17.46
Kint: Trace Ox (H)	13.06	14.40	15.67	17.28
Kint: No Ox (L)	13.44	14.82	16.13	17.78
Kint: No Ox (H)	13.16	14.51	15.79	17.41
Lower Seds: Trace Ox (L)	13.33	14.70	16.00	17.64
Lower Seds: Trace Ox (H)	13.09	14.43	15.70	17.31

Lower Seds: No Ox (L)	13.26	14.62	15.92	17.55
Lower Seds: No Ox (H)	13.55	14.94	16.26	17.93
Main Volcanics: Partial Ox (L)	13.07	14.41	15.68	17.29
Main Volcanics: Partial Ox (H)	12.75	14.06	15.31	16.87
Main Volcanics: Trace Ox (L)	14.81	16.32	17.77	19.59
Main Volcanics: Trace Ox (H)	13.26	14.61	15.91	17.54
Main Volcanics: No Ox (L)	13.65	15.05	16.38	18.06
Main Volcanics: No Ox (H)	13.49	14.87	16.18	17.84
Upper Seds: Partial Ox (L)	13.53	14.91	16.23	17.89
Upper Seds: Partial Ox (H)	13.20	14.56	15.84	17.47
Upper Seds: Trace Ox (L)	13.21	14.57	15.85	17.48
Upper Seds: Trace Ox (H)	13.29	14.66	15.95	17.59
Upper Seds: No Ox (L)	13.69	15.09	16.42	18.11
Upper Seds: No Ox (H)	14.18	15.63	17.02	18.76
Lower Sand: Partial Ox (L)	15.36	16.93	18.43	20.32
Lower Sand: Partial Ox (H)	15.53	17.12	18.63	20.54
Lower Sand: Trace Ox (L)	15.92	17.55	19.11	21.06
Lower Sand: Trace Ox (H)	15.23	16.79	18.27	20.14
Lower Sand: No Ox (L)	15.18	16.73	18.21	20.08
Lower Sand: No Ox (H)	15.36	16.93	18.43	20.32
Amy Sequence: Partial Ox (L)	12.51	13.80	15.02	16.56
Amy Sequence: No Ox (L)	9.23	10.18	11.08	12.21
Average	**13.14**	**14.49**	**15.77**	**17.39**

TABLE 16.7
Livengood Project Summary of Abrasion Test Results
Phillips Report 093029_15
October, 2009

Description	Rocktype	Alteration Type	Ai
Upper Seds: Partial Ox	Siltstone	Sericite	0.0023
Upper Seds: No Ox	Siltstone	Sericite	0.1497
Upper Seds: Partial Ox	Sandstone	Sericite	0.0120
Upper Seds: Partial Ox	Shale	Albite Mica	0.0848
Lower Seds: No Ox	Shale	Sericite	0.0189
Main Volcanics: No Ox	Andesite	Mixed Albite Mica Kspar	0.0391
Main Volcanics: Partial Ox	Volcanic Breccia	Albite	0.2872
Main Volcanics: Partial Ox	Volcanic Breccia	Clay Mica	0.1151
Main Volcanics: Partial Ox	Tuff	Sericite	0.1627
Main Volcanics: Partial Ox	Tuff	Albite Mica	0.0643
Amy Sequence: Trace Ox	Chert	Albite Mica	0.2040
Cambrian: Partial Ox	Serpentinite	No K or Na	0.0111
Cambrian: Partial Ox	Listwanite	Dolomite Clay Mica	0.0161
Cambrian: Trace Ox	Serpentinite	No K or Na	0.0343

16.5.2 Gravity Centrifugal Concentration Evaluation

The Knelson® Gravity Recoverable Gold (GRG) tests were performed. The test consists of three sequential liberation and recovery stages. The progressive grinding was necessary in order to obtain an accurate GRG value, an indication of the size distribution of the GRG and a measure of progressive liberation. It also limits any smearing of coarse gold particles that may be present in the as-crushed sample.
The GRG test is based on the treatment of a sample mass of typically 20 Kg using a laboratory Knelson Concentrator (KC-MD3). **Table 16.8** summarizes the test procedure.

TABLE 16.8
Procedures for Knelson Concentrator Testwork

Sample Require-ments	30 Kg of sample is required to perform a standard GRG test. 20 Kg of sample is required for the GRG test and the other 10 Kg sample is used for a grinding test prior to running the GRG.		
	Particle Size Requirements	Operating Variables	Sample collection
Stage 1	90 - 100% -850 µm	Feed Rate: 800-1000 g/min Fluid'n Water (FW): 3.5 l/min	• Total Knelson concentrate for fire assay to extinction* • 300 gr. tail sample for fire assay • Bulk tails to stage 2
Stage 2	45 - 60% -75 µm	Feed Rate: 600-900 g/min F.W: 3.5 l/min	• Total Knelson concentrate for fire assay to extinction* • 300 gr. tails sample for fire assay • Bulk tails to stage 3
Stage 3	75 - 80% -75 µm	Feed Rate: 400-800 g/min F.W: 3.5 l/min	• Total Knelson concentrate for fire assay to extinction* • 300 gr. of tails for fire assay

* the concentrate can be panned for a visual observation of the concentrate - the panned products should then be assayed to extinction.

Note that it is not necessary to perform the test grind with 10 kg as this step has been previously performed in the grind studies portion of the test work.

Stage recoveries were based on the concentrate and tail assay of each stage. However, overall recovery is based on the assays of the three concentrates produced and the tails product of the third recovery stage, whose assays are more reliable than those of the first two, which still contain some of the GRG. Gold assays on the products will be by fire assay and in duplicate when sufficient sample exists.

Table 16.9 provides the results from this test work.

TABLE 16.9
LIVENGOOD PROJECT
KNELSON CONCENTRATOR - GRAVITY RECOVERABLE SUMMARY

Description	Calculated Head, Au g/t	Conc + Mid Wt. %	Conc + Mid Assay, Au g/t	Conc + Mid Rec % Au	Conc + Mid Assay, Ag g/t	Conc + Mid Rec % Ag
Overburden: Partial Ox (L)	0.55	1.3%	20.79	49.6%	14.3	7.3%
Cambrian: Partial Ox (L)	0.62	1.5%	28.16	66.4%	13.8	9.1%
Cambrian: Partial Ox (H)	1.34	1.3%	76.95	76.2%	15.6	10.9%
Cambrian: Trace Ox (L)	0.63	1.5%	29.88	69.8%	9.1	6.2%
Cambrian: Trace Ox (H)	1.59	1.5%	89.01	82.0%	13.8	10.7%
Kint: Partial Ox (L)	0.80	1.5%	19.07	35.6%	5.5	3.4%
Kint: Partial Ox (H)	1.67	1.5%	36.90	33.1%	10.2	5.4%
Kint: Trace Ox (L)	0.96	1.5%	25.72	40.6%	7.6	4.1%
Kint: Trace Ox (H)	1.41	1.5%	41.93	45.1%	7.2	5.1%
Kint: No Ox (L)	0.77	1.5%	15.77	31.1%	4.5	2.8%
Kint: No Ox (H)	1.40	1.5%	37.31	40.9%	5.6	2.8%
Lower Seds: Trace Ox (L)	1.12	1.5%	38.88	52.3%	5.8	3.2%
Lower Seds: Trace Ox (H)	1.21	1.5%	40.88	51.7%	12.0	8.3%
Lower Seds: No Ox (L)	0.75	1.5%	32.79	63.9%	6.3	4.4%
Lower Seds: No Ox (H)	1.21	1.5%	55.36	67.2%	9.5	6.5%
Main Volcanics: Partial Ox (L)	0.79	1.4%	22.37	40.1%	7.2	4.8%
Main Volcanics: Partial Ox (H)	1.80	1.4%	87.75	70.2%	12.9	9.9%
Main Volcanics: Trace Ox (L)	0.90	1.5%	23.83	38.7%	5.5	2.9%
Main Volcanics: Trace Ox (H)	1.65	1.5%	54.10	48.3%	7.9	4.2%
Main Volcanics: No Ox (L)	0.86	1.5%	23.20	40.1%	4.1	3.0%
Main Volcanics: No Ox (H)	1.84	1.5%	52.81	43.5%	6.2	3.4%
Upper Seds: Partial Ox (L)	0.84	1.4%	30.70	50.4%	6.6	6.3%
Upper Seds: Partial Ox (H)	1.42	1.4%	57.79	58.9%	8.9	7.0%
Upper Seds: Trace Ox (L)	0.80	1.4%	36.33	63.5%	8.2	4.6%
Upper Seds: Trace Ox (H)	1.42	1.4%	73.57	72.9%	10.1	6.6%
Upper Seds: No Ox (L)	0.84	1.4%	39.56	65.3%	8.3	4.7%
Upper Seds: No Ox (H)	1.11	1.4%	58.55	73.8%	8.1	5.3%
Lower Sand: Partial Ox (L)	1.09	1.5%	42.34	57.6%	8.0	6.6%
Lower Sand: Partial Ox (H)	1.42	1.4%	63.66	65.0%	11.8	6.0%
Lower Sand: Trace Ox (L)	0.99	1.4%	44.22	63.7%	9.2	5.3%
Lower Sand: Trace Ox (H)	1.34	1.5%	58.08	64.6%	9.4	7.6%
Lower Sand: No Ox (L)	0.72	1.4%	28.13	56.5%	6.0	3.1%
Lower Sand: No Ox (H)	1.48	1.4%	74.67	71.8%	11.5	5.8%
Amy Sequence: Partial Ox (L)	0.40	1.3%	15.00	49.3%	4.2	2.6%
Amy Sequence: No Ox (L)	0.57	1.4%	24.19	60.2%	7.1	3.6%
Averages		**1.45%**	**42.9**	**56.0%**	**8.6**	**5.5%**

(L) - 0.5 ≤ Au g/t ≤ 1.0; (H) - 1.0 ≤ Au g/t ≤ 5.0

The gold in the Livengood ores appears to respond well to gravity separation ranging up to 73.8% for higher grade mineralization in unoxidized Upper Sediments. However, on average, the gravity separation was able to recover ~56% of the gold shown to exist in assays.

16.5.3 Bottle Roll Leach Tests

Composite samples were be used to run 72 hour bottle roll tests. For those composites with adequate amount of sample, bottle roll tests were run at 10 mesh, 100 mesh, and 200 mesh grinds. Each bottle roll test had solution removed for Au assay at 2, 4, 8, 12, 24, 36, 48, and 72 hour intervals. Cyanide and pH levels were also be checked as often as necessary to maintain reagents at adequate leach conditions. Reagent consumptions were monitored and lime and cyanide consumptions were calculated.

Composites with insufficient amounts had only 72 hour bottle rolls run on them at the -200 mesh grind size.

Table 16.10 provides a summary of the bottle roll test results. As the results indicate, most of the Livengood ores did respond positively to cyanide leaching. The bottle roll results were considerably better than the cyanide shake leach results. However, gold leach recoveries appear to be highly variable by ore type. The degree of oxidation also appears to have an effect on the gold cyanide leachability.

16.5.4 Bottle Roll CIL Tests

After reviewing the data from the bottle roll leach tests and the cyanide shake leach tests, it was determined that bottle roll Carbon-in-Leach (CIL) tests should be performed to establish if the poor response to cyanide leaching by some of the ore types was due to ore "preg robbing" issues. Thus, the same composite samples were used to run 92 hour bottle roll CIL tests. All of the bottle roll CIL tests were run at 200 mesh grinds. Cyanide and pH levels were checked as often as necessary to maintain reagents at adequate leach conditions. Reagent consumptions were monitored and lime and cyanide consumptions were calculated.

Results of the CIL bottle roll tests show that recoveries improved significantly (**Table 16.11**)., some ore types as high as 49.5% increase in overall gold recovery, with the addition of carbon in the cyanide leach process. It appears that some of the ores have "preg robbing" characteristics, which explains the poor response observed in the cyanide shake leach tests. Fortunately, the presence of activated carbon offsets, to a major degree, the "preg robbing" nature of the ores.

TABLE 16.10
LIVENGOOD PROJECT
SUMMARY OF CYANIDE BOTTLE ROLL TESTS

Description	Calculated Head, Au g/t	Overall Average Assay, Au g/t	Au Extracted, %	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
Overburden: Partial Ox (L)	0.63	0.66	87%	0.43	2.80
Cambrian: Partial Ox (L)	0.80	0.75	80%	0.26	3.00
Cambrian: Partial Ox (H)	1.35	1.82	83%	0.35	2.08
Cambrian: Trace Ox (L)	0.61	0.66	87%	0.32	2.00
Cambrian: Trace Ox (H)	1.48	1.82	90%	0.33	2.50
Kint: Partial Ox (L)	0.69	0.73	54%	0.51	4.50
Kint: Partial Ox (H)	1.67	1.55	60%	0.99	3.33
Kint: Trace Ox (L)	0.82	0.76	24%	0.38	2.50
Kint: Trace Ox (H)	1.24	1.23	22%	0.41	2.50
Kint: No Ox (L)	0.84	0.76	32%	0.29	2.80
Kint: No Ox (H)	2.42	1.51	32%	0.81	3.00
Lower Seds: Trace Ox (L)	1.05	0.88	0%	0.28	2.33
Lower Seds: Trace Ox (H)	1.20	1.36	1%	0.38	2.00
Lower Seds: No Ox (L)	0.62	0.65	0%	1.78	2.00
Lower Seds: No Ox (H)	1.36	1.28	0%	0.35	2.00
Main Volcanics: Partial Ox (L)	0.75	0.68	56%	0.30	2.67
Main Volcanics: Partial Ox (H)	1.43	1.68	77%	0.36	3.17
Main Volcanics: Trace Ox (L)	0.82	0.75	36%	0.51	2.00
Main Volcanics: Trace Ox (H)	1.45	1.42	42%	2.21	2.00
Main Volcanics: No Ox (L)	0.97	0.91	49%	0.20	2.00
Main Volcanics: No Ox (H)	1.66	2.10	39%	2.13	2.00
Upper Seds: Partial Ox (L)	0.71	1.05	64%	0.35	2.00
Upper Seds: Partial Ox (H)	1.45	1.50	80%	0.37	2.00
Upper Seds: Trace Ox (L)	0.89	0.97	37%	0.22	2.00
Upper Seds: Trace Ox (H)	1.67	1.58	73%	0.42	2.00
Upper Seds: No Ox (L)	0.76	0.85	26%	0.31	2.00
Upper Seds: No Ox (H)	1.28	1.68	55%	0.32	2.00
Lower Sand: Partial Ox (L)	0.91	0.86	49%	1.98	2.00
Lower Sand: Partial Ox (H)	1.10	1.52	61%	0.63	2.00
Lower Sand: Trace Ox (L)	1.09	0.94	48%	0.39	2.50
Lower Sand: Trace Ox (H)	1.35	1.32	67%	0.40	2.33
Lower Sand: No Ox (L)	0.68	0.75	21%	0.45	2.00
Lower Sand: No Ox (H)	1.32	1.28	55%	0.55	2.33
Amy Sequence: Partial Ox (L)	0.39	0.70	49%	0.24	2.60
Amy Sequence: No Ox (L)	0.52	0.51	4%	0.22	2.50
Overall Average	**1.09**	**1.13**	**47%**	**0.59**	**2.38**

