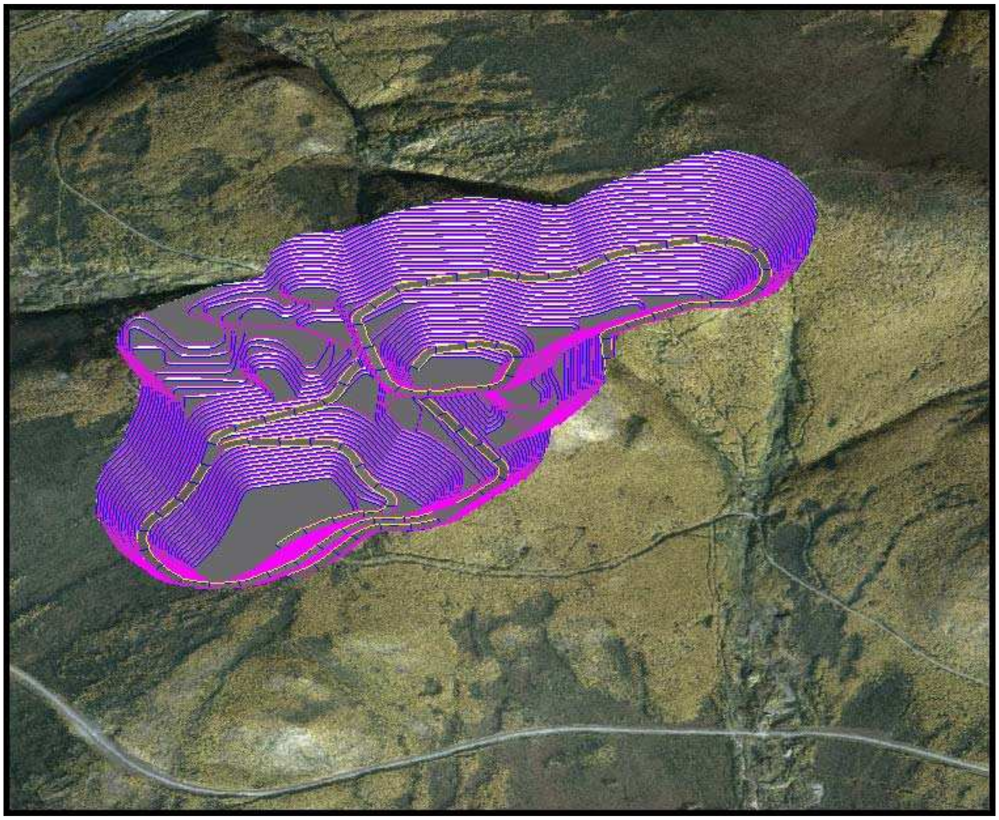

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SUMMARY REPORT ON THE LIVENGOOD PROJECT, TOLOVANA DISTRICT, ALASKA

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Cover Image: Geo-referenced surface photography of the Livengood area showing the $850 pit defined by incremental revenue optimization using the Livengood June 2010 in-situ resource model.

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1.0 Summary

1.1 Introduction

The Livengood project is now in transition from an exploration project to undertaking a Pre-feasibility Study. As part of this shift to prefeasibility assessment, a Preliminary Assessment (**PA**) was performed to evaluate preliminary project concepts including possible mineralization processing methods, estimates of capital and operating costs, and preliminary pit design scenarios, with respect to the resource estimate prepared on data to May 31, 2010 and reported in a previous technical report (Klipfel, Carew and Pennstrom, 2010b) released in June, 2010.

Individual sections of this report have been prepared by Qualified Persons representing different technical specialties. Mr. Timothy Carew (P.Geo) of Reserva International, LLC of Reno, NV was responsible for the geologic description and compilation of the report, and also for the resource evaluation. William Pennstrom (Metallurgical Engineer) of Pennstrom Consulting Inc. of Denver, Colorado was responsible for the metallurgical section of the report and for the financial analysis. R. John Bell (Civil Engineer) of MTB Project Management Professionals, Inc. of Denver, Colorado was responsible for the costing review and preparation of the capital cost estimates. Quinton de Klerk (Mining Engineer) of Cube Consulting Pty Ltd. of Perth, Australia was responsible for open pit optimization and production scheduling.

Field investigations at the Livengood property continue, with a total of 7 drilling rigs working at the site during the Summer 2010 Program. The focus of the work has been expanded to include environmental baseline data collection, geotechnical data collection for design, site alternative assessment for project infrastructure location and groundwater hydrogeological testing in support of the Pre-feasibility Study. Drilling activities have been expanded to include district exploration and site condemnation, as well as continuing the resource definition and infill drilling at Money Knob. The geologic database supporting this report is the 434 diamond and reverse circulation holes that had been drilled on the property to May 31, 2010, and provided the basis for reporting an in-situ gold resource estimated and presented in the June 2010 technical report.