TABLE 16.11
LIVENGOOD PROJECT
SUMMARY OF CIL CYANIDE BOTTLE ROLL TESTS

Description	Calculated Head, Au g/t	Overall Average Assay, Au g/t	CIL Au Rec, %	BRT Au Rec, %	Difference between CIL and BRT Au Rec, %
Overburden: Partial Ox (L)	0.63	0.66	86.7%	87.2%	-0.5%
Cambrian: Partial Ox (L)	0.44	0.44	89.0%	80.0%	9.0%
Cambrian: Partial Ox (H)	1.33	1.33	94.0%	83.3%	10.7%
Cambrian: Trace Ox (L)	0.58	0.58	95.0%	87.0%	8.0%
Cambrian: Trace Ox (H)	1.64	1.64	95.0%	89.8%	5.2%
Kint: Partial Ox (L)	0.68	0.68	59.0%	54.0%	5.0%
Kint: Partial Ox (H)	1.52	1.52	59.0%	60.3%	-1.3%
Kint: Trace Ox (L)	0.78	0.78	42.0%	24.0%	18.0%
Kint: Trace Ox (H)	1.33	1.33	40.0%	21.5%	18.5%
Kint: No Ox (L)	0.76	0.76	49.0%	31.6%	17.4%
Kint: No Ox (H)	1.21	1.21	43.0%	32.2%	10.8%
Lower Seds: Trace Ox (L)	0.84	0.84	40.0%	0.0%	40.0%
Lower Seds: Trace Ox (H)	5.18	5.18	79.0%	0.5%	78.5%
Lower Seds: No Ox (L)	0.51	0.51	41.0%	0.0%	41.0%
Lower Seds: No Ox (H)	1.20	1.20	63.0%	0.0%	63.0%
Main Volcanics: Partial Ox (L)	0.76	0.76	71.0%	55.7%	15.3%
Main Volcanics: Partial Ox (H)	2.14	2.14	85.0%	76.7%	8.3%
Main Volcanics: Trace Ox (L)	0.92	0.92	63.0%	36.2%	26.8%
Main Volcanics: Trace Ox (H)	1.24	1.24	39.0%	41.7%	-2.7%
Main Volcanics: No Ox (L)	1.11	1.11	65.0%	49.0%	16.0%
Main Volcanics: No Ox (H)	2.31	2.31	23.0%	38.7%	-15.7%
Upper Seds: Partial Ox (L)	0.74	0.74	72.0%	64.0%	8.0%
Upper Seds: Partial Ox (H)	1.37	1.37	87.0%	80.3%	6.7%
Upper Seds: Trace Ox (L)	0.63	0.63	67.0%	36.5%	30.5%
Upper Seds: Trace Ox (H)	1.46	1.46	83.0%	73.0%	10.0%
Upper Seds: No Ox (L)	0.78	0.78	73.0%	26.2%	46.8%
Upper Seds: No Ox (H)	1.25	1.25	82.0%	54.8%	27.2%
Lower Sand: Partial Ox (L)	0.93	0.93	57.0%	49.2%	7.8%
Lower Sand: Partial Ox (H)	1.99	1.99	53.0%	61.4%	-8.4%
Lower Sand: Trace Ox (L)	1.01	1.01	60.0%	47.7%	12.3%
Lower Sand: Trace Ox (H)	1.32	1.32	62.0%	66.5%	-4.5%
Lower Sand: No Ox (L)	0.80	0.80	70.0%	20.5%	49.5%
Lower Sand: No Ox (H)	1.03	1.03	76.0%	54.5%	21.5%
Amy Sequence: Partial Ox (L)	0.47	0.47	79.0%	48.8%	30.2%
Overall Average	**1.20**	**1.20**	**65.9%**	**48.0%**	**17.9%**

16.6 Future Metallurgical Test Work

Test work currently envisioned for the Livengood project includes the following:

CIL Tests
- Grind Size Effects
- Leach Time Effects
- Cyanide Strength Tests
- Carbon Addition Concentration Tests
- Lead Nitrate Tests

Organic Carbon Chemical Oxidation Tests
- Oxidizer Tests

Gravity Tests
- Grind Size Effects
- Gravity Concentrate Fine Grind and Cyanidation Tests
- Gravity Tails Leach and CIL Tests

Flotation Tests
- Collector and Depressant Tests
- Grind Size Effects
- Flotation Time Effects
- Flotation Concentrate and Tails Leach Tests

Gravity and Flotation Concentrate Tests
- Fine Grinding
- Pre-aeration
- Chemical Oxidation
- High Intensity Cyanide Leach Tests

Column Tests
- Crush Size Effects
- Leach Time Effects
- Cyanide Strength Tests

Metallurgical test work program costs should be budgeted for Livengood at $750,000 for the next six months to cover the test work detailed above.

16.7 Mineral Processing

Based on the test work discussed previously and on the current estimated ore resource, process options were investigated. The process envisioned for this document involves crushing the run of mine ore in a three stage crushing circuit to less than 3/4 inch, and placing the crushed ore on a lined heap leach pad and utilizing conventional heap leaching technologies. **Figure 16.2** presents a simple block flow diagram of the proposed circuit. The placement of ore on the heap would likely not be performed during the coldest three months of the year due to arctic winter temperatures. However, leaching would continue twelve months of the year since all piping, pregnant ponds, and solution application drip irrigators would be buried within the heap to prevent freezing.

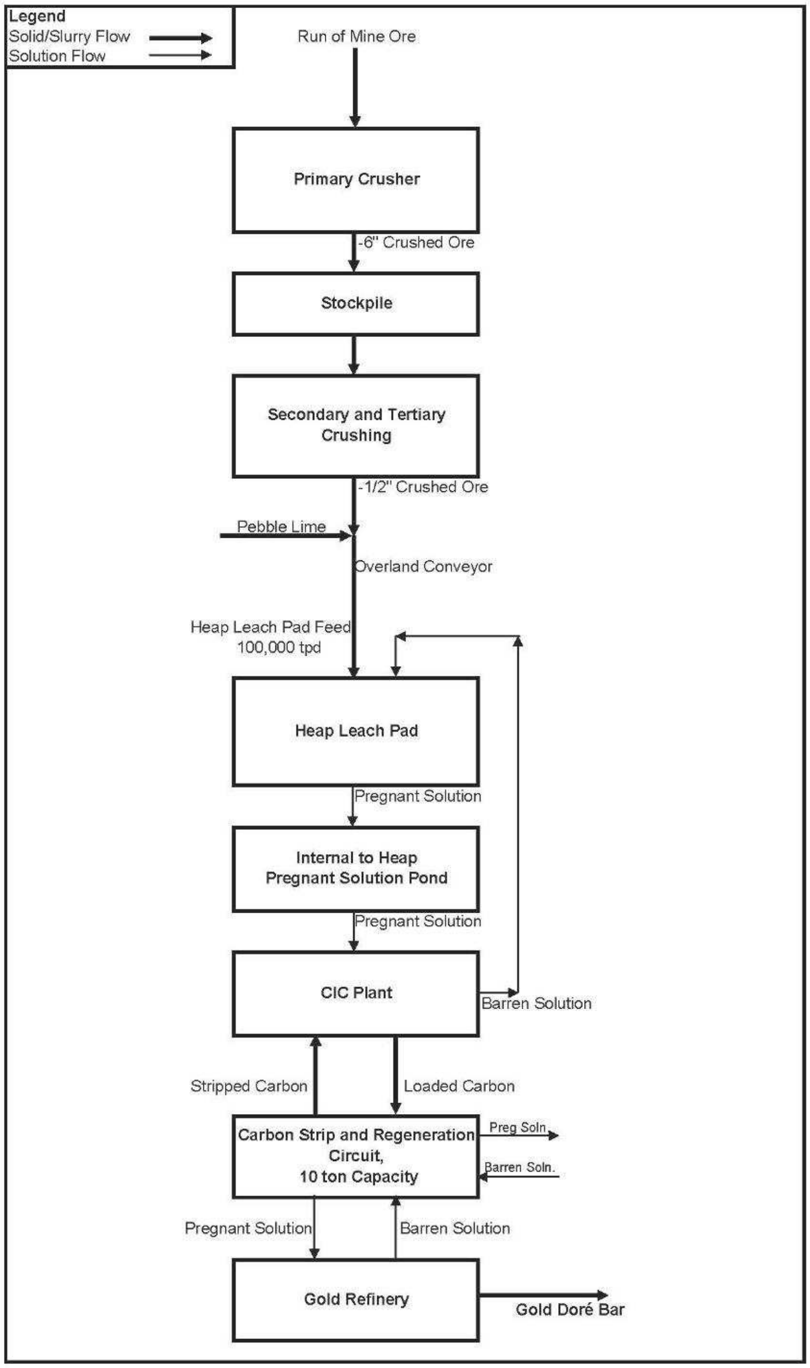


Figure 16.2 Proposed Livengood process block flow diagram showing heap leach process streams.

Future mineral processing investigations will likely focus on a combination of a Mill / CIL process facility in conjunction with the heap leach facility. **Figure 16.3** presents a simple block flow diagram of the alternate circuit.

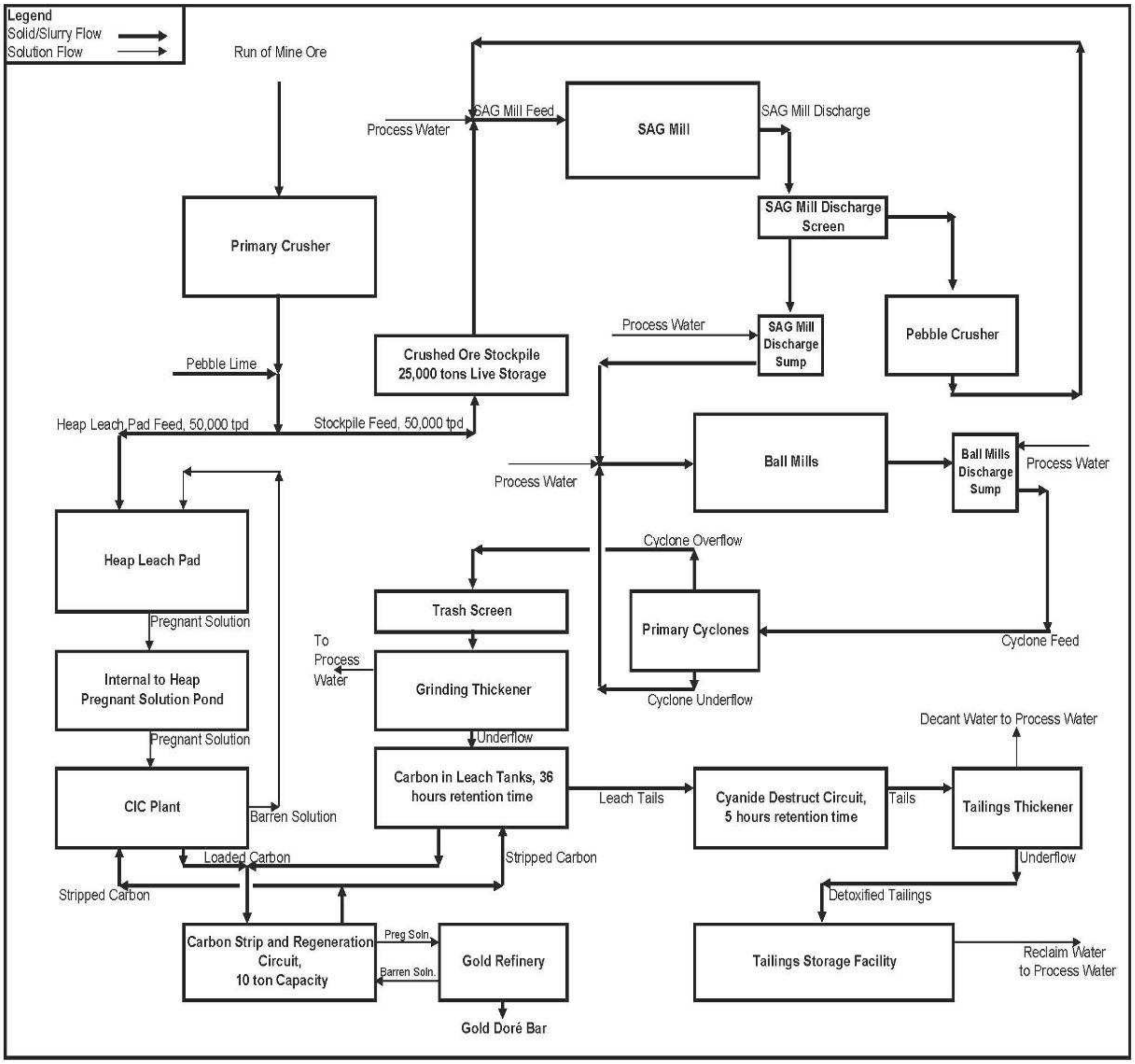


Figure 16.3 Alternate Livengood process block flow diagram showing both heap leach and mill / cil process streams

16.8 Gold Recovery

Utilizing existing test work data and industry experience, and applying the process scenarios described previously, an estimation of the gold recovery by ore type has been performed. **Table 16.12** provides the gold recoveries as currently estimated. As shown in the table, the Lower Sediments are carrying a zero percent gold recovery and for the purposes of this study are not included as ore. The Lower Sediment samples that have zero recovery are the shale facies of this unit (c.f. the Lower Sand). Because there appear to be some carbon effects in the volcanic and upper sediments as well, it is thought that this could relate to the local presence of shale in these units. As a consequence the recovery of all shale in the deposit has been assumed to be zero and blocks containing shale have been treated as waste. These ore types will be brought back into future studies when the Mill / CIL circuit is included as one of the proposed circuits.

TABLE 16.12

Gold Recovery Estimates by Ore Type for Heap Leach and Mill/CIP Process Scenarios

Ore Type	Heap Leach % Au Rec	Mill / CIL % Au Rec
Overburden	75.0%	87.0%
Cambrian Oxidized	80.0%	92.0%
Cambrian Trace	82.0%	95.0%
Cambrian Unoxidized	70.0%	75.0%
Upper Seds Oxidized	75.0%	87.0%
Upper Seds Trace	45.0%	82.0%
Upper Seds Unoxidized	35.0%	77.0%
Kint Oxidized	50.0%	59.0%
Kint Trace	22.0%	46.0%
Kint Unoxidized	22.0%	46.0%
Main Volcanics Oxidized	60.0%	79.0%
Main Volcanics Trace	40.0%	65.0%
Main Volcanics Unoxidized	35.0%	65.0%
Lower Seds Oxidized	0.0%	72.0%
Lower Seds Trace	0.0%	60.0%
Lower Seds Unoxidized	0.0%	60.0%
Lower Sand Oxidized	50.0%	63.0%
Lower Sand Trace	50.0%	63.0%
Lower Sand Unoxidized	15.0%	63.0%
Amy Sequence Oxidized	35.0%	79.0%
Amy Sequence Trace	35.0%	79.0%
Amy Sequence Unoxidized	27.0%	35.0%

16.9 Process Operating Costs

Process operating costs have been developed on each of the identified Livengood ore stratigraphic units (**Table 16.13**). The test work performed on each of the ore types has provided preliminary data for calculating reagent and grinding media consumptions, and power consumptions by ore type. The size of the facility envisioned and the unit processes involved also enabled preliminary maintenance and manpower requirements to be created for a 100,000 tpd process facility at Livengood. Alaskan wage rates were applied to the various staff and operating and maintenance positions. Unit costs for reagents and power were obtained from a survey of properties operating in the region.

The operating costs provided include power costs at $0.135 per kWh.

The 100 ktpd Stand Alone Heap Leach operating costs have been used in the financial calculations provided later in this report.

TABLE 16.13

Primary Ore Types Estimated Process Operating Costs

For Heap Leach at 100,000 TPD Throughput Rate

Ore Type	100 ktpd Stand Alone Heap Leach Op. Cost, $/t
Overburden	$ 2.79
Cambrian Oxidized	$ 3.05
Cambrian Trace	$ 3.38
Cambrian Unoxidized	$ 3.38
Upper Seds Oxidized	$ 3.23
Upper Seds Trace	$ 3.36
Upper Seds Unoxidized	$ 3.36
Kint Oxidized	$ 3.97
Kint Trace	$ 3.58
Kint Unoxidized	$ 3.58
Main Volcanics Oxidized	$ 3.62
Main Volcanics Trace	$ 3.17
Main Volcanics Unoxidized	$ 2.70
Lower Seds Oxidized	$ 3.23
Lower Seds Trace	$ 3.36
Lower Seds Unoxidized	$ 3.36
Lower Sand Oxidized	$ 3.23
Lower Sand Trace	$ 3.36
Lower Sand Unoxidized	$ 3.36
Amy Sequence Oxidized	$ 3.05
Amy Sequence Trace	$ 3.38
Amy Sequence Unoxidized	$ 3.58

16.10 Process Capital Cost

Preliminary capital costs were prepared for this study (**Table 16.14**). The capital costs that were developed are based on the following assumptions:

- Strip Ratio of 1:1, waste to ore
- 100,000 tpd ore production rate
- Heap Leach processing scenario
- Process facility would be relatively close to the Livengood ore deposits
- Power to the site would be via a new power line included in the capital cost
- The existing access to the site would be sufficient to support construction and operating requirements for Livengood

Capital costs for the different areas shown in the table were obtained from recent projects of similar size and scope, located in north western Canada and Alaska. In an area where the size may have been different, the area was scaled using the rule of thumb factors.

TABLE 16.14
LIVENGOOD CAPITAL COST ESTIMATE
100,000 TPD HEAP LEACH

Area	**Total (US$ Millions)**
Crushing	105.3
Carbon Stripping and Regeneration	18.6
Refining	4.9
Reagents	22.0
Plant Mobile Equipment	50.8
Leach Pad and CIC Plant	98.8
Site Roads	10.5
Power Line	32.0
Administrative Building	4.9
Mine Truckshop and Warehouse	25.7
Subtotal	**$373.6**
Indirect Costs @ 50%	**$186.8**
Total Installed Costs	**$560.4**

16.11 Mining Capital Estimate

ITH commissioned SRK to estimate the capital costs associated with several different development scenarios for Livengood. The direct mining capital was estimated for operations ranging in size from 50ktpd to 150ktpd assuming a strip ratio of 1:1 (**Table 16.14**). The direct capital costs for the 100ktpd operations were extrapolated from the two capital estimates in **Table 16.15** and adjusted for the actual strip ratio of 0.78:1. A contingency of 15% was added resulting in a final direct mining capital cost of USD$104.7M.

Table 16.15

Direct Mining Capital Estimates Assuming a 1:1 Strip Ratio (US$000's)

Capital Item	150ktpd	50ktpd
Cable Shovels	$46,488	$16,776
Front End Loader	$8,567	$4,592
Haul Trucks	$50,050	$19,699
Rotary Drills	$18,824	$17,376
Dozers	$5,702	$4,615
Graders	$3,276	$1,602
Water Trucks	$1,906	$1,454
Fuel / Lube Trucks	$360	$332
Service Trucks	$436	$402
Tire Trucks	$411	$379
Bulk Loader	$293	$180
Light Plants	$854	$526
Pumps	$1,150	$455
Pickups	$953	$643
Crane	$358	$330
Forklift	$130	$120
Cold Weather Heaters	$912	$360
Direct Min	**$140,668**	**$69,842**

16.12 Sustaining Capital

The initial capital estimate for the heap leach included the construction of only three years capacity for the heap leach pad. Each additional three years capacity was estimated to cost US$80M or US$27M per year. Because the overall mine life was 13 years this capital was added in US$40M allocations in years 2, 3, 5, 6, 8 and 9 with US$27M added in Year 11. An additional USD$30M was added in USD$10M allocations in years 8, 9, and 10 for general replacement of equipment due to wear and tear. The total sustaining capital was USD$297M.

17.0 Mineral Resource Estimate and Preliminary Economic Assessment

A mineral resource estimate was prepared for the Livengood deposit using information available through September 25th 2009. The drill data was maintained in a Gemcom® GEMS database, the basic statistical analysis was performed using SAGE2001® and WinGSLib® for validation purposes.

The resource model was constructed using Gemcom GEMS® and WinGSLIB. The mineral resource model was estimated using multiple indicator kriging (**MIK**) for gold. Two oxidation indicators were used to estimate the oxidation and a single indicator was used to estimate the distribution of Kint dikes, Lower Sands, Amy Sequence and Shale. A three-dimensionally defined lithology model, developed by ITH geologists, was used to constrain the gold estimation. A summary mineral resource at cutoff grades of 0.3, 0.5, 0.7, and 0.9 g/t gold is shown in **Table 17.1.**

Reasonable economic assumptions were made and pits were optimized utilizing Whittle® software to allow a Preliminary Economic Assessment (PEA) for a heap leach operation to be developed that would demonstrate reasonable prospects for economic extraction. The assumptions are believed to be reasonable but should be considered preliminary in nature. A mining cost of $1.80 per tonne mined was assumed. This is consistent with a neighboring operating mine with similar conditions and scale.

TABLE 17.1
SUMMARY MINERAL RESOURCE

Classification	Au Cutoff (g/t)	Tonnes (millions)	Au (g/t)	Million Ounces Au
Indicated	0.30	525	0.65	11.0
Inferred	0.30	336	0.61	6.6
Indicated	0.50	297	0.85	8.1
Inferred	0.50	164	0.84	4.4
Indicated	0.70	158	1.07	5.4
Inferred	0.70	78	1.11	2.8
Indicated	0.90	78	1.36	3.4
Inferred	0.90	38	1.46	1.8

Heap leach processing costs ranging from $3.30 to $4.22 per tonne heap leached were used. The processing costs varied by rock type and oxidation state as indicated by preliminary metallurgical testing. An additional $0.60 per tonne processed was assumed for general and administrative costs. Cutoff grades were calculated on a break-even cost basis and varied by process cost and metallurgical recovery for different rock types and oxidation states. A base case assumption of $850 per troy ounce gold was used. Economic value was applied to both Indicated and Inferred material to develop the preliminary pit shells. The **base case PEA is summarized in Table 17.2.**

TABLE 17.2

Livengood Project - Heap Leach PEA – Base Case Summary*

In-pit resource - Indicated :	**308Mt @ 0.68 g/t gold for 6.7M contained ounces**
In-pit resource - Inferred	**132Mt @ 0. 71g/t gold for 3.01M contained ounces**
Over all strip ratio of :	**1 to 0.78 (ore-waste)**
Annual Production:	**459,033 ounces gold, total of 5,783,813 recoverable ounces**
Average gold recovery:	**60%**
Mining rate:	**100,000 ore tonnes per day, 178,000 total tonnes per day**
Mining cost per tonne:	**$1.80**
Processing cost per tonne:	**$3.80**
G&A cost per tonne:	**$0.60**
Cost per ounce:	**$533**
Initial capital costs:	**$665M, life of project sustaining capital $297M**
Contingency:	**20%**

*(All values in USD and based on $700 Whittle optimized pit shell)

It is believed that reasonable assumptions have been made and are sufficient to demonstrate reasonable prospects for economic extraction. These results are preliminary in nature, they include inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them, and there is no certainty that this preliminary assessment will be realized.

17.1 Data Used

17.1.1 Sample Data

The data available for this model comprised 83,200 meters of core and RC drilling, plus trench data. Historical drilling and sampling is shown in **Table 17.3**. Drilling performed by TGA is shown in **Table 17.4**. It can be seen that the historical data represents about 6% of the total information used.

TABLE 17.3

HISTORICAL DRILLING AND SAMPLING

Year	Company	Drill Type	Number of Holes	Meters
1976	Homestake	Percussion	4	153
1981	Occidental	Percussion	6	310
1989	AMAX	Trench	2	160
1990	AMAX	RC	3	320
1997	Placer Dome	Core	9	1,100
2003	Anglogold	RC	8	1,514
2004	Anglogold	Trench	8	276
2004	Anglogold	Core	4	762
Total			47	4,746

TABLE 17.4

ITH DRILING AND SAMPLING

Year	Drill Type	Number of Holes	Meters
2006	Core	7	1,227
2007	Core	15	4,411
2008	Core	7	2,040
2008	Trench	4	80
2008	RC	108	28,619
2009	RC	167	46,823
Total		308	83,200

17.1.2 Other Data

Topography

The topographic surface used is based on a 4m DEM derived from 2008 aerial photography.

Density

Densities used in the resource are based on 98 determinations from core samples and are shown in **Table 17.5**.

TABLE 17.5
DENSITY DETERMINATIONS

Lithology Unit	N	Mean	StdDev	Max	Min
Amy Sequence	4	2.67	0.04	2.72	2.65
Cambrian	12	2.82	0.07	2.95	2.69
Combined Cambrian-Amy		2.78			
Kint	3	2.56	0.18	2.76	2.44
Lower Sediments	21	2.74	0.05	2.84	2.62
Main Volcanics	36	2.72	0.13	2.86	2.11
Upper Sediments	22	2.68	0.13	2.79	2.23
Average of all readings	98	2.72			

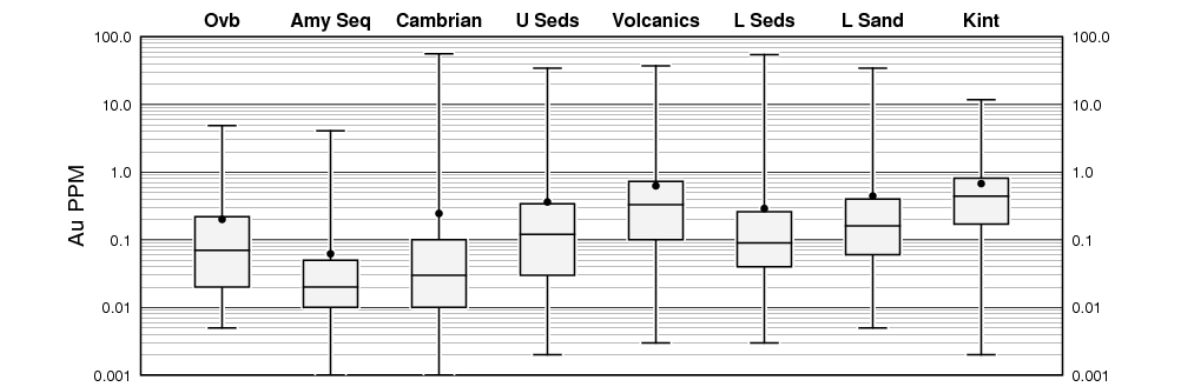
17.2 Data Analysis

Multi-element assay information is available for less than 50% of the samples. A statistical summary of this data from a previous report (July 09) is shown in **Table 17.6**. The only element of economic significance is gold, which was the only element modeled in the resource model. No significant correlations were found between the various elements. There were numerous weak to moderate correlations, but nothing that could be exploited to improve the gold estimate. Based on the lack of significant correlations previously determined, the exercise was not updated for this estimate

TABLE 17.6
STATISTICAL SUMMARY OF ASSAY DATA

Element	Units	N	Mean	Maximum	Std.Dev.	C.V.
Au	ppm	34786	0.40	56.2	1.22	3.0
Ag	ppm	12969	0.41	440	4.07	10.0
Cu	ppm	12969	42	1120	34	0.8
Pb	ppm	12969	19	9240	128	6.7
As	ppm	12971	2169	137000	4181	1.9
Sb	ppm	12969	221	138000	2394	10.8
Zn	ppm	12969	186	3440	221	1.2
Fe	%	12708	4.3	21.3	1.4	0.3
Mo	ppm	12969	5.5	74.0	6.9	1.3
S	%	12081	1.4	18.4	1.4	1.0
Te	ppm	12063	0.16	25.1	0.5	3.0

Each of the assay intervals were also logged for lithology, alteration and mineralization. Of all of the available qualitative data, the lithology appears to exert the most influence on the gold mineralization (**Figure 17.1**). It is still a matter of geological debate as to exactly why this is so, but the volcanic unit is preferentially mineralized relative to the units above and below it. Also, the Kint dikes, which appear to be the conduits for much of the mineralization, are also well mineralized. Not only are the volcanics and Kint dikes higher grade, they are uniformly well mineralized as shown by the relatively low coefficient of variation (C.V.) of each unit.



	Ovb	Amy Seq	Cambrian	U Seds	Volcanics	L Seds	L Sand	Kint	
Number of data	679	2604	9384	18583	8299	5695	2068	2071	Number of data
Mean	0.2	0.062	0.245	0.362	0.628	0.29	0.444	0.676	Mean
Maximum	4.86	4.06	56.2	34.2	37.4	54.5	34.4	11.75	Maximum
Upper quartile	0.22	0.05	0.1	0.34	0.73	0.26	0.4	0.81	Upper quartile
Median	0.07	0.02	0.03	0.12	0.33	0.09	0.16	0.44	Median
Lower quartile	0.02	0.01	0.01	0.03	0.1	0.04	0.06	0.17	Lower quartile
Minimum	0.005	0.001	0.001	0.002	0.003	0.003	0.005	0.002	Minimum
Variance	0.1551	0.0339	1.8873	0.9688	1.6029	1.0067	1.5588	0.9221	Variance
CV	1.97	2.958	5.614	2.72	2.017	3.457	2.812	1.42	CV
Skewness	5.452	12.319	20.666	11.621	10.209	31.194	14.533	5.038	Skewness

Figure 17.1 Gold distribution by lithology unit.

17.3 Geological Model

ITH geologists produced a three dimensional wire framed geological model of the major lithologic units and major fault structures. South of the Lillian Fault, the rock units modeled were the Cambrian, Upper Sediments, Main Volcanics, and the Lower Sediments. North of the Lillian fault most everything is undifferentiated Upper Sediments with a small amount of Volcanics modeled. These represent the major lithologic units that host the mineralization. No other controls were modeled.

17.4 Composite Statistics

All of the available drilling was composited into fixed length 10m composites. Composite residuals <4m in length were added to the previous composite. These composites were back-tagged with the lithology using the defined geological model using the three-dimensional wire frames.

The composite data was declustered using a cell declustering technique. The composite statistics are weighted using the decluster weights (**Table 17.7**).

TABLE 17.7

GOLD COMPOSITE STATISTICS

Mean:	0.31
Variance:	0.30
C. of V.:	1.66
Min:	0
Q1:	0.04
Median:	0.15
Q3:	0.39
Max:	10.99

17.4.1 Gold Indicator Statistics

The composite data was used to set the gold indicator thresholds. Since the coefficient of variation of the composite data is relatively low, only nine indicator thresholds were needed to fully define the gold distributions. The indicator thresholds were chosen at the low end to have approximately 20% of the data per class and at the high end to have 11 to 12% of the metal per class (**Table 17.9**). With MIK, top cutting of the assays is not necessary. In this case all composite values greater than 3.1 g/t Au (the highest threshold) are treated the same as "high grade" and the median value of 4.36 g/t Au is used to evaluate the highest class.

TABLE 17.9

GOLD INDICATOR STATISTICS

		Data		Metal			
	Threshold	%	Cum%	%	Cum%	Mean	Median
1	0.05	20.0	20.0	1.1	1.1	0.021	0.02
2	0.15	21.1	41.1	5.1	6.2	0.090	0.09
3	0.35	24.0	65.1	14.9	21.1	0.234	0.23
4	0.50	10.4	75.5	11.4	32.5	0.413	0.41
5	0.65	7.7	83.2	11.6	44.2	0.565	0.56
6	0.85	6.3	89.5	12.4	56.5	0.739	0.74
7	1.10	4.0	93.5	10.2	66.7	0.954	0.95
8	1.60	3.8	97.3	13.2	79.9	1.296	1.27
9	3.10	2.0	99.3	11.2	91.1	2.062	1.97
Max	10.99	0.7	100.0	8.9	100.0	5.068	4.36

17.4.2 Contact Analysis

Because significant grade contrasts were noted between the different rock types from the assay statistics, contact analysis was performed using the composite data to evaluate grade discontinuities at the lithology contacts. Wherever a contact was crossed with a drill hole, the grade profile was examined on either side of the contact. Contacts were evaluated from the Cambrian to the Upper Sediments, from the Upper Sediments into the Main Volcanics, and from the Main Volcanics into the Lower Sediments.

Between the Cambrian and Upper Sediments the grade contrast is fairly significant. In the vicinity of the contact, the average grade of the Cambrian is 0.30 g/t Au while the Upper Sediments is 0.45 g/t Au (**Figure 17.2**).

Between the Upper Sediments and the Main Volcanics the grade contrast is also fairly significant. The contact between the Main Volcanics and the Lower Sediments is the most significant with the grade in the Main Volcanics being 0.63 g/t Au and the Lower Sediments 0.43 g/t Au.

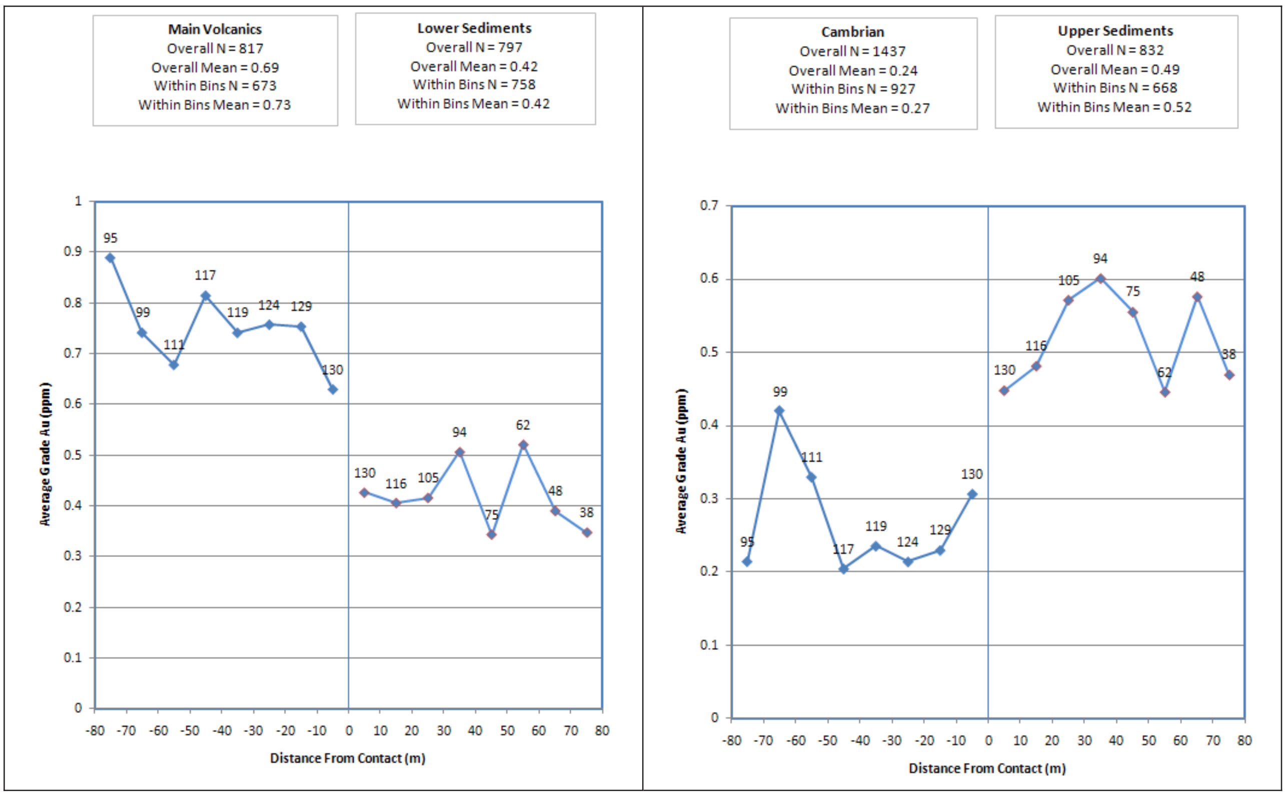


Figure 17.2 Contact plots.