This report is the tenth in a series of technical reports and the ninth in support of resource estimates regularly updated as new drill information has become available. This report also describes prefeasibility concepts including possible mineralization processing methods, estimates of capital and operating cost, and preliminary pit design scenarios along with the geological and resource estimation procedures that have been undertaken by ITH. The currently reported resource estimate includes material in the SW Zone and between the Core and Sunshine zones as determined by drilling data through May 31, 2010. It does not include drill results from ITH's 2010 Summer drill program that is currently in progress.

This report updates the June, 2010 technical report with the addition of information related to the results of the PA. The PA is based on the resource estimate completed in June 2010. The new information presented in this update report is based on interpretations of the geologic data, metallurgical data and in-situ resource model data presented in the June 2010 report (Klipfel, Carew and Pennstrom, 2010b) to support the development of pre-conceptual configurations of the potential mining project alternatives for mineralization at Money Knob. The project configurations that are the basis of the PA are for a heap leach only mining project and a combined heap leaching and milling project using gravity/flotation pre-concentration with Carbon-in-Leach leaching of the concentrates. Other processing alternatives are being considered and will be the subject of trade-off studies conducted as part of Pre-feasibility Study investigations that began in June 2010.

A group of cost, process recovery and production rate assumptions were created from the existing data as the basis for the PA analysis being reported in this update of the technical report. The assumptions were used with the June 2010 in-situ resource model (Carew, 2010b) to generate preliminary open pit mine designs and production schedules using incremental revenue optimization. Two open pit designs were considered: (1) an open pit constrained to the oxidized portion of the deposit, with relatively high drill data density (the heap leach only case) and (2) an unconstrained open pit that was revenue optimized with respect to the cost and gold recovery assumptions defined for the study (heap leach with gravity/flotation mill case).

Operating and capital cost estimates were generated for the two project configurations and were used, in conjunction with mining and processing schedules, to generate preliminary projections of financial performance. The preliminary financial performance was variable across the different alternatives analyzed and presented in more detail in Section 1.9, but in all cases the project showed positive financial performance at a long term gold price of $950 per gold ounce or higher.

1.2 Description and Location

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.

ITH controls 100% of its ~125 square kilometre Livengood land package, which is made up of 115 Alaska State mining claims, fee simple land leased from the Alaska Mental Health Land Trust, and four leases with private holders of state and federal patented and unpatented mining and placer claims.

1.3 History

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.

More recently, technical studies have been performed to generate metallurgical data for process definition, to generate preliminary open pit designs, and to develop pre-conceptual information on the location and capacities of potential tailings, waste and heap leaching facilities. Pre-conceptual project configurations have been generated from these studies which have been used as the basis for the projected operating and capital cost estimation. A PA for a large, open pit mining project was generated for the project concepts to guide ITH as it carries out the current Pre-feasibility Study.

1.4 Geology and Mineralization

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 in or 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. Mineralization is controlled partly by lithologic units, but 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 covers the most significant portion of the area with anomalous gold in surface soil samples, but still represents only about 25% of the total anomaly area.

1.5 Exploration, Drilling and Sampling

ITH has conducted drilling campaigns on the Livengood property since 2006. These programs initially identified mineralization in the Core Zone and then identified the Northeast, Sunshine, and Southwest zones through step out drilling and drill testing of areas with anomalous values in surface soil samples.

Nearly 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 to test other features and aspects of mineralization. Most holes have been spaced at 75m along lines 75m apart. A few holes are more closely spaced.

Diamond core holes represent approximately 10% of the total number of holes drilled. Core is recovered using triple tube techniques to ensure good recovery (>95%) and confidence in core orientation. The core is oriented using the ACT system and/or the EZ Mark tool.

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 collected over the course of each five-foot interval and captured for a primary sample, an equivalent secondary sample ("Met" sample) and a third batch of chips for logging purposes.

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

Down hole surveys of core and reverse circulation drill holes are completed using a Gyro-Shot survey instrument manufactured by Icefield Tools Corporation. Results of surveys and duplicate tests show normal minor deviation in azimuth and inclination for drill holes.

All RC samples are "logged in" on site, analyzed with a field portable Thermo Fisher Scientific NITON XRF before being sealed in super sacks, and delivered to ALS Chemex in Fairbanks for preparation. All core samples are initially logged at the drill rig for recovery, oriented features, RQD, and basic geologic features. More thorough logging and core mark-up is done at the Livengood camp. Core is sawed in half and bagged according to geologic intervals up to 1.5m and sealed in super sacks for delivery to ALS Chemex in Fairbanks.

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.