Because of the sharp contrasts in gold grade between the different units, it was decided to treat the boundaries between the different units as hard boundaries. That is, the blocks of a given unit were estimated using only the composite data that fell within the same unit. This is geologically reasonable since many of the contacts are associated with thrust faulting. But it is not known if there has been any post-mineralization movement of these faults. The Main Volcanics are unquestionably better mineralized than the surrounding units. The reason for this is not fully understood. With this, it is not geologically unreasonable to see grade discontinuities at the contacts for this reason either.

The use of hard boundaries will have an impact on the local estimates because the data has been partitioned. Overall, whether hard boundaries or soft boundaries are used or not would have a minimal effect on the global estimate. The issue as to whether hard or soft boundaries are more appropriate should be resolved as more drilling is done and additional information is gathered.

17.5 Spatial Statistics

17.5.1 Gold Indicator Variograms

Indicator variograms were calculated for each of the indicator thresholds within each of the lithologic domains. Variogram models were fitted for each. Because the data was so heavily partitioned the results from the individual domains were generally unsatisfactory. Many of the areas are relatively thin, especially in the Main Volcanics, making it very difficult to infer a model of vertical continuity. For this reason, the use of the partitioned data for variogram calculations was abandoned and all of the data was used to calculate a set of average indicator variograms that were used over all domains. The average indicator variograms that were used for estimation of the gold indicators in all domains are shown in **Table 17.10**.

17.5.2 Oxide Indicator Variograms

The oxidation model was estimated using two oxide indicators, one for oxidized and one for trace (**Table 17.11**). Both the oxidized indicator variogram and the trace indicator variogram dip shallowly to the south-east.

TABLE 17.10

AVERAGE GOLD INDICATOR VARIOGRAMS

Indicator	Sill	Range X	Range Y	Range Z
1	0.42			
	0.19	158	137	131
	0.39	1170	658	140
2	0.51			
	0.19	281	215	127
	0.37	927	890	138
3	0.53			
	0.27	137	132	96
	0.20	860	618	140
4	0.60			
	0.23	133	130	102
	0.17	790	519	129
5	0.69			
	0.19	151	120	104
	0.19	770	444	1134
6	0.73			
	0.17	136	134	51
	0.10	721	286	144
7	0.79			
	0.14	184	148	33
	0.07	626	176	160
8 & 9	0.93			
	0.03	310	125	50
	0.04	471	162	112

TABLE 17.11

OXIDE INDICATOR VARIOGRAMS

Indicator	Sill	Range X	Range Y	Range Z
Oxidized	0.19			
	0.36	125	124	116
	0.45	3900	3000	260
Trace	0.36			
	0.25	403	264	131
	0.39	4900	1880	240

17.5.3 KINT Dike Variograms

A continuous dike indicator was defined using the percentage of Kint dike within each logged interval. The presence and behavior of the dikes north and south of the Lillian Fault are significantly different. Different variograms were fitted for each of these dike domains (**Table 17.12**). The variogram in the north was not rotated. The variogram in the south was rotated with the horizontal plane dipping 20° to the south.

TABLE 17.12

KINT DIKE VARIOGRAMS

Domain	Sill	Range X	Range Y	Range Z
North	0.2			
	0.7	79	100	412
	0.1	120	200	412
South	0.2			
	0.8	500	250	50

17.5.4 Amy Sequence, Lower Sands and Shale Variograms

Continuous indicators were defined using the percentage of Amy Sequence, Lower Sands and Shale within each logged interval (**Table 17.13**). The Amy Sequence material occurs only in the Cambrian, south of the Lillian Fault. The Lower Sands material occurs only in the Lower Sediments, and the Shale occurs throughout the model, largely paralleling the stratigraphy. The units dip steeply north on the north side of the Lillian fault because of overturning, so separate shale variograms were calculated for the areas north and south of the fault.

TABLE 17.13

LOWER SANDS, SHALE & AMY SEQ. VARIOGRAMS

Domain	Sill	Range X	Range Y	Range Z
L. Sand	0.18			
	056	200	143	88
	0.26	1758	597	274
Amy Seq.	0.11			
	0.38	326	118	50
	0.51	852	463	150
N. Shale	0.31			
	0.59	150	57	98
	0.10	150	1500	2060
S. Shale	0.50			
	0.50	340	340	75

The Amy Sequence variogram dips shallowly to the East, while the Lower Sand variogram is essentially horizontal. The North Shale variogram dips at 35º to the North-east, and the South Shale variogram dips at a low angle to the South.

17.6 Resource Model

17.6.1 Model Extents

The resource model was constructed to encompass the drilling data and the defined geological model. The entire project is done using UTM NAD27 Alaska coordinate system. The model extents are shown in **Table 17.14**.

TABLE 17.14

Model Extents

	Minimum (m)	Maximum (m)	Extent (m)	Block Size (m)	No. of Blocks
East	427,500	430,500	3,000	15	200
North	7,264,300	7,266,700	2,400	15	160
Elevation	50	560	510	10	51

The selected block size was chosen because it is envisioned that the deposit will be mined with bulk mining methods that would not warrant smaller blocks but also because the drill hole spacing would not support a smaller block size.

17.6.2 Gold Estimation

The gold contained within each block was estimated using MIK with nine indicator thresholds. The block model was tagged with the geological model using a block majority coding method. The contact analysis indicated that there are significant grade discontinuities at the lithologic boundaries. Hard boundaries were used between each of the units. That is, each unit was estimated using only data that also fell within the same unit. The blocks that fell outside of the defined model were estimated as a separate unit. There was no potentially economic mineralization outside of the geological model and it was not estimated. The gold kriging plan is shown in **Table 17.15** for all units.

TABLE 17.15

GOLD KRIGING PLAN

Minimum No. of Composites	8
Maximum No. of Composites	48
Maximum Composites per Octant	6
Maximum No. of Composites per Hole	4
Block Discretization	4 x 4 x 1
Search Distances (m)	300 (N/S), 150 (E/W), 150 (Vert.)
Search Rotation	Horizontal plane tilted +10° to the North

An octant search was used. The kriging plan forces data to be available from a minimum of two octants and from two separate drill holes for an estimate to be made. Each of the gold indicators was estimated independently.

17.6.3 Oxidation Estimation

Two levels of oxidation were estimated: oxidized and trace oxidation. These levels correspond to the metallurgical testing and were therefore necessary to estimate to allow the application of the metallurgical recoveries to the model. The oxidation level has been visually logged for each sample interval ITH geologists. Two oxidation indicators were used to estimate the oxidation. Historically, oxidation has been logged using ten different descriptors ranging from "complete" to" none". Any interval described as "moderate" or greater was classified as oxidized. Any interval described as anything except "none" was classified as trace or better. The two indicators were tagged on each of the samples as 1 (meeting the criteria) or 0 (not meeting the criteria). Each indicator represents the probability of the sample being oxidized. These indicators were composited into 10m composites with the rest of the data. The two indicators were estimated independently. The kriging plans are shown in **Table 17.16 and Table 17.17**.

TABLE 17.16

OXIDIZED KRIGING PLAN

Minimum No. of Composites	8
Maximum No. of Composites	48
Maximum Composites per Octant	6
Maximum No. of Composites per Hole	4
Block Discretization	4 x 4 x 1
Search Distances (m)	300 (N/S), 150 (E/W), 100 (Vert.)
Search Rotation	None

TABLE 17.17

TRACE OXIDIZATION KRIGING PLAN

Minimum No. of Composites	8
Maximum No. of Composites	48
Maximum Composites per Octant	6
Maximum No. of Composites per Hole	4
Block Discretization	4 x 4 x 1
Search Distances (m)	300 (N/S), 150 (E/W), 100 (Vert.)
Search Rotation	None

The blocks were then coded as fully oxidized (coded as 1) if the probability of being oxidized was greater than 50%. The blocks were coded as trace (coded as 2) oxidized if the probability of trace oxidization was greater than 50% and not already tagged as oxidized. The remaining unoxidized blocks were coded as 3. As would be expected, the fully oxidized material is nearer the surface and consequently mostly in the Cambrian rocks. The trace oxidization is pervasive. Significant unoxidized material is not encountered except in the lower sediments.

17.6.4 KINT Dike Estimation

The Kint dikes are significant metallurgically. It was therefore necessary to estimate them. The dikes are small enough that the drilling information is insufficient to build a deterministic model of the dike locations. Consequently, the dikes were estimated using a probabilistic model. In each block in the model, the probability of encountering dike was treated as the dike proportion within the block.

A single continuous dike indicator was used to estimate the presence of dikes. The presence of dikes was logged for each logged interval. The percentage of dike within the interval was logged, as in many cases the dike represented less than 100% of the interval. The dike indicator was set to be the proportion of dike within the interval. This indicator was then composited into 10m composites along with the rest of the data.

The presence and distribution of dikes is significantly different north and south of the Lillian Fault. The two domains were estimated separately. The kriging plan to estimate the proportion of dike within each block is shown in **Table 17.18** and **Table 17.19**.

TABLE 17.18

KINT DIKE INDICATOR KRIGING PLAN – SOUTHERN DOMAIN

Minimum No. of Composites	8
Maximum No. of Composites	48
Maximum Composites per Octant	6
Maximum No. of Composites per Hole	4
Block Discretization	4 x 4 x 1
Search Distances (m)	500 (N/S), 250 (E/W), 50 (Vert.)
Search Rotation	Horizontal plane tilted +20° to the North

TABLE 17.19

KINT DIKE INDICATOR KRIGING PLAN – NORTHERN DOMAIN

Minimum No. of Composites	8
Maximum No. of Composites	48
Maximum Composites per Octant	6
Maximum No. of Composites per Hole	4
Block Discretization	4 x 4 x 1
Search Distances (m)	120 (N/S), 200 (E/W), 400 (Vert.)
Search Rotation	None

The Kint dikes are important for metallurgical but make up a very small portion of the total resource. The Kint dikes average between 3 and 4% of the tonnage.

17.6.5 Amy Sequence, Lower Sands and Shale Estimation

The Amy Sequence, Lower Sands and Shale units are significant metallurgically. It was therefore necessary to estimate them. The occurrences are small enough that the drilling information is insufficient to build a deterministic model of their locations. Consequently, these were estimated using a probabilistic model. In each block in the model, the probability of encountering these units was treated as the material proportion within the block.

A single continuous indicator was used to estimate the presence of the units. The presence of Amy Sequence, Lower Sands and Shale was logged for each logged interval. The percentage of these units within the interval was logged, as in many cases the lithology represented less than 100% of the interval. The unit indicator was set to be the proportion of lithology within the interval. This indicator was then composited into 10m composites along with the rest of the data. The kriging plan to estimate the proportion of these units within each block is shown in **Table 17.20.** Note that the Amy Sequence occurs only in the Cambrian, and that the Lower sands occur only in the Lower Sediments.

TABLE 17.20

LOWER SANDS, SHALE & AMY SEQ. INDICATOR KRIGING PLAN

Minimum No. of Composites	8
Maximum No. of Composites	48
Maximum Composites per Octant	6
Maximum No. of Composites per Hole	4
Block Discretization	4 x 4 x 1
Search Distances (m) – Lower Sand	Major 300, Int. 150, Minor 100
Search Rotation – Lower Sand	Major 0º → Azimuth 290º
Search Distance (m) – Amy Sequence	Major 300, Int. 150, Minor 100
Search Rotation – Amy Sequence	Major 0º → Azimuth 70º
Search Distance (m) – South Shale	Major 300, Int. 300, Minor 75
Search Rotation – South Shale	Major -10º → Azimuth 191º
Search Distance (m) – North Shale LS*	Major 300, Int. 230, Minor 50
Search Rotation – North Shale LS*	Major -75º → Azimuth 25º
Search Distance (m) – North Shale	Major 300, Int. 230, Minor 50
Search Rotation – North Shale	Major -56º → Azimuth 58º

North Shale LS* occurs north of the Lillian Fault but below the North Volcanics

17.7 Model Validation

Various forms of model validation were undertaken and are shown below. In all cases, the model appears to be unbiased and fairly represent the drilling data. The composite data was declustered by estimating a nearest-neighbor value into each block.

17.7.1 Global Bias Check

The global average of the declustered composite values is 0.391 g/t Au and the corresponding average block value is 0.378. The estimated block values are 3% lower than the composite values. This is reasonable and within the expectations of the model.

17.7.2 Visual Validation

The model was visually compared to the composite gold data in both N-S and E-W sections. The estimates were checked to see that they appeared to be consistent with the data and that they were geologically reasonable. In all cases everything appeared reasonable.

17.8 Post-processing of MIK Model

The post-processing of the indicator kriging was done with the POSTIK module in WinGSLIB. It is necessary to provide a maximum grade of the distribution. This grade can be calculated as:

$$Z_{max} = Z_{cn} + [2^{1/2} / (2^{1/2} - 1)](med_n - Z_{cn})$$

Where Z_{cn} is the uppermost indicator threshold, and med_n is the median of values > Z_{cn}

From the data in **Table 17.10**, the maximum grade used in the post-processing was calculated to be 7.4ppm.

17.8.1 Change of Support

The multiple indicator kriging produces an estimate of the distribution of grade within a block rather than just a single average grade of a block. The distribution produced is the distribution of composite sized units within the block not minable units. It is therefore necessary to correct the distribution so that the distribution represents selective mining units (SMU's) not composite sized units. This correction is called a change of support correction. Since the average grade of the block is the same whether mined in one scoop or mined by a core drill, the correction does not change the average grade of the block only reduces the variance of the distribution.

The variance reduction factor is the ratio of the variance of an SMU within a block to the variance of a composite within a block. This is calculated using average variogram values. The variance of the SMU within the block is the variance of a composite within a block minus the variance of a composite within an SMU. Since the estimated blocks are small relative to the data spacing the effective block size was taken to be 37.5m by 37.5m (approximately ½ the drill spacing).

The method used for the change of support was an indirect lognormal correction. This correction uses the ratio of standard deviations rather than the ratio of variances. This is just the square root of the ratio of variances.

The mining SMU was assumed to be 5m by 5m selectivity. This is reasonable for the envisioned size of the operation. If the envisioned size of the operation were to grow significantly, the SMU size should be increased.

The following factors were derived using the variogram model.

$$\bar{\gamma}(\cdot, b_{375,375}) = 0.473$$
$$\bar{\gamma}(\cdot, b_{5,5}) = 0.427$$

$$RF(5,5) = \sqrt{\frac{\bar{\gamma}(\cdot, b_{375,375}) - \bar{\gamma}(\cdot, b_{5,5})}{\bar{\gamma}(\cdot, b_{375,375})}}$$
$$= 0.31$$

This correction is applied on a block-by-block basis with a global reduction target of 0.31. This is done on a trial and error basis to find the block reduction factor that will achieve the target global variance reduction of 0.31. A reduction factor of 0.21 was used by block.