1.6 Quality Assurance/Quality Control and Data Verification

The QA/QC program implemented by ITH meets or exceeds industry standards. A QA/QC program includes insertion of blanks and standards (1/10 samples) and duplicates (1/20 samples). Blanks help assess the presence of any contamination that might be introduced by analytical equipment and help calibrate the low end of the assay detection limits. Commercial standards are used to assess the accuracy of the analyses. Duplicates help assess the homogeneity of the sample material and the overall sample variance. ITH has undertaken rigorous protocols to assure accurate and precise results. Among other methods, weights are tracked throughout the various steps performed in the laboratory to minimize and track errors.

Core and RC check samples have been collected during each drilling campaign by Paul Klipfel. Results from these samples, as well as blanks and standards included, are consistent with ITH's initial results. This includes a similar increase in variance for samples at higher grades, a pattern consistent with nugget effect. No systematic high or low bias has been observed. Additional RC check samples were collected by Mr. Carew in 2010, including blanks and standards. The results for these samples are pending at this time.

Data entry and database validation procedures have been checked and found to conform to industry practices. Procedures are in place to minimize data entry errors. These include pre-numbered, pre-tagged, bar-coded bags, and bar-coded data entry methods which relate all information to sample and drill interval information. Likewise, data validation checks are run on all information used in the geologic modeling and resource estimation process. Database entries for a random sample (10%) of drillholes used for the resource estimate were checked against the original Assay Certificates by Mr. Carew and the error rate was found to be within acceptable limits.

1.7 Mineral Processing and Metallurgical Testing

Metallurgical test work indicates that the Money Knob mineralization would be suitable for the two treatment options considered: oxidized, near surface mineralized material that could be treated by Carbon in Column (CIC) cyanide leaching (for example, heap leaching); and deeper, sulphide zones that will require Carbon in Leach (CIL) cyanide leaching approaches. Both the oxidized and sulphide zones have substantial free gold that can be recovered by gravity concentration, and gold associated with sulphide minerals in the deeper zones can be concentrated by flotation techniques.

Project concepts envision a heap leaching operation to address the near surface oxidized mineralization. This heap leaching operation would be followed by the expansion of the mine to the deeper, sulphide mineralization and construction of a gravity/flotation mill with CIL leaching of concentrates. The scheduling of the mine expansion and mill construction has been examined for different production rates and circumstances.

Test work undertaken to date is designed to determine optimal processes using combined methods. This work involves studies to determine chemical and physical characteristics of the mineralization and metallurgical response to process treatment parameters according to mineralization type. Test work includes assessment of grindability, abrasiveness, optimal particle size for downstream treatment, and response to leach, flotation, or gravity recoveries as a function of oxidation and lithology. Previous work completed was sufficient to enable an estimate of heap leach recoverable gold for a portion of the mineralization as reported in the October 2009 technical report. The additional work on gold recovery from gravity, carbon in pulp (CIP), CIL, and flotation methods is on-going with the initial results presented in this report providing the basis to estimate gold recovery from the mill process.

Key findings to date include the following points:

- Most Livengood mineralization can be considered moderately soft to moderately hard with an average Bond Ball Work index of 15.8 ranging from 11.1 to 19.1.
- The majority of mineralization types are considered non-abrasive with an average abrasion index of 0.0809 and a range of 0.0023 to 0.2872.
- All Livengood mineralization responds to cyanide leaching to some degree.
- Some unoxidized mineralization with organic carbon has “active” or “preg-robbing” carbon.
- Leach times and gravity concentration indicate that some mineralization contains coarse gold.
- Gold recovery exceeded 90% at 10 mesh for some mineralization.
- Gold recovery improved for some mineralization with finer grinding.
- Gold recovery for various leach tests suggests that organic carbon is present in varying degrees in some mineralized materials, particularly in unoxidized mineralization.
- Carbon in Leach bottle roll tests indicate an average 84% recovery for the Sunshine Zone.
- Gold with sulfide is not classified as refractory mineralization.

- Combined gravity and flotation produced, on average, 90% recovery of gold.
- Conventional milling using gravity recovery combined with intensive Carbon in Leach leaching of gravity recovered gold concentrate achieved gold recoveries averaging 86%.

Metallurgical testing is on-going to confirm initial conclusions on process flow sheets and assumed process recoveries. A series of tests that simulate the mill flow sheet assumed for gravity/flotation with CIL leaching of concentrates are in progress. These tests focus on the main components of the mill feed where achieving the current process recovery assumptions will require an improvement in the leach recovery over current test results. Further column leach testing is planned to begin in Q4 2010 and in Q1 of 2011 to verify heap leach assumptions. Column leach composite samples are being developed at a 1/2 inch top size from existing core, and PQ size core that is being produced in Q3 2010 will be used to develop 1 1/2 inch top size column tests. Trenching for a bulk sample to test run-of-mine size material in large columns is planned for Q4 2010.