17.8.2 Calculation of Metallurgical Recoveries

The metallurgical recoveries used are a function of the rock type, the oxidation state, and the estimated percentages of Kint, Amy Sequence, Shale and Lower Sands – these percentages act as modifiers on the base recoveries. The metallurgical recoveries for the heap leach processing method were calculated and retained within the block model. The recovery within the dikes and shale is significantly lower than the surrounding rocks. The average block recovery was calculated as a weighted average between the modifier and non-modifier material using the estimated modifier percentages as the weighting factors.

17.9 Resource Classification

The resource was broken down into two categories: Indicated and Inferred. The estimation variance from the estimation of the second indicator (median indicator) was used to determine the classification. Along with the estimation of variance, the number of composites used, number of drill holes used and the distance to the nearest composite was saved for each block estimated. The estimation variance provides a good measure of the confidence in the estimate. The estimation variance will remain relatively low when data is near and evenly spaced around the block being estimated. When the estimate starts extrapolating away from data, the estimation variance will rise rapidly. An examination of probability plots of block kriging variance to identify natural breaks in the distribution, along with visual inspection of the model relative to the composite data were used to determine the acceptable estimation variance thresholds**. Blocks estimated with an estimation variance less than 0.20 should be considered Indicated and blocks with an estimation variance less than 0.40 should be considered Inferred. Blocks with an estimation variance greater than 0.40 were considered to be too unreliable for further consideration.**

17.10 Preliminary Economic Assessment

17.10.1 Metallurgy

Processing by heap leach only was considered in the initial property evaluation. It should be noted that continuing metallurgical testing indicates that milling should be possible - with a mill in place, synergy between the mill and heap leach would drop the heap leach costs significantly relative to a heap leach only operation. Process costs and process recoveries were developed for the heap leach option for each of the different rock types (discussed in **Section 16**).

17.10.2 Mining

No detailed engineering analysis of mining costs was performed. The mining cost used was derived from operating costs of a similarly sized neighboring mine and similar-sized mines in Colorado and Nevada. A mining cost of $1.80 per metric ton was used throughout.

ITH commissioned SRK to estimate the capital costs associated with several different development scenarios for Livengood. The direct mining capital was estimated for operations ranging in size from 50ktpd to 150ktpd assuming a strip ratio of 1:1 (**Table 17.21**). The direct capital costs for the 100ktpd operations were extrapolated from the two capital estimates in Table **17.21** and adjusted for the actual strip ratio of 0.78:1. A contingency of 15% was added resulting in a final direct mining capital cost of USD$104.7M.

TABLE 17.21
Direct Mining Capital Estimates assuming a 1:1 Strip Ratio (US$000's)

Capital Item	150ktpd	50ktpd
Cable Shovels	$46,488	$16,776
Front End Loader	$8,567	$4,592
Haul Trucks	$50,050	$19,699
Rotary Drills	$18,824	$17,376
Dozers	$5,702	$4,615
Graders	$3,276	$1,602
Water Trucks	$1,906	$1,454
Fuel / Lube Trucks	$360	$332
Service Trucks	$436	$402
Tire Trucks	$411	$379
Bulk Loader	$293	$180
Light Plants	$854	$526
Pumps	$1,150	$455
Pickups	$953	$643
Crane	$358	$330
Forklift	$130	$120
Cold Weather Heaters	$912	$360
Direct Min	**$140,668**	**$69,842**

The initial capital estimate for the heap leach included the construction of only three years capacity for the heap leach pad. Each additional three years capacity was estimated to cost US$80M or

USD27M per year. Because the overall mine life was 13 years this capital was added in US$40M allocations in years 2, 3, 5, 6, 8 and 9 with US$27M added in Year 11. An additional USD$30M was added in USD$10M allocations in years 8, 9, and 10 for general replacement of equipment due to wear and tear. The total sustaining capital was USD$297M.

No geotechnical data is currently available for the project. A pit slope angle of 45° was assumed throughout. This is a slope angle commonly achieved in mining and believed to be a reasonable assumption.

17.10.3 General Overhead

A general overhead cost of $0.60 per metric ton processed was assumed. Within the context of this study, this is believed to be reasonable. No consideration has been given to the costs of marketing. Taxes and royalty charges were excluded from this preliminary analysis of the project. Royalty rates vary from 0-5% across the project and average approximately 2.5%..

17.11 Whittle Pit Optimization and Analysis

Pit shells were derived for a range of gold prices using the Whittle® optimization and analysis package. Processing options were restricted to heap leach only, using Indicated + Inferred resources.

17.11.1 Heap Leach Option

This initial stage, heap leach PEA, utilizes the October 2009 resource model which includes all assays completed through September 25, 2009 (308 diamond and reverse circulation holes for a total of 83,200 metres). The economic analysis of this option involved a Whittle pit optimization study at various gold prices using a 100,000 tonne per day production scenarios. A base case production scenario was selected using the USD $700 Whittle pit which was then optimized resulting in the final strip ratio and cash cost shown in **Table 17.22**. A life of mine, mining and production plan was combined with initial and sustaining capital estimates and evaluated at USD $850, $950 and $1,050 gold prices using a 5% and 7.5% discount rate to assess the projects sensitivity to gold prices **(Table 17.23)**. The results of this study highlights the potential value of this initial stage of the Livengood project as well as its high leverage to gold price.

TABLE 17.22
Livengood Project - Heap Leach PEA – Base Case Summary*

In-pit resource - Indicated :	**308Mt @ 0.68 g/t gold for contained ounces**
In-pit resource - Inferred	**132Mt @ 0. 71g/t gold for contained ounces**
Over all strip ratio of :	**1 to 0.78 (ore-waste)**
Annual Production:	**459,033 ounces gold, total of 5,783,813 recoverable ounces**
Average gold recovery:	**60%**
Mining rate:	**100,000 ore tonnes per day, 178,000 total tonnes per day**
Mining cost per tonne:	**$1.80**
Processing cost per tonne:	**$3.80**
G&A cost per tonne:	**$0.60**
Cost per ounce:	**$533**
Initial capital costs:	**$665M, life of project sustaining capital $297M**
Contingency:	**20%**

*All values in USD and based on $700 Whittle optimized pit shell

TABLE 17.23
Base Case Gold Price Sensitivity Analysis
(Values in USD)

Gold Price (USD)	**NPV$^{(5\%)}$ (M)**	**NPV$^{(7.5\%)}$ (M)**	**IRR (%)**
$850	$440	$293	14.6%
$950	$867	$660	22.7%
$1,050	$1,291	$1,029	30.3%

17.12 Discussion

The initial PEA for the Livengood deposit has demonstrated that the project has strong potential based on a heap leach only option investigated. It is important to note that this PEA covers only the heap leach, oxide component of the October 13, 2009 resource with approximately 40% of the deposit (un-oxidized mineralization) excluded pending the possible addition of a milling circuit. Milling metallurgical test work currently in progress suggests that high gold recoveries can be obtained from all ore types utilizing an initial gravity concentration circuit followed by standard CIL processing of the tails. On completion and validation of these results it is recommended that the current PEA be updated to include any additional drillhole data, and the investigation of heap only, heap plus mill and mill only options for the project.

18.0 Other Relevant Data and Information

No additional information or explanation is known by the authors to be necessary to make the technical report understandable and not misleading.

19.0 Interpretation and Conclusions

The Livengood property is centered on Money Knob and adjacent ridges and is an area considered by many for a long time to be the lode source for gold in the Livengood placer deposits which have produced in excess of 500,000 ounces of gold. An anomalous areagold in soil samples occurring in a northeast trend cover an area of approximately 6 x 2 km with a principal concentration of surface anomalies in a smaller area measuring approximately 1.6 x 0.8 km. Drilling by past companies, AGA, and ITH identified wide intervals (>100 m @ ≥ 1.0 g/t Au) of gold mineralization with local higher grade narrow intervals beneath the soil anomaly and in rocks beneath thrust surfaces which are not expressed geochemically at the surface. The presence of mineralization over broad areas beneath thrust faults and the ever expanding area of drill hole intercepts is encouraging and suggests that there is still further discovery potential at Livengood.

The style of mineralization shows some similarities with several types of gold deposits including orogenic, sediment-hosted disseminated (SHD or Carlin type), and Intrusion-Related-Gold Systems (IRGS) of the Tintina Gold Belt. However, the geochemical and metallogenic associations of As, Sb, ± Bi, and lack of some features typical of SHD's indicates that Livengood is most comparable to IRGS type deposits and is typical of other such deposits within the host Tintina Gold Belt.

Gold mineralization at Livengood is hosted in a thrust interleaved sequence of Late Proterozoic to Palaeozoic ophiolitic rocks thrust emplaced over a Devonian sequence of sedimentary and volcanic rocks. Mineralization is related to a ~90 million year old set of monzonite to diorite dikes that intrude the thrust stack along thrust faults. Mineralization is hosted primarily by Devonian volcanics and Cretaceous dikes, but occurs in all rock types and consists of gold associated with arsenopyrite and to a lesser extent pyrite. Other associated minerals include stibnite, marcasite, pyrrhotite, and minor to trace amounts of chalcopyrite and sphalerite.

Four stages of alteration are currently recognized. These include biotite, albite, sericite, and carbonate. These stages are interpreted to reflect alteration of host rocks by a fluid with decreasing temperature and evolving chemistry over time.

Overall, mineralization and alteration appear to be controlled by the thrust fault architecture and possibly by later normal faults.

The original surface geochemical anomaly in soil that attracted exploration in this location in the first place, probably reflects only a portion of the mineralization present. Mineralization has been shown to continue down-dip along and/or beneath thrust surfaces and therefore be blind at the surface. This point along with the fact that the area drilled currently represents only a portion of the original surface geochemical anomaly suggests that the identification of more mineralization over a broader area is likely.

Drill results through May 2009 have been used to revise previous resource estimates for the Money Knob area. The current resource estimate has significantly increased the tonnage and total number of ounces contained in estimated Indicated and Inferred categories of resource. The amount of gold in the resource varies significantly according to the choice of cutoff grade. A range of tonnes and grade with corresponding contained ounces of gold are presented in **Table 17.1**. Application of multiple indicator kriging methods have provided new insights into the character of mineralization and offered an improved, more robust block model for the resource estimation. Comparison of block model with geologic sections interpreted by ITH geologists (**Figures 7.8-7.12**) reveals good correspondence (**Figures 19.1 to 19.3**). These sections also show the potential of mineralized material to continue to depth, particularly down-dip.

It is concluded that a substantial gold resource has been identified and that further drilling is appropriate for continued evaluation and likely expansion of this resource. ITH has now advanced the Livengood project to the point that a Preliminary Economic Assessment has been completed. Work in progress at this time is aimed at advancing this assessment. Toward this goal, ITH is currently working on metallurgical and processing test work as described in Section 16. This work should allow assessment of the best material processing and gold recovery techniques, operating and capital costs, possible mine size and scheduling, and initial cost benefit analysis with estimated NPV, ROI, for different mine and processing scenarios. Toward this end, the following activities should be considered for the 2010 exploration program:

1. Continue step out drilling to identify the extent of mineralization, particularly:
   a. to the northeast of the Sunshine Zone
   b. immediately northwest of the known Sunshine Zone,
   c. to the southwest along the trend of the surface geochemical anomaly

2. Continue systematic drilling on lines 75m apart and at 75m spacings along those lines to:
   a. Improve continuity of mineralization over a broader area, particularly in areas that are now categorized as Inferred Resource, and
   b. Improve understanding of the structural relations and architecture that hosts the deposit.

3. Drill several holes at closer spacing between lines and between holes, particularly where current drill hole spacing only allows inferred categories of blocks in the block model.

4. Drill holes in select key locations between current north-south lines to:
   a. Validate lateral correlation of mineralization between north-south lines of holes, and
   b. Raise confidence in strike continuity of mineralization.

5. Utilize 3D modeling software to model the structural architecture. This should help understand the mineralization better and offer predictive capabilities for exploration.

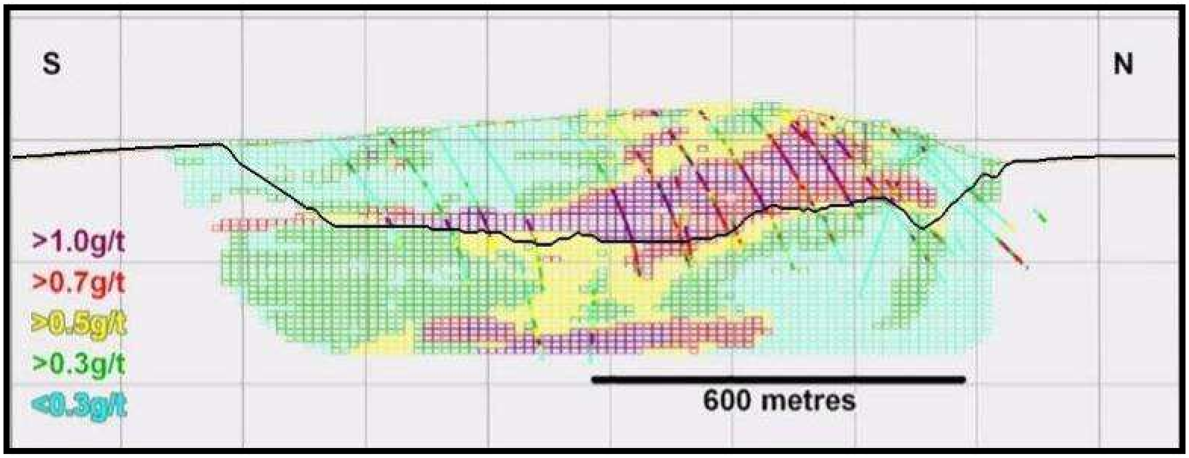


Figure 19.1. Block model for section 428925 E.

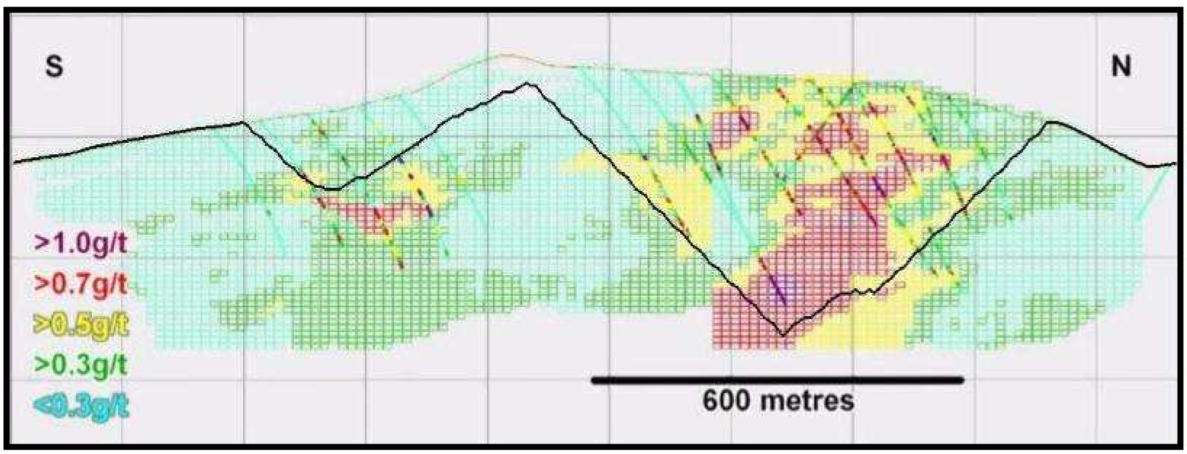


Figure 19.2. Block model for section 429525 E.

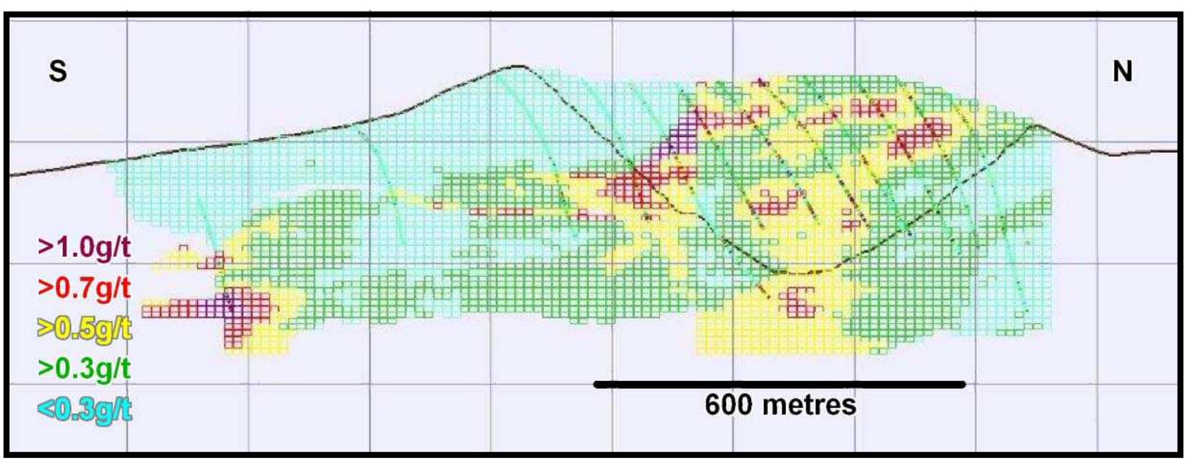


Figure 19.3. Block model for section 429675 E.