1.8 Resource Estimation

This report presents a resource estimate updated from the March 2010 estimate by incorporating data from an additional 64 drill holes. The resource model was constructed using Gemcom GEMS® and the Stanford GSLIB (Geostatistical Software Library) MIK post processing routine. The resource was estimated using Multiple Indicator Kriging techniques.

Model parameters include, among others, two oxidation indicators and a single lithology indicator for each minor lithology. A three-dimensionally defined lithology model, based on interpretations by ITH geologists, was used to code the rock type block model. A three-dimensionally defined probability grade shell (0.1 g/t) was used to constrain the gold estimation. Gold contained within each block was estimated using nine indicator thresholds. The block model was tagged with the geologic model using a block majority coding method. Because there are significant grade discontinuities at lithologic contacts, hard boundaries were used between each of the lithologic units so that data for each lithology was used only for that unit.

A summary of the estimated in-situ mineral resource is presented below for cutoff grades of 0.3 (the assumed cutoff utilized in the PA), 0.5, and 0.7 g/t gold.

Model validation checks include global bias check, visual validation, and swath plots. In all cases, the model appears to be unbiased and fairly represent the drilling data.

TABLE 1.1
RESOURCE ESTIMATION SUMMARY
JUNE 2010

Classification	Au Cutoff (g/t)	Tonnes (millions)	Au (g/t)	Million Ounces Au
Indicated	0.30	789	0.62	15.7
Inferred	0.30	229	0.55	4.9
Indicated	0.50	409	0.83	10.9
Inferred	0.50	94	0.79	2.4
Indicated	0.70	202	1.07	6.9
Inferred	0.70	40	1.06	1.4

Based on the study herein reported, delineated mineralization of the Livengood Deposit is classified as a resource according to the following definitions from National Instrument 43-101 and from CIM (2005):

"In this Instrument, the terms "mineral resource", "inferred mineral resource", "indicated mineral resource" and "measured mineral resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as those definitions may be amended."

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

The current basis of project information is not sufficient to convert the in-situ mineral resources to mineral reserves, and mineral resources that are not mineral reserves do not have demonstrated economic viability.

It is important to note that, compared to the March 2010 resource estimate, the estimated tonnage has increased in the Indicated category and has decreased in the Inferred category for all cutoff grades shown (0.30, 0.50, and 0.70 g/t gold). This change was due to addition of newly defined estimated resources in the SW Zone and between the Core and Sunshine Zones.

As part of ITH's quality assurance program, ITH commissioned an independent review of the resource estimation methodology. The review supports the MIK approach to estimation, but suggests that the block panel size and SMU size should be larger for the currently spaced drill grid and that the currently used 10m composite length should be reduced to 3m. In addition, the review also recommends reducing the size of the search neighborhood selected for the estimation. Using these recommendations, an alternative resource calculation was made. Overall tonnes and grade compare favorably where the two models have a common volume. The ITH model contains material estimated as projected below current drilling which was not present in the alternative calculation. This material is primarily from the Inferred category. ITH believes their understanding of geology and mineralization allows this projection but is testing the extrapolation in the Summer 2010 drill program.

1.9 Pre-feasibility and Preliminary Assessment

ITH initiated pre-feasibility studies in June of 2010 in order to determine the most effective mine development strategy. A PA of alternative project configurations was performed to provide guidance in the Pre-feasibility Study, the results of which are incorporated in this report. The PA evaluated both the mining of the oxide portion of the deposit, and the expansion of the mining into the deeper, sulphide portion of the deposit. Two processing configurations were addressed:

1) Open pit mining of the oxide portion of the Money Knob with processing limited to heap leach only; and

2) Open pit mining of both the oxide and sulphide zones with a combination of heap leaching and mill processing (gravity and flotation concentration with CIL). Heap leach processing will allow production of approximately 40% of the currently estimated mineable resource.

The PA is preliminary in nature, and is based on technical and economic assumptions which will be evaluated in the Pre-feasibility Study. The PA is based on the Livengood in-situ resource model (June, 2010) which consists of material in both the indicated and inferred classification. Inferred mineral resources are considered too speculative geologically to have technical and economic considerations applied to them. The current basis of project information is not sufficient to convert the in-situ mineral resources to mineral reserves, and mineral resources that are not mineral

reserves do not have demonstrated economic viability. Accordingly, there can be no certainty that the results estimated in the PA will be realized. The PA results are only intended as an initial, first-pass review of the potential project economics based on preliminary information.