6. Continue and advance metallurgical, ore characterization, and mineral processing studies. This should include:
    a. Expanded use of petrographic evaluation of rock types, alteration, and metallic mineralogy;
    b. Use of SEM studies to evaluate in detail the trace element content of metallic minerals in support of metallurgical test work; and
    c. Use SEM studies to better characterize gold mineralization, its exact mineral association, and relationship to gangue.

7. Continue and expand environmental base line studies.

8. Assess geotechnical characteristics of the mineralized zone.

9. Complete the combined mill/heap leach preliminary economic analysis that is currently in progress. This should evaluate the basic economic, logistic, and processing factors for a mining operation at Livengood.

20.0 Recommendations

20.1 Recommended Exploration Program

Exploration of the Livengood project should continue with the aim of advancing the project toward a prefeasibility status. Activities that will help advance the project in this direction include those listed in the previous section.

ITH plans to drill 50,000 m in 2010 to accomplish this goal. The proposed program is an appropriate amount of drilling for the needs of the project and the time available in the field season.

20.2 Budget for 2010

ITH has proposed expenditure of approximately $11 million dollars in 2010 for further evaluation of the Livengood project (**Table 20.1**). This budget will be allocated primarily to drilling and geological analysis of the deposit. The budget is appropriate for the amount of drilling planned and feasible within the time allocated. The authors recommend implementation of this program in order to accomplish ITH's goal of advancing the Livengood project.

TABLE 20.1
2010 Exploration Budget

Expenditure	2010 $ M	Comments
Land	0.4	Claim and lease fees
Geological and Contract Services	2.2	Contract/consulting fees
Drilling	4.6	Drilling, supplies, surveying, preparation, hole abandonment
Geochemistry	1.2	Rock, soil, drill core and cuttings, prep and assay
Environmental and Metallurgy Studies	1.5	
Admin and Operations	0.8	Office, salaries, travel, reporting, permitting
TOTAL	**10.7**	

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22.0 Date and Signature Page

The effective date of this technical report, entitled "October, 2009 Summary Report on the Livengood Project, Tolovana District, Alaska" is October 31, 2009.

Dated: November 27, 2009

Signed:

(signed) *Paul Klipfel* [Sealed: CPG#10821]
Dr. Paul Klipfel, Ph.D, CPG#10821

(signed) *Timothy J. Carew* [Sealed]
Timothy J. Carew, P.Geo.

(signed) *William Pennstrom, Jr.* [Sealed]
William Pennstrom, Jr. M.A.

23.0 Certificates of Authors

CERTIFICATE OF PAUL D. KLIPFEL, PH.D.

I, Paul D. Klipfel, Ph.D., do hereby certify that:

1. I am President of :
   Mineral Resource Services, Inc.
   4889 Sierra Pine Dr.
   Reno, NV 89519

2. I have graduated from the following Universities with degrees as follows:

a.	San Francisco State University,	B.A. geology	1978
b.	University of Idaho,	M.S. economic geology	1981
c.	Colorado School of Mines	M.S. mineral economics	1988
d.	Colorado School of Mines	Ph.D. economic geology	1992

3. I am a member in good standing of the following professional associations:
   a. Society of Mining Engineers
   b. Society of Economic Geologists
   c. Geological Society of America
   d. Society for Applied Geology
   e. American Institute of Professional Geologists
   f. Sigma Xi

4. I have worked as a mineral exploration geologist for 30+ years since my graduation from San Francisco State University.

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 professional associations 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 all sections of the technical report titled "**October 2009 Summary Report on the Livengood Project, Tolovana District, Alaska"** and dated October 31, 2009 (the "Technical Report") relating to the Livengood property except sections 16 and 17. I have visited the Livengood property on six occasions, the most recent being October 5-8, 2009.

7. Prior to being retained by ITH in 2006, I have not had prior involvement with the property that is the subject of the Technical Report.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in section 1.4 of NI 43-101.

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

Dated this 27th day of November, 2009

(signed) *Paul Klipfel* [Sealed: AIPG#10821]
Signature of Qualified Person

Paul D. Klipfel, Ph.D, CPG[AIPG]
Print name of Qualified Person

CERTIFICATE OF TIMOTHY J. CAREW

I, Timothy J. Carew, P. Geo. do hereby certify that:

1. I am the Principal of :
   Reserva International LLC
   P.O. Box 19848
   Reno, NV 89511 USA

2. I have graduated from the following Universities with degrees as follows:
   a. University of Rhodesia, B.Sc. Geology 1973
   b. University of Rhodesia, B.Sc. (Hons) Geology 1976
   c. University of London (RSM) M.Sc. Mineral Prod. Management 1982

3. I am a member in good standing of the following professional associations:
   a. Association of Professional Engineers and Geoscientists of British Columbia
   b. Institute of Mining, Metallurgy and Materials
   c. Canadian Institute of Mining and Metallurgy
   d. Society of Mining Engineers

4. I have worked in mining geology and engineering for over 35 years since my graduation from the University of Rhodesia.

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 professional associations 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 section 17 of the technical report titled “**October 2009 Summary Report on the Livengood Project, Tolovana District, Alaska”** and dated October 31, 2009 (the “Technical Report”) relating to the Livengood property. I have visited the Livengood property on two occasions for a total of nine days, the most recent being from October 28th, 2009.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I am not aware of any material fact or material change with respect to the subject matter of Section 17 of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in section 1.4 of NI 43-101.

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

Dated this 27th day of November, 2009

(signed) *Timothy J. Carew*
Signature of Qualified Person

Timothy J. Carew P.E.
Print name of Qualified Person

CERTIFICATE OF WILLIAM PENNSTROM JR.

I, William J. Pennstrom Jr., do hereby certify that:

1. I am self employed as a Consulting Process Engineer and President of:

   Pennstrom Consulting Inc.
   2728 Southshire Rd.
   Highlands Ranch, CO 80126

2. I graduated in 1983 with a Bachelors of Science degree in Metallurgical Engineering from the University of Missouri - Rolla, Rolla, Missouri and in 2001 with a Master of Arts degree in Management from Webster University, St. Louis, Missouri.

3. I am a Founding Registered Member of the Society for Mining, Metallurgy, and Exploration (SME) and am a recognized Qualified Professional (QP) Member with expertise in Metallurgy of the Mining and Metallurgical Society of America (MMSA).

4. I have worked in the Mineral Processing Industry for a total of 29 years since before, during, and after my attending the University of Missouri.

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 purpose of NI 43-101.

6. I am responsible for the preparation of section 16 of the technical report titled "**October 2009 Summary Report on the Livengood Project, Tolovana District, Alaska"** and dated October 31, 2009 (the "Technical Report") relating to the Livengood property. I have visited the Livengood Project site for two days during May of 2009.

7. Prior to being retained by ITH in May, 2009, I have not had prior involvement with the property that is the subject of the Technical Report.

8. I am not aware of any material fact or material change with respect to the subject matter of Section 16 of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests per Section 1.5 of NI 43-101.

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

Dated the 27th day of November, 2009.

(signed) *William Pennstrom Jr.*
Signature of Qualified Person

William Pennstrom Jr.
Print name of Qualified Person

25.0 Appendices

Appendix 1: Claim/Property Information

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
Alaska State Lease				
Alaska Mental Health Land Trust	9400248	AMHLT - ML	1-Jul-2004	F008N005W
Federal Patented Claims				
Griffin heirs	MS 1990, Patent 1041576	Mastodon	18-Jan-2007	F008N005W
Griffin heirs	MS 1990, Patent 1041576	Piedmont	18-Jan-2007	F008N005W
Griffin heirs	MS 1990, Patent 1041576	Yukon	18-Jan-2007	F008N005W
Federal Unpatented Claims				
Richard Hudson	55469	ANNE	21-Apr-2003	F008N005W24
Richard Hudson	55466	BLACK ROCK	21-Apr-2003	F008N005W24
Richard Hudson	55471	BRIDGET	21-Apr-2003	F008N005W24
Richard Hudson	55453	DOROTHEA	21-Apr-2003	F008N005W23
Richard Hudson	55470	EILEEN	21-Apr-2003	F008N005W24
Richard Hudson	55455	FOSTER	21-Apr-2003	F008N005W24
Richard Hudson	55454	LENORA	21-Apr-2003	F008N005W23
Richard Hudson	55459	NICKIE	21-Apr-2003	F008N005W24
Richard Hudson	55464	OLD SMOKY	21-Apr-2003	F008N005W23
Richard Hudson	55468	PATRICIA	21-Apr-2003	F008N005W13
Richard Hudson	55460	PATRICK	21-Apr-2003	F008N005W23
Richard Hudson	55458	SAUNDERS	21-Apr-2003	F008N005W23
Richard Hudson	55452	SHARON	21-Apr-2003	F008N005W23
Richard Geraghty	55462	SUNSHINE #1	21-Apr-2003	F008N005W23
Richard Geraghty	55463	SUNSHINE #2	21-Apr-2003	F008N005W23
Richard Hudson	55467	TRAPLINE	21-Apr-2003	F008N005W24
Richard Hudson	55457	TWERPIT	21-Apr-2003	F008N005W24
Richard Hudson	55456	VANCE	21-Apr-2003	F008N005W24
Richard Hudson	55461	WHITE ROCK	21-Apr-2003	F008N005W23
Richard Hudson	55465	WITTROCK	21-Apr-2003	F008N005W23
Ronald Tucker	37580	Lillian No. 1	30-Sep-1968	F008N005E22
Ronald Tucker	37581	Satellite	30-Sep-1968	F008N005E22
Ronald Tucker	37582	Nickel Bench R.L.*	30-Jun-1972	F008N005E22 & 15
Ronald Tucker	37583	The Nickel*	12-Aug-1965	F008N005E22
Ronald Tucker	37584	Overlooked*	6-Sep-1975	F008N005E22
Ronald Tucker	37585	The Lad*	12-Aug-1965	F008N005E22
State Claims				
Karl Hanneman and Bergelin Family Trust	330936	LUCKY 55	14-May-1981	F009N004W33
Karl Hanneman and Bergelin Family Trust	330937	LUCKY 56	14-May-1981	F009N004W33
Karl Hanneman and Bergelin Family Trust	330938	LUCKY 64	13-May-1981	F009N004W32 F009N004W33
Karl Hanneman and Bergelin Family Trust	330939	LUCKY 65	14-May-1981	F009N004W33

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
Karl Hanneman and Bergelin Family Trust	330940	LUCKY 66	14-May-1981	F009N004W33
Karl Hanneman and Bergelin Family Trust	330941	LUCKY 72	12-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330942	LUCKY 73	13-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330943	LUCKY 74	13-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330944	LUCKY 75	14-May-1981	F008N004W04
Karl Hanneman and Bergelin Family Trust	330945	LUCKY 76	14-May-1981	F008N004W04
Karl Hanneman and Bergelin Family Trust	330946	LUCKY 82	12-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330947	LUCKY 83	13-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330948	LUCKY 84	13-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330949	LUCKY 85	14-May-1981	F008N004W04
Karl Hanneman and Bergelin Family Trust	330950	LUCKY 86	14-May-1981	F008N004W04
Karl Hanneman and Bergelin Family Trust	330951	LUCKY 91	12-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330952	LUCKY 92	12-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330953	LUCKY 93	13-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330954	LUCKY 94	13-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330955	LUCKY 95	14-May-1981	F008N004W04
Karl Hanneman and Bergelin Family Trust	330956	LUCKY 96	14-May-1981	F008N004W04
Karl Hanneman and Bergelin Family Trust	330957	LUCKY 101	12-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330958	LUCKY 102	12-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330959	LUCKY 103	12-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330960	LUCKY 104	12-May-1981	F008N004W05
Karl Hanneman and Bergelin Family Trust	330961	LUCKY 105	12-May-1981	F008N004W04
Karl Hanneman and Bergelin Family Trust	330962	LUCKY 106	12-May-1981	F008N004W04
Karl Hanneman and Bergelin Family Trust	330963	LUCKY 202	13-May-1981	F008N004W08
Karl Hanneman and Bergelin Family Trust	330964	LUCKY 203	13-May-1981	F008N004W08
Karl Hanneman and Bergelin Family Trust	330965	LUCKY 204	15-May-1981	F008N004W08
Karl Hanneman and Bergelin Family Trust	330966	LUCKY 205	13-May-1981	F008N004W09
Karl Hanneman and Bergelin Family Trust	330967	LUCKY 206	14-May-1981	F008N004W09
Karl Hanneman and Bergelin Family Trust	330968	LUCKY 207	14-May-1981	F008N004W09
Karl Hanneman and Bergelin Family Trust	330969	LUCKY 208	14-May-1981	F008N004W09
Karl Hanneman and Bergelin Family Trust	330970	LUCKY 302	13-May-1981	F008N004W08
Karl Hanneman and Bergelin Family Trust	330971	LUCKY 303	13-May-1981	F008N004W08
Karl Hanneman and Bergelin Family Trust	330972	LUCKY 304	15-May-1981	F008N004W08
Karl Hanneman and Bergelin Family Trust	330973	LUCKY 305	13-May-1981	F008N004W09
Karl Hanneman and Bergelin Family Trust	330974	LUCKY 306	14-May-1981	F008N004W09
Karl Hanneman and Bergelin Family Trust	330975	LUCKY 307	14-May-1981	F008N004W09
Karl Hanneman and Bergelin Family Trust	330976	LUCKY 308	14-May-1981	F008N004W09
Karl Hanneman and Bergelin Family Trust	330977	LUCKY 404	15-May-1981	F008N004W08
Karl Hanneman and Bergelin Family Trust	330978	LUCKY 405	13-May-1981	F008N004W09
Karl Hanneman and Bergelin Family Trust	330979	LUCKY 406	14-May-1981	F008N004W09
Karl Hanneman and Bergelin Family Trust	338477	LUCKY 198	17-Sep-1981	F008N004W07
Karl Hanneman and Bergelin Family Trust	338478	LUCKY 199	17-Sep-1981	F008N004W07
Karl Hanneman and Bergelin Family Trust	338479	LUCKY 295	17-Sep-1981	F008N005W12
Karl Hanneman and Bergelin Family Trust	338480	LUCKY 296	17-Sep-1981	F008N005W12
Karl Hanneman and Bergelin Family Trust	338481	LUCKY 297	17-Sep-1981	F008N004W07
Karl Hanneman and Bergelin Family Trust	338482	LUCKY 298	17-Sep-1981	F008N004W07