The Heap Leach Only project configuration was evaluated using the following approach:

- heap leach metallurgical recovery assumptions and operating cost estimates were used in conjunction with the in-situ resource model to select an open pit mining shell using revenue optimization;
- the pit shell optimization was constrained to the oxidized portion of the deposit by assigning zero metallurgical recovery to the deeper, sulphide zone;
- an open pit design was developed from the open pit mining shell which considered access ramps, mining losses and increased waste required for an actual mining geometry;
- a production schedule for mining recoverable mineralization above 0.3 g/t cut-off grade, recoverable gold production and waste material was developed assuming mineralization production rate of 100 ktpd;
- capital costs were estimated for a project physical configuration that considered equipment, the location of the open pit and potential sites for waste dumps, heap leach pad and the process facility; and
- a financial model was created for the production schedule, capital cost estimate and schedule, and estimated operating costs to project the financial performance of the heap leach only project configuration.

Key statistics for the analysis of the Heap Leach Only project configuration are listed in **Table 1.2**.

All costs are 2010 USD, with no escalation. A long term gold price of $950 per ounce has been used for the financial performance projection, which is consistent with current outlooks and price levels averaged over the past 3 years. The projected performance of the Heap Leach Only configuration at $950 is relatively strong, with an IRR of 26.9% and a Net Present Value (NPV) at 5% discount rate of $579 M. Sensitivity of the financial performance was evaluated for a long term gold price between $750 and $1500 per ounce. The sensitivity to gold price indicates that the financial performance weakens quickly at gold prices below the long term assumption, dropping to an IRR of 6.5% and an NPV@5% of $34M for a gold price assumption of $750. Alternatively, at higher gold price assumptions, the financial performance increases substantially with the IRR increasing to 43.5% for an increase of gold price to $1150. Sensitivity of financial performance to assumed processing recoveries was also high, but performance was less sensitive to changes in operating and capital costs assumed.

Full exploitation of the Livengood resource will require the addition of a mill process which would allow extraction of the deeper, sulphide zones. A project configuration

TABLE 1.2
KEY STATISTICS FOR THE LIVENGOOD HEAP LEACH ONLY PROJECT CONFIGURATION

Parameter		Heap Leach Only
Long Term Gold Price	$US/oz	$950
IRR	%	26.9%
NPV @0.0%	k $US	$ 915,338
NPV @5.0%	k $US	$ 579,103
NPV @7.5%	k $US	$ 455,882
NPV @10.0%	k $US	$ 354,531
Initial Capex	k $US	$ 679,851
Deferred Capex	k $US	-
Sustaining Capex	k $US	$ 153,482
Life of Mine (LOM)	years	7.1
LOM mineralization production	Mt	259.3
Mined grade at 0.3 g/t gold cut -off grade	g/t	0.62
Contained gold mined	koz	5,177
Estimated LOM gold production	koz	3,648
Cash operating cost	$US/oz	$486
Total cost	$US/oz	$704
Stripping ratio	Waste:ore	1.10
Assumed LOM heap leach gold recovery	%	70.5%

incorporating a heap leach processing facility and a mill using gravity and flotation concentration with CIL for recovery of the gold from concentrates was evaluated at two different mill throughput assumptions. The combination heap leach and mill was evaluated using the following approach:

- heap leach and milling metallurgical recovery assumptions, and operating cost estimates were used in conjunction with the in-situ resource model to select an open pit mining shell using revenue optimization;
- the optimization process was only constrained by the recovery and cost assumptions for the different lithologic units;
- an open pit design was developed from the open pit mining shell which considered access ramps, mining losses and increased waste required for an actual mining geometry;

- two production schedules for mining recoverable mineralization above a 0.3 g/t gold cut-off grade, recoverable gold production and waste material were developed assuming an initial mineralization production rate of 100 ktpd for the heap leaching and the two mill throughputs of approximately 54 ktpd and 100 ktpd;
- following mill startup, the mining rate was set to maintain the mill production rate, with the heap leach production varying accordingly;
- capital costs were estimated for a project physical configuration that considered equipment, the location of the open pit and potential sites for waste dumps, heap leach pad and the process facility; and
- a financial model was created for the production schedule, capital cost estimate and schedule, and estimated operating costs to project the financial performance of the combined heap leach and mill project configuration.

Key statistics for the analysis of the combined Heap Leach and Mill project configuration at the two different mill throughputs are listed in **Table 1.3**.

Internal rates of return for the Heap Leach and Mill configuration are lower for both mill throughput assumptions at 15.4% and 18.5%, for the 50ktpd and 100ktpd throughputs, respectively. This is due to the larger investment required for construction of the mill and the longer mine life. However, the NPV@5% is greater for the combination Heap Leach and Mill project configurations at $813 M for the 50 ktpd mill throughput and $1,112 M for the 100 ktpd mill throughput. The greater NPV reflects the substantially greater gold production due to exploitation of the deeper, sulphide zones.