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
Karl Hanneman and Bergelin Family Trust	338483	LUCKY 299	17-Sep-1981	F008N004W07
Karl Hanneman and Bergelin Family Trust	338484	LUCKY 392	21-Sep-1981	F008N005W11
Karl Hanneman and Bergelin Family Trust	338485	LUCKY 395	18-Sep-1981	F008N005W12
Karl Hanneman and Bergelin Family Trust	338486	LUCKY 396	18-Sep-1981	F008N005W12
Karl Hanneman and Bergelin Family Trust	338487	LUCKY 397	18-Sep-1981	F008N004W07
Karl Hanneman and Bergelin Family Trust	338488	LUCKY 398	18-Sep-1981	F008N004W07
Karl Hanneman and Bergelin Family Trust	338489	LUCKY 399	17-Sep-1981	F008N004W07
Karl Hanneman and Bergelin Family Trust	338490	LUCKY 400	23-Sep-1981	F008N004W07 F008N004W08
Karl Hanneman and Bergelin Family Trust	338491	LUCKY 491	21-Sep-1981	F008N005W11
Karl Hanneman and Bergelin Family Trust	338492	LUCKY 492	21-Sep-1981	F008N005W11
Karl Hanneman and Bergelin Family Trust	338493	LUCKY 493	21-Sep-1981	F008N005W12
Karl Hanneman and Bergelin Family Trust	338494	LUCKY 494	21-Sep-1981	F008N005W12
Karl Hanneman and Bergelin Family Trust	338495	LUCKY 495	18-Sep-1981	F008N005W12
Karl Hanneman and Bergelin Family Trust	338496	LUCKY 496	18-Sep-1981	F008N005W12
Karl Hanneman and Bergelin Family Trust	338497	LUCKY 497	18-Sep-1981	F008N004W07
Karl Hanneman and Bergelin Family Trust	338498	LUCKY 498	18-Sep-1981	F008N004W07
Karl Hanneman and Bergelin Family Trust	338499	LUCKY 499	17-Sep-1981	F008N004W07
Karl Hanneman and Bergelin Family Trust	338500	LUCKY 500	23-Sep-1981	F008N004W07 F008N004W08
Karl Hanneman and Bergelin Family Trust	338501	LUCKY 504	10-Sep-1981	F008N004W08
Karl Hanneman and Bergelin Family Trust	338502	LUCKY 505	10-Sep-1981	F008N004W09
Karl Hanneman and Bergelin Family Trust	338503	LUCKY 589	21-Sep-1981	F008N005W14
Karl Hanneman and Bergelin Family Trust	338504	LUCKY 590	21-Sep-1981	F008N005W14
Karl Hanneman and Bergelin Family Trust	338505	LUCKY 591	21-Sep-1981	F008N005W14
Karl Hanneman and Bergelin Family Trust	338506	LUCKY 592	21-Sep-1981	F008N005W14
Karl Hanneman and Bergelin Family Trust	338507	LUCKY 593	21-Sep-1981	F008N005W13
Karl Hanneman and Bergelin Family Trust	338508	LUCKY 594	21-Sep-1981	F008N005W13
Karl Hanneman and Bergelin Family Trust	338509	LUCKY 595	18-Sep-1981	F008N005W13
Karl Hanneman and Bergelin Family Trust	338510	LUCKY 596	18-Sep-1981	F008N005W13
Karl Hanneman and Bergelin Family Trust	338511	LUCKY 597	18-Sep-1981	F008N004W18
Karl Hanneman and Bergelin Family Trust	338512	LUCKY 598	18-Sep-1981	F008N004W18
Karl Hanneman and Bergelin Family Trust	338513	LUCKY 599	17-Sep-1981	F008N004W18
Karl Hanneman and Bergelin Family Trust	338514	LUCKY 689	22-Sep-1981	F008N005W14
Karl Hanneman and Bergelin Family Trust	338515	LUCKY 690	22-Sep-1981	F008N005W14
Karl Hanneman and Bergelin Family Trust	338516	LUCKY 691	22-Sep-1981	F008N005W14
Karl Hanneman and Bergelin Family Trust	338517	LUCKY 692	22-Sep-1981	F008N005W14
Karl Hanneman and Bergelin Family Trust	338518	LUCKY 693	22-Sep-1981	F008N005W13
Karl Hanneman and Bergelin Family Trust	338519	LUCKY 694	22-Sep-1981	F008N005W13
Karl Hanneman and Bergelin Family Trust	338520	LUCKY 697	18-Sep-1981	F008N004W18
Karl Hanneman and Bergelin Family Trust	338521	LUCKY 698	18-Sep-1981	F008N004W18
Karl Hanneman and Bergelin Family Trust	338522	LUCKY 699	17-Sep-1981	F008N004W18
Karl Hanneman and Bergelin Family Trust	347943	LC 407	5-Jun-1982	F008N004W18
Karl Hanneman and Bergelin Family Trust	347945	LC 502	5-Jun-1982	F008N004W08
Karl Hanneman and Bergelin Family Trust	347946	LC 503	5-Jun-1982	F008N004W08
Karl Hanneman and Bergelin Family Trust	347947	LC 506	7-Jun-1982	F008N004W09
Karl Hanneman and Bergelin Family Trust	347948	LC 507	7-Jun-1982	F008N004W09

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
Karl Hanneman and Bergelin Family Trust	347949	LC 600	5-Jun-1982	F008N004W17 F008N004W18
Karl Hanneman and Bergelin Family Trust	347950	LC 601	5-Jun-1982	F008N004W17
Karl Hanneman and Bergelin Family Trust	347951	LC 602	5-Jun-1982	F008N004W17
Karl Hanneman and Bergelin Family Trust	347952	LC 603	5-Jun-1982	F008N004W17
Karl Hanneman and Bergelin Family Trust	347953	LC 604	6-Jun-1982	F008N004W17
Karl Hanneman and Bergelin Family Trust	347954	LC 605	6-Jun-1982	F008N004W16
Karl Hanneman and Bergelin Family Trust	347955	LC 695	10-Jun-1982	F008N005W13
Karl Hanneman and Bergelin Family Trust	347956	LC 696	10-Jun-1982	F008N005W13
Karl Hanneman and Bergelin Family Trust	347957	LC 700	6-Jun-1982	F008N004W17 F008N004W18
Karl Hanneman and Bergelin Family Trust	347958	LC 701	6-Jun-1982	F008N004W17
Karl Hanneman and Bergelin Family Trust	347959	LC 702	6-Jun-1982	F008N004W17
Karl Hanneman and Bergelin Family Trust	347960	LC 703	6-Jun-1982	F008N004W17
Karl Hanneman and Bergelin Family Trust	347961	LC 704	6-Jun-1982	F008N004W17
Karl Hanneman and Bergelin Family Trust	347962	LC 790	12-Jun-1982	F008N005W14
Karl Hanneman and Bergelin Family Trust	347963	LC 791	12-Jun-1982	F008N005W14
Karl Hanneman and Bergelin Family Trust	347964	LC 792	11-Jun-1982	F008N005W14
Karl Hanneman and Bergelin Family Trust	347965	LC 793	11-Jun-1982	F008N005W13
Karl Hanneman and Bergelin Family Trust	347966	LC 794	11-Jun-1982	F008N005W13
Karl Hanneman and Bergelin Family Trust	347967	LC 795	10-Jun-1982	F008N005W13
Karl Hanneman and Bergelin Family Trust	347968	LC 796	10-Jun-1982	F008N005W13
Karl Hanneman and Bergelin Family Trust	347969	LC 797	10-Jun-1982	F008N004W18
Karl Hanneman and Bergelin Family Trust	347970	LC 798	9-Jun-1982	F008N004W18
Karl Hanneman and Bergelin Family Trust	347971	LC 799	8-Jun-1982	F008N004W18
Karl Hanneman and Bergelin Family Trust	347972	LC 800	8-Jun-1982	F008N004W17 F008N004W18
Karl Hanneman and Bergelin Family Trust	347973	LC 801	8-Jun-1982	F008N004W17
Karl Hanneman and Bergelin Family Trust	347974	LC 802	8-Jun-1982	F008N004W17
Karl Hanneman and Bergelin Family Trust	347975	LC 803	8-Jun-1982	F008N004W17
Karl Hanneman and Bergelin Family Trust	347976	LC 891	12-Jun-1982	F008N005W14
Karl Hanneman and Bergelin Family Trust	347977	LC 892	11-Jun-1982	F008N005W14
Karl Hanneman and Bergelin Family Trust	347978	LC 893	11-Jun-1982	F008N005W13
Karl Hanneman and Bergelin Family Trust	347979	LC 894	11-Jun-1982	F008N005W13
Karl Hanneman and Bergelin Family Trust	347980	LC 895	10-Jun-1982	F008N005W13
Karl Hanneman and Bergelin Family Trust	348802	LC 688	4-Jun-1982	F008N005W15
Karl Hanneman and Bergelin Family Trust	348803	LC 787	4-Jun-1982	F008N005W15
Karl Hanneman and Bergelin Family Trust	348804	LC 788	4-Jun-1982	F008N005W15
Karl Hanneman and Bergelin Family Trust	348805	LC 884	31-May-1982	F008N005W16
Karl Hanneman and Bergelin Family Trust	348805	LC 884	31-May-1982	F008N005W16
Karl Hanneman and Bergelin Family Trust	348806	LC 885	31-May-1982	F008N005W15
Karl Hanneman and Bergelin Family Trust	348807	LC 886	25-May-1982	F008N005W15
Karl Hanneman and Bergelin Family Trust	348808	LC 887	2-Jun-1982	F008N005W15
Karl Hanneman and Bergelin Family Trust	348809	LC 888	4-Jun-1982	F008N005W15
Karl Hanneman and Bergelin Family Trust	348810	LC 984	31-May-1982	F008N005W21
Karl Hanneman and Bergelin Family Trust	348811	LC 985	31-May-1982	F008N005W22
Karl Hanneman and Bergelin Family Trust	348812	LC 986	25-May-1982	F008N005W22
Karl Hanneman and Bergelin Family Trust	348813	LC 987	4-Jun-1982	F008N005W22

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
Karl Hanneman and Bergelin Family Trust	348814	LC 1083	30-May-1982	F008N005W21
Karl Hanneman and Bergelin Family Trust	348815	LC 1084	30-May-1982	F008N005W21
Karl Hanneman and Bergelin Family Trust	348816	LC 1085	30-May-1982	F008N005W22
Karl Hanneman and Bergelin Family Trust	348817	LC 1086	25-May-1982	F008N005W22
Karl Hanneman and Bergelin Family Trust	348818	LC 1183	29-May-1982	F008N005W21
Karl Hanneman and Bergelin Family Trust	348819	LC 1184	29-May-1982	F008N005W21
Karl Hanneman and Bergelin Family Trust	348820	LC 1185	29-May-1982	F008N005W22
Karl Hanneman and Bergelin Family Trust	348821	LC 1186	25-May-1982	F008N005W22
Karl Hanneman and Bergelin Family Trust	348822	LC 1282	28-May-1982	F008N005W21
Karl Hanneman and Bergelin Family Trust	348823	LC 1283	28-May-1982	F008N005W21
Karl Hanneman and Bergelin Family Trust	348824	LC 1284	28-May-1982	F008N005W21
Karl Hanneman and Bergelin Family Trust	348825	LC 1285	28-May-1982	F008N005W22
Karl Hanneman and Bergelin Family Trust	348826	LC 1286	26-May-1982	F008N005W22
Karl Hanneman and Bergelin Family Trust	348827	LC 1287	26-May-1982	F008N005W22
Karl Hanneman and Bergelin Family Trust	348828	LC 1288	2-Jun-1982	F008N005W22
Karl Hanneman and Bergelin Family Trust	348829	LC 1382	27-May-1982	F008N005W28
Karl Hanneman and Bergelin Family Trust	348830	LC 1383	27-May-1982	F008N005W28
Karl Hanneman and Bergelin Family Trust	348831	LC 1384	27-May-1982	F008N005W28
Karl Hanneman and Bergelin Family Trust	348832	LC 1385	27-May-1982	F008N005W27
Karl Hanneman and Bergelin Family Trust	361326	LUCKY 90	24-Oct-1983	F008N004W06
Karl Hanneman and Bergelin Family Trust	361327	LUCKY 100	24-Oct-1983	F008N004W06
Karl Hanneman and Bergelin Family Trust	361328	LUCKY 200	24-Oct-1983	F008N004W07
Karl Hanneman and Bergelin Family Trust	361329	LUCKY 294	28-Oct-1983	F008N005W12
Karl Hanneman and Bergelin Family Trust	361330	LUCKY 300	24-Oct-1983	F008N004W07
Karl Hanneman and Bergelin Family Trust	361331	LUCKY 394	28-Oct-1983	F008N005W12
Karl Hanneman and Bergelin Family Trust	361332	LUCKY 401	24-Oct-1983	F008N004W08
Karl Hanneman and Bergelin Family Trust	361333	LUCKY 402	24-Oct-1983	F008N004W08
Karl Hanneman and Bergelin Family Trust	361334	LUCKY 403	24-Oct-1983	F008N004W08
Karl Hanneman and Bergelin Family Trust	361335	LUCKY 501	24-Oct-1983	F008N004W08

* - Placer claim

Note: Meridian Township Range and Section (MTRS) Location is the Federal land location system. Example F006S013E12 is a section of land located in the Fairbanks Meridian, Township 6 South, Range 13 East, Section 12.

APPENDIX 2: List Of Drill Holes

HOLE	EASTING	NORTHING	ELEVATION	HOLE LENGTH (m)
BAF-1	430060.00	7266021.00	518.20	213.40
BAF-2	430073.00	7266149.00	525.50	152.40
BAF-3	429760.00	7266096.00	506.00	150.90
BAF-4	430073.00	7265881.00	476.70	216.40
BAF-5	430078.00	7265765.00	460.20	189.90
BAF-6	429745.00	7265979.00	515.10	134.10
BAF-7	430056.00	7266034.00	518.20	304.80
BAF-8	430342.00	7266042.00	524.90	152.40
L-1	429726.00	7265450.00	503.00	31.00
L-2	429350.00	7265457.00	506.00	73.00
L-3	429050.00	7265715.00	468.00	46.00
L-4	429045.00	7265688.00	470.00	20.00
L-5	428910.00	7265675.00	454.00	70.00
L-6	428805.00	7265640.00	441.00	70.00
LC-TR-01	428883.00	7266132.00	358.10	91.40
LC-TR-02	428859.00	7266041.00	358.10	68.60
MK-04-01	428734.38	7265596.00	421.50	109.70
MK-04-02	428492.13	7265738.00	361.60	305.70
MK-04-03	428674.66	7265520.50	412.20	208.80
MK-04-04	428547.66	7265813.50	354.40	137.80
MK-04-TP1	429594.00	7265670.00	510.00	2.00
MK-04-TP2	429583.00	7265653.00	512.00	2.00
MK-04-TR1	429541.09	7265537.00	524.70	34.00
MK-04-TR2E	429598.03	7265763.00	514.80	85.00
MK-04-TR2S	429598.03	7265763.00	514.80	20.00
MK-04-TR2W	429597.06	7265763.50	514.80	85.00
MK-04-TR3	429602.97	7265704.00	516.40	33.40
MK-04-TR5	429570.00	7265621.00	512.00	15.00
MK-06-05	429099.00	7266101.00	403.00	305.10
MK-06-06	429299.00	7266298.00	405.00	205.40
MK-06-07	428772.31	7265845.00	412.80	276.50
MK-06-08	428915.28	7265897.00	408.70	288.30
MK-06-09	427614.00	7264251.00	223.70	124.70
MK-06-10	427533.00	7264335.00	228.20	10.40
MK-06-11	427691.00	7264430.00	242.30	17.10
MK-07-12	428915.28	7265897.00	408.70	282.90
MK-07-13	428773.31	7265847.50	412.80	351.10
MK-07-14	428774.81	7265846.00	412.80	44.80
MK-07-15	428774.81	7265849.00	412.80	281.60
MK-07-16	430220.00	7265985.00	531.30	332.80
MK-07-17	428773.41	7265621.50	427.70	421.80
MK-07-18	428853.63	7265780.00	431.80	301.10
MK-07-19	429002.63	7265704.00	458.40	436.20

MK-07-20	428851.72	7265720.00	435.30	244.30
MK-07-21	428925.81	7265760.50	440.20	310.00
MK-07-22	428703.31	7265764.00	408.50	382.80
MK-07-23	429075.75	7265779.50	458.80	290.20
MK-07-24	429529.81	7265631.00	508.90	372.20
MK-07-25	428399.63	7265253.00	368.20	330.40
MK-07-26	429900.00	7265470.00	438.00	28.40
MK-08-27	429592.59	7265927.30	499.90	201.80
MK-08-28	429518.31	7266005.70	485.90	229.20
MK-08-29	429896.00	7265778.70	470.10	266.70
MK-08-30	428891.91	7265737.88	438.70	345.20
MK-08-31	429142.44	7265606.61	479.10	376.40
MK-08-32	429186.50	7265431.15	474.10	400.00
MK-08-33	429066.25	7265091.11	427.50	300.00
MK-09-34	428771.9	7265545	427.53	296.27
MK-09-35	428851.1	7265491	437.15	276.45
MK-09-36	428782.5	7265215	409.49	697.69
MK-09-37	429109.1	7265406	463.73	527.3
MK-09-38	429251.3	7265388	477.33	215.8
MK-09-39	429524.8	7265999	487.82	309.37
MK-08-TR01	428869.84	7266061.44	342.40	21.30
MK-08-TR02	428834.63	7266031.09	338.80	28.00
MK-08-TR03	428834.63	7266031.09	338.80	4.10
MK-08-TR04	428869.84	7266061.44	342.40	26.10
MK-1	428945.00	7265820.00	442.00	76.00
MK-2	428825.00	7265850.00	427.00	77.00
MK-3	429500.00	7266190.00	465.00	28.00
MK-4	429493.00	7266117.00	478.00	15.20
MK-4B	429493.00	7266117.00	478.00	106.70
MK-5	428660.00	7265925.00	368.00	0.00
MK-6	428680.00	7265940.00	367.00	0.00
MK-RC-0001	428996.00	7265778.00	449.00	321.60
MK-RC-0002	429001.81	7265854.50	426.10	335.30
MK-RC-0003	428703.19	7265998.50	335.90	222.50
MK-RC-0004	428612.00	7265921.00	343.50	274.00
MK-RC-0005	428561.81	7265841.50	350.00	269.80
MK-RC-0006	429045.69	7265695.50	460.70	353.60
MK-RC-0007	428846.00	7265843.00	423.60	286.50
MK-RC-0008	428925.00	7265691.60	445.90	213.40
MK-RC-0009	428997.91	7265632.10	456.50	246.90
MK-RC-0010	428547.69	7265470.90	393.20	240.80
MK-RC-0011	428925.69	7265626.30	448.00	225.60
MK-RC-0012	428997.00	7265544.70	459.50	307.90
MK-RC-0013	428624.19	7265480.10	403.20	225.60
MK-RC-0014	428176.91	7265590.70	357.30	217.90
MK-RC-0015	428323.09	7265696.50	349.20	195.10
MK-RC-0016	428319.50	7265542.50	367.70	134.10