Sensitivity to gold price assumption is similar to the Heap Leach Only project configuration, with the IRR dropping to a -0.8% and 1.3% (50 ktpd/100ktpd throughput) for a decrease in gold price assumption to $750. Increasing the gold price assumption illustrates the substantial leverage of the Livengood Project to the gold price, where a $200 price increase (to $1150 per ounce) increases the IRR to 29.2% and 32.6% (50 ktpd/100 ktpd mill throughput), respectively. Financial performance was also highly sensitive to process recovery assumptions, but was less sensitive to changes in operating and cost assumptions.

ITH plans to focus on the development of the heap leaching operation in the oxidized zone, however, it recognizes that significant potential value would remain to be exploited and that construction of a mill would be required to exploit the full potential of Money Knob mineralization. ITH will conduct a two phase Pre-feasibility Study with the projected completion of Phase I - Heap Leach Operation in July 2011. A second phase, with projected completion in December 2011, will address the requirements for eventual addition of a mill to the project configuration. This two phase approach is required to assure that designs and decisions made for the Heap Leach Only operation do not adversely impact the potential for the addition of a mill.

Site drilling operations will be expanded to include condemnation and geotechnical investigations for the Pre-feasibility Study. Metallurgical testing for Phase I will consist of additional column leach tests at 1/2 inch, 1.5 inch and run-of-mine top sizes that are

TABLE 1.3
KEY STATISTICS FOR THE LIVENGOOD HEAP LEACH AND MILL PROJECT CONFIGURATION

Parameter		Heap Leach and 50 ktpd Mill	Heap Leach and 100 ktpd Mill
Long Term Gold Price	$US/oz	$950	$950
IRR	%	15.4%	18.5%
NPV @0.0%	k $US	$ 1,982,082	$ 2,236,376
NPV @5.0%	k $US	$ 813,143	$ 1,112,868
NPV @7.5%	k $US	$ 495,034	$ 759,768
NPV @10.0%	k $US	$ 275,370	$ 496,163
Initial Capex	k $US	$ 635,631	$ 682,839
Deferred Capex	k $US	$ 750,214	$ 1,026,658
Sustaining Capex	k $US	$ 503,596	$ 578,476
Life of Mine (LOM)	years	21	13
LOM mineralization production	Mt	648.3	648.3
Mined grade at 0.3 g/t gold cut-off grade	g/t	0.65	0.65
Contained gold mined	koz	13,625	13,625
Estimated LOM gold production	koz	10,580	10,580
Cash operating cost	$US/oz	$ 560	$534
Total cost	$US/oz	$ 739	$ 734
Stripping ratio	Waste:ore	1.07	1.07
LOM mill gold recovery	%	81.3%	81.3%
LOM leach gold recovery	%	72.6%	72.6%

scheduled to begin in October 2010. Engineering studies required to support the Phase I Pre-feasibility Study are:

- Metallurgical engineer to design the CIC process plant (out for tender);
- Site location, geotechnical assessment and design of the heap leach pad, waste dumps and water storage facilities (underway);
- Site infrastructure, reticulation and road corridor placement and design (to be defined);
- Geotechnical design of pit slopes (to be defined);
- Open pit design and mining production scheduling (underway);
- Open pit dewatering, site water balance and storage requirements (underway); and
- Construction cost and production operating cost estimation (to be defined).

1.10 Conclusions

It is concluded that a substantial gold resource has been identified at Money Knob and the surrounding area. Dedicated drilling has continuously enlarged the resource over the past several years. Current metallurgical studies are underway and results indicate that gold is recoverable through heap leach, and combined mill, CIP, CIL, gravity, and flotation techniques. Continuation of planned and in-progress metallurgical and mineralization processing studies will enable assessment of the best material processing and gold recovery techniques. As results for this work are completed, new cost estimates that incorporate optimized gold recovery techniques will be used for a more comprehensive development plan and economic assessment. At this stage in the evaluation, and based on the results of the PA, the report concludes that mineralization at Money Knob merits continued engineering, economic assessment and planning to proceed on that basis.

1.11 Recommendations

The Livengood project is now in transition from an exploration project to a Pre-feasibility Study. In support of this, ITH has added senior staff. Exploration of the Livengood project should continue with the aim of completing the current Pre-feasibility Study. ITH plans to drill 50,000 m in 2010 to accomplish this goal, and will continue field operations into the deep winter season. The proposed program is an appropriate amount of drilling for the needs of the project and the time available in the field season. Activities that will help advance the project in this direction include those listed below:

- conduct groundwater hydrogeologic characterization for both regional and open pit groundwater modeling;
- develop a regional groundwater model and site water balance;
- develop geotechnical data to support pit slope designs;
- perform site alternatives assessments to identify locations for tailings, waste, heap

leach, mill and water storage facilities;
- perform condemnation drilling and geotechnical investigations at potential facilities sites;
- verify metallurgical recovery assumptions by conducting expanded metallurgical testing;
- perform comminution studies to provide a basis for crushing and grinding design;
- develop detailed metallurgical process flow sheets and perform process trade-off studies and mill design;
- perform air quality and weather monitoring studies;
- develop engineering designs of process plant facilities;
- perform environmental baseline data collection, wetlands surveys and water quality surveys;
- develop community engagement strategy;
- develop permitting strategy;
- continue step out drilling to identify the extent of mineralization;
- focus infill drilling on areas where Inferred resource blocks can be converted to Indicated resource blocks laterally and at depth;
- drill close spaced holes to define a variographic cross;
- complete Phase I of the Pre-feasibility Study for a heap leaching operation; and
- complete Phase II of the Pre-feasibility Study to identify the potential schedule for mill construction and the milling project design.

ITH plans expenditures of approximately $37.5 million dollars in 2011 for the continuation of exploration, definition and condemnation drilling, and for technical studies to produce the Pre-feasibility Study. This expenditure is further subdivided into $21M for completion of Phase I of the Pre-feasibility Study on the heap leaching operation by mid-year and then an additional $16.5M for the completion of Phase II of the Pre-feasibility Study to investigate the inclusion of a mill in the project. This budget will be allocated to drilling, geological and geotechnical analysis of the deposit, metallurgical and comminution studies, facilities site planning, environmental and social base line studies, and project component design. The budget is significant, but appropriate for the studies and 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.

2.0 Introduction and Terms of Reference

2.1 Introduction

Reserva International LLC ("RI"), Pennstrom Consulting Inc. ("PCI"), Cube Consulting Pty. Ltd. ("Cube") and MTB Project Management Professionals, Inc. ("MTB") 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 and undergoing Pre-feasibility Study by ITH through its wholly-owned subsidiary, Talon Gold Alaska, Inc. ("TGA").

This report on the Livengood project is based on geologic data, resource estimation data and metallurgical testing results published in the June 2010 technical report (Klipfel, Carew and Pennstrom, 2010b). In this update, existing metallurgical data has been used to define processing alternatives and develop process recovery estimates by PCI. Pre-conceptual project configurations have been developed around the processing alternatives, including open pit mining design concepts based on pit optimization and production scheduling by Cube Consulting, process capital and operating cost estimates by PCI and project capital and operating cost review and scheduling by MTB. Preliminary economic analyses have been generated for three different project configurations based on financial models developed by PCI.

Each author is a Qualified Person and is responsible for various sections of this report according to their expertise and contribution. Mr. Timothy Carew is responsible for all sections of this report except Sections 16 and 18, as well as compilation of information. Mr. Carew was responsible for developing the resource modeling so that it incorporated the geologic interpretation and allowed the consideration of the metallurgical impacts on potential production plans. He was responsible for development of the initial pit optimization approach utilized in the October 2009 technical report, and assisted in transfer of the approach to the optimization performed by Cube Consulting for the PA. Mr. William Pennstrom Jr. is solely responsible for section 16. Section 18 information was developed jointly by Mr. Pennstrom, Mr de Klerk and Mr. Bell, of PCI, Cube Consulting and MTB respectively. 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 eight 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, et al., 2009a; Klipfel, et al., 2009b; Klipfel, et al., 2010a; Klipfel, et al, 2010b, Klipfel, 2010c). Gold assays and analyses of other elements along with geological, structural, engineering, and metallurgical data is from 434 holes drilled by ITH and previous explorers, including 50 RC holes and 5 diamond core holes drilled so far in 2010 as well as data from previous drilling programs.

Information presented in this report is based on data provided to RI, PCI, Cube and MTB by ITH as of August 31, 2010. 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:

- Timothy Carew in the course of four site visits and generation of modelling data from primary data provided by ITH.

- Bill Pennstrom, who has made one site visit to Livengood and one visit to a local operating mine to identify operating costs at that mine, and has assembled the process capital estimates and the financial models.
- Quinton de Klerk, who supervised and reviewed the pit optimization and production scheduling work.
- John Bell, who has made one site visit to Livengood, reviewed the Capex and Opex cost estimates and assembled the Capital Expenditure Schedules.

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

Mr. Timothy Carew, of RI in Reno, Nevada, Mr. William Pennstrom Jr. of PCI in Denver, Colorado, Mr. Quinton de Klerk of Cube in Perth, Australia, and Mr. John Bell of MTB in Denver, Colorado 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 August 31, 2010, and is in support of the Preliminary Analysis. Data on drill results from the currently on-going Livengood Summer 2010 drill program, released to the public on July 27, August 17, September 10 and October 7, 2010, have not been utilized in this report.