MK-RC-0017	428779.09	7265774.00	423.20	297.20
MK-RC-0018	428710.91	7265834.00	396.90	252.40
MK-RC-0019	428550.00	7265925.00	330.00	54.90
MK-RC-0020	428549.69	7265909.80	331.50	213.40
MK-RC-0021	428470.00	7265852.10	330.50	213.40
MK-RC-0022	428847.91	7265920.70	399.80	280.40
MK-RC-0023	428849.31	7265622.60	437.70	288.00
MK-RC-0024	428697.81	7265630.00	413.90	207.30
MK-RC-0025	428920.91	7265909.10	404.50	213.40
MK-RC-0026	428622.91	7265760.00	385.80	167.60
MK-RC-0027	428559.09	7265703.80	381.60	129.50
MK-RC-0028	428844.53	7266105.70	350.00	93.00
MK-RC-0029	429057.91	7265856.70	432.50	256.00
MK-RC-0030	428777.19	7265548.20	425.80	243.80
MK-RC-0031	428926.47	7265548.00	447.20	303.30
MK-RC-0032	428554.91	7265783.10	363.50	91.40
MK-RC-0033	428849.41	7265566.50	437.10	335.30
MK-RC-0034	429073.81	7265553.40	467.90	365.80
MK-RC-0035	429071.91	7265468.10	467.90	330.70
MK-RC-0036	429001.59	7265463.40	453.20	257.90
MK-RC-0037	429149.41	7265558.70	483.50	295.70
MK-RC-0038	428784.09	7265918.70	392.50	234.70
MK-RC-0039	428999.09	7265410.20	450.70	277.40
MK-RC-0040	428927.38	7265860.42	418.90	335.30
MK-RC-0041	428850.69	7265504.08	437.50	262.10
MK-RC-0042	428778.56	7265473.11	425.90	274.30
MK-RC-0043	428940.28	7265472.30	446.40	265.20
MK-RC-0044	428698.09	7265487.46	417.60	237.70
MK-RC-0045	428922.00	7265395.50	441.10	317.00
MK-RC-0046	429084.03	7265622.27	470.50	323.10
MK-RC-0047	429152.56	7265477.69	475.40	326.80
MK-RC-0048	429144.00	7265399.25	466.90	350.50
MK-RC-0049	428697.66	7265404.66	416.90	274.30
MK-RC-0050	429225.06	7265481.30	488.50	350.80
MK-RC-0051	428699.75	7265549.36	416.60	239.30
MK-RC-0052	428625.53	7265847.83	366.60	249.90
MK-RC-0053	428543.97	7265549.99	393.20	204.20
MK-RC-0054	429297.22	7265483.50	493.40	341.40
MK-RC-0055	428706.44	7265926.89	368.90	262.10
MK-RC-0056	428477.38	7265559.88	384.50	195.10
MK-RC-0057	429374.31	7265486.84	504.80	304.80
MK-RC-0058	428700.06	7266242.25	334.30	213.40
MK-RC-0059	429450.22	7265478.31	511.60	262.10
MK-RC-0060	429077.13	7265328.34	453.50	336.80
MK-RC-0061	429225.78	7265326.36	468.30	302.10
MK-RC-0062	429150.22	7265323.46	460.50	312.40
MK-RC-0063	429299.63	7265329.00	474.40	359.70

MK-RC-0064	429072.38	7265252.31	445.30	363.30
MK-RC-0065	429302.81	7265425.01	484.80	346.00
MK-RC-0066	429156.28	7265243.08	452.10	304.80
MK-RC-0067	429155.28	7265174.77	448.20	349.00
MK-RC-0068	429227.25	7265403.32	476.20	396.20
MK-RC-0069	429147.53	7265098.42	434.70	256.00
MK-RC-0070	429452.13	7265548.90	509.90	378.00
MK-RC-0071	428928.31	7265326.22	435.50	301.80
MK-RC-0072	428997.91	7265323.84	444.90	262.10
MK-RC-0073	429521.63	7265549.72	513.20	335.30
MK-RC-0074	428474.03	7265632.47	377.30	158.50
MK-RC-0075	428477.16	7265481.85	386.50	243.80
MK-RC-0076	429151.06	7265033.41	425.50	285.00
MK-RC-0077	428475.91	7265930.18	312.10	152.40
MK-RC-0078	429225.91	7265026.63	428.20	298.70
MK-RC-0079	428399.41	7265859.17	320.00	161.50
MK-RC-0080	428626.69	7265396.63	402.60	262.10
MK-RC-0081	428841.59	7265250.01	419.90	243.80
MK-RC-0082	429073.56	7265037.48	421.60	317.00
MK-RC-0083	428911.13	7265169.42	420.60	300.20
MK-RC-0084	429224.53	7265250.71	458.20	374.90
MK-RC-0085	429599.09	7265554.41	510.80	326.10
MK-RC-0086	429377.88	7265391.25	491.40	36.60
MK-RC-0087	429148.47	7264949.83	417.20	254.50
MK-RC-0088	429003.38	7265008.70	413.50	115.80
MK-RC-0089	429003.38	7265008.70	413.50	374.90
MK-RC-0090	429070.13	7264946.92	413.30	201.20
MK-RC-0091	429007.06	7264947.97	407.40	283.50
MK-RC-0092	429377.88	7265391.25	491.40	344.42
MK-RC-0093	429226.13	7265103.86	439.00	323.09
MK-RC-0094	429750.00	7265475.00	504.00	327.66
MK-RC-0095	429600.00	7266000.00	513.00	268.22
MK-RC-0096	428780.91	7265217.91	410.00	262.13
MK-RC-0097	429897.41	7265464.74	447.73	237.74
MK-RC-0098	428925.00	7265112.11	415.29	219.46
MK-RC-0099	429296.66	7264946.83	419.03	268.22
MK-RC-0100	429214.03	7264951.65	418.33	274.32
MK-RC-0101	429294.00	7265027.91	429.73	295.66
MK-RC-0102	429296.25	7265176.16	453.02	274.32
MK-RC-0103	429229.09	7265170.67	449.21	306.63
MK-RC-0103a	429225.00	7265175.00	449.78	6.10
MK-RC-0104	429159.75	7264696.23	386.59	128.02
MK-RC-0105	429138.44	7264694.52	387.76	190.50
MK-RC-0106	429071.19	7265245.22	445.85	335.28
MK-RC-0107	429296.03	7264725.13	378.26	224.03
MK-RC-0108	429296.72	7265103.06	442.38	271.27
MK-RC-0109	428934.3	7265034.7	409.7	284.99

MK-RC-0110	428996.0	7265174.3	430.5	353.57
MK-RC-0111	429446.9	7265637.8	504.2	303.58
MK-RC-0112	429376.1	7265625.5	500.4	356.62
MK-RC-0113	429296.7	7265617.7	493.5	334.37
MK-RC-0114	429229.3	7265624.3	486.7	307.85
MK-RC-0115	428694.1	7264869.6	369.1	263.96
MK-RC-0116	428636.1	7264959.9	369.9	295.66
MK-RC-0117	428775.0	7265085.7	397.6	182.88
MK-RC-0118	428761.0	7264784.0	370.4	289.56
MK-RC-0119	428774.3	7265081.3	397.7	225.55
MK-RC-0120	428610.5	7264794.5	353.3	313.94
MK-RC-0121	428693.6	7265241.3	401.2	231.65
MK-RC-0122	428773.4	7264966.5	385.0	295.66
MK-RC-0123	428694.8	7265247.4	401.6	332.84
MK-RC-0124	428627.5	7265097.7	380.2	301.75
MK-RC-0125	428764.9	7265308.5	414.6	306.93
MK-RC-0126	428851.3	7265319.4	425.8	263.65
MK-RC-0127	428617.2	7265252.4	391.9	307.85
MK-RC-0128	429302.2	7265768.1	476.9	320.04
MK-RC-0129	428846.6	7265012.9	398.6	262.13
MK-RC-0130	429150.7	7265775.7	462.4	286.51
MK-RC-0131	428848.7	7264870.7	386.8	260.6
MK-RC-0132	428928.8	7264939.7	401.0	220.98
MK-RC-0133	428845.8	7265095.3	407.4	326.14
MK-RC-0134	428627.3	7265628.6	404.3	182.88
MK-RC-0135	429376.7	7265704.6	492.0	301.75
MK-RC-0136	428854.1	7265401.7	432.0	297.18
MK-RC-0137	429466.4	7265926.7	482.7	280.42
MK-RC-0138	428992.3	7265089.2	421.8	269.75
MK-RC-0139	429368.1	7265988.9	456.9	289.56
MK-RC-0140	428700.3	7265164.6	396.1	318.52
MK-RC-0141	429304.2	7265999.5	443.2	198.12
MK-RC-0142	428686.4	7265103.8	388.4	280.42
MK-RC-0143	430273.2	7266146	542.38	301.75
MK-RC-0144	429677.2	7265407	513.99	310.9
MK-RC-0145	430421	7266012	477.81	311.51
MK-RC-0146	429818.9	7265396	473.5	256.03
MK-RC-0147	429245.4	7264877	408.21	350.52
MK-RC-0148	430417.4	7266143	504.45	307.85
MK-RC-0149	429826	7265555	464.11	170.69
MK-RC-0150	429380	7264892	412.05	193.55
MK-RC-0151	429673.3	7265549	504.29	266.7
MK-RC-0152	430124.4	7265924	486.84	306.93
MK-RC-0153	429372.8	7265019	429.49	262.13
MK-RC-0154	429373.2	7265177	454.74	344.42
MK-RC-0155	429984.4	7265930	483.41	300.23
MK-RC-0156	429670.2	7265842	503.93	316.99

MK-RC-0157	429374.2	7265251	466.4	301.75
MK-RC-0158	429672	7265916	507.81	324.61
MK-RC-0159	429825.1	7265848	491.69	272.8
MK-RC-0160	429673.9	7266070	503.64	316.99
MK-RC-0161	429458.4	7264796	389.32	242.93
MK-RC-0162	429524.4	7266078	480.71	263.65
MK-RC-0163	429376.4	7264800	389.75	325.22
MK-RC-0164	429302	7264795	391.9	334.67
MK-RC-0165	429746.2	7265846	500.61	249.94
MK-RC-0166	429740.3	7265918	509.18	240.79
MK-RC-0167	429676.3	7265703	497.76	286.51
MK-RC-0168	429356.5	7264949	419.58	312.42
MK-RC-0169	430124.6	7266079	531.45	339.85
MK-RC-0170	429526	7265862	494.19	301.75
MK-RC-0171	429454.4	7264940	413.51	276.76
MK-RC-0172	429602.6	7264877	391.76	298.7
MK-RC-0173	429520.3	7264951	412.08	242.32
MK-RC-0174	429602.4	7265860	502.14	321.56
MK-RC-0175	428413.1	7265552	377.14	198.12
MK-RC-0176	429447.4	7265018	430.48	248.41
MK-RC-0177	429969.4	7266055	502.98	278.89
MK-RC-0178	429302.7	7264870	407.19	316.99
MK-RC-0179	428545.1	7265409	393.24	73.15
MK-RC-0180	429670.9	7265997	507.25	347.47
MK-RC-0181	429372.5	7265122	446.07	262.13
MK-RC-0182	428817.4	7265678	432.97	274.32
MK-RC-0183	429301.9	7265247	463.9	332.23
MK-RC-0184	428545.1	7265409	393.24	268.22
MK-RC-0185	429599	7266016	499.65	289.56
MK-RC-0186	429176.4	7265350	465.01	365.76
MK-RC-0187	429971.6	7265855	470.45	317.3
MK-RC-0188	429602.9	7266076	496.19	268.22
MK-RC-0189	429451.7	7265098	440.82	233.17
MK-RC-0190	429889.9	7265852	483.59	286.51
MK-RC-0191	430205.1	7265555	391.17	170.69
MK-RC-0192	429522.7	7265105	435.33	300.53
MK-RC-0193	430351.3	7265706	413.67	368.81
MK-RC-0194	429524.1	7265026	425.6	251.46
MK-RC-0195	430349.7	7266235	529.9	319.74
MK-RC-0196	430493.3	7265844	427.6	359.66
MK-RC-0197	428480.5	7265398	385.59	313.94
MK-RC-0198	430637.3	7265919	451.05	335.28
MK-RC-0199	430343.5	7266154	523.27	341.38
MK-RC-0200	428400.6	7265467	377.58	277.37
MK-RC-0201	429533.4	7265188	450.49	298.7
MK-RC-0202	430278.3	7266088	539.29	365.76
MK-RC-0203	430501.2	7265922	443.48	365.76

MK-RC-0204	429597.9	7265251	455.69	213.36
MK-RC-0206	429829	7265995	508.01	402.34
MK-RC-0207	429974.1	7265999	500.76	399.29
MK-RC-0208	429525.2	7265255	465.99	262.13
MK-RC-0209	429900.3	7265926	492.04	408.43
MK-RC-0210	429448.8	7265248	467.86	278.89
MK-RC-0211	429754.4	7266003	510.63	402.34
MK-RC-0212	429598.3	7265193	441.6	214.88
MK-RC-0213	429901.2	7266006	504.06	411.48
MK-RC-0214	429599.7	7265095	424.97	201.17
MK-RC-0215	429756.2	7266074	505.81	396.24
MK-RC-0216	429680.1	7265175	427.24	216.41
MK-RC-0217	429818.7	7265922	501.72	396.24
MK-RC-0218	429600.5	7265024	413.88	224.03
MK-RC-0219	429598.1	7264949	403.46	356.62
MK-RC-0220	429602.7	7265774	501.13	396.24
MK-RC-0221	430029.1	7265465	401.76	353.57
MK-RC-0222	429530.8	7265926	492.43	399.29
MK-RC-0223	429678.5	7265773	499.54	341.38
MK-RC-0224	429467.2	7265933	481.72	376.43
MK-RC-0225	429968.1	7265308	397.59	347.47
MK-RC-0226	429747.1	7265700	485.81	341.38
MK-RC-0227	429898.7	7265164	386.41	256.03
MK-RC-0228	429527.7	7266161	463.33	254.51
MK-RC-0229	429605.8	7265705	503.25	237.74
MK-RC-0231	429457.4	7266073	466.34	335.28
MK-RC-0233	429454.4	7266001	473.3	350.52
MK-RC-0234	429606.5	7265701	503.74	423.67
MK-RC-0236	429600.7	7265622	505.76	396.24
MK-RC-0237	429519.4	7266000	487.49	396.24
MK-RC-0239	429673.2	7265628	497.41	426.72
MK-RC-0240	429598	7266145	480.22	320.02
MK-RC-0242	429750.3	7265775	492.54	426.72
MK-RC-0244	429673.5	7266147	483.75	396.22
MK-RC-0250	429824.2	7265702	471.17	353.57
MK-RC-0256	429525.6	7265703	503.85	426.72
MK-RC-0257	429527	7265768	499.12	274.32
MN-1	428864.00	7266045.00	358.10	106.70
MN-2	428864.00	7266045.00	358.10	106.70
MN-3	428745.00	7266065.00	335.30	106.70
TL-10	428183.00	7265586.00	358.00	79.00
TL-11	429528.00	7266520.00	370.00	105.00
TL-12	429223.00	7266654.00	318.00	200.00
TL-13	429054.00	7266654.00	307.00	150.00
TL-14	427780.00	7265504.00	266.50	124.00
TL-6	433265.00	7269380.00	277.00	43.90
TL-7	428443.00	7266477.00	317.00	101.00

TL-8	428443.00	7266477.00	317.00	192.00
TL-9	428443.00	7266477.00	317.00	105.00