Mr. Carew, Mr. Pennstrom, Mr. de Klerk and Mr. Bell 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

ADEC	Alaska Department of Environmental Conservation
ADFG	Alaska Department of Fish and Game
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
CPM	Critical Path Management
g/t	grams/tonne
IRGS	Intrusion Related Gold System
ITH	International Tower Hill Mines Ltd.
KWh/T	kilowatt-hours per Ton
LOM	Life of Mine
M	million
MRS	Mineral Resource Services Inc.
Mtpa	million tonnes per annum
MW	megawatts

Opt	troy ounces per Ton
oz(s)	troy ounce(s)
PA	Preliminary Assessment
PCI	Pennstrom Consulting Inc.
QA/QC	Quality Assurance/Quality Control
QP	qualified person
ROM	run of mine
SHPO	State Historic Preservation Office
t	tonne
TGA	Talon Gold Alaska, Inc.
tpa	tonnes per annum
tpd	tonnes per day
ktpd	thousand tonnes per day
Mtpa	million tonnes per annum
tph	tonnes per hour
USACE	US Army Corps of Engineers
$ or USD	United States dollars
WBS	Work Breakdown Structure

2.4 Purpose of Report

The purpose of this report is to provide an independent evaluation of the Livengood project, its exploration history, in-situ resource and mine development potential based on exploration work, metallurgical evaluation and engineering scenarios through August 31, 2010, a resource assessment based on that data, the discovery opportunity and development prognosis based on known geology current exploration results, and cost, engineering design, and metallurgical recovery models 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, Mr. Carew has spent an aggregate of 30 days on the site during four visits, including discussions with on-site geologic staff and review of various aspects of the program Data provided by ITH included reports by Dr. Paul Klipfel (CPG). Dr Klipfel has spent an aggregate of thirty days on the site during eight visits reviewing core, examining outcrop, and discussing the project with on-site geologic staff and with Mr. Jeffrey Pontius, CEO of ITH, and Mr. Carl Brechtel, COO of ITH. In addition, Dr. Klipfel has undertaken independent petrographic evaluation of samples from the project.

Drilling, sampling, QA/QC, 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), Mr. Karl Hanneman, Livengood Project Manager, and Mr. Chris Puchner M.Sc. (ITH Chief Geologist; AIPG CPG 07048). Mr. Puchner is a Qualified Person 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), HDR Inc. (environmental studies), SLR Inc. (hydrology studies), Kappes Cassiday and Associates, (metallurgical test work), McClelland Laboratories Inc. (metallurgical test work), Hazen Research Inc. (metallurgical test work), Resource Development Inc. (metallurgical test work), and SRK (hydrologic investigations and acid base accounting, and engineering). Knight Piesold Consulting have performed pre-conceptual evaluations for tailings, waste dump and heap leaching facilities. Cube Consulting Pty. Ltd. has performed open pit optimization and production scheduling studies for the pre-conceptual mining. MTB Project Management Professionals, Inc. have reviewed the capital and operating cost assumptions, assembled capital expenditure schedules and assisted with Project Management systems development.

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

2.6 Field Examination

Mr. Carew has visited Livengood for a total of 30 days on four separate trips in 2009 and 2010, with the most recent visit from October 24-27, 2010. During the course of these visits, modelling work was conducted collaboratively with ITH geologic staff, database information and contained data were reviewed and validated. Visits also included review of the geologic and tectonic setting of the property, surface and down-hole survey procedures as well as examination of outcrop and drill core. Independent check samples were collected during the last visit, the results of which are pending.

Dr. Klipfel has visited the property eight times, with the most recent visit from August 21-26, 2010. 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 and core orientation procedures as well as detailed examination of outcrop, drill core and RC chips.

More recently, with the shift to engineering and metallurgical evaluation, and involvement of other specialists, field examinations are giving way to engineering review and evaluation by numerous independent parties.

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

Mr. Bell spent two days on site in October of 2010. Site characteristics were reviewed with ITH staff and the capital cost estimate and costing was reviewed with Mr. Carew.

Mr. de Klerk has not visited the Livengood site prior to publication.

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. In addition, numerous studies have been undertaken by independent third party specialists whose results are incorporated into the current PA.

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 independent contractors 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 land holding and is located at 65°30'52''N, 148°27'50''W (UTM 6W 429600, 7265520; WGS84).

The explored area and current resource footprint reported here lies on the northwest flank of Money Knob and adjacent ridge lines and slopes, the extent of which remains to be determined. 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.

4.2 Claims and Agreements

The Livengood Property (**Figure 4.2**) consists of an aggregate area of approximately 12,499 ha (30,939 acres) 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.

In February 2010, TGA increased its land position through the addition of AMHLT leased ground and Alaska State claims. 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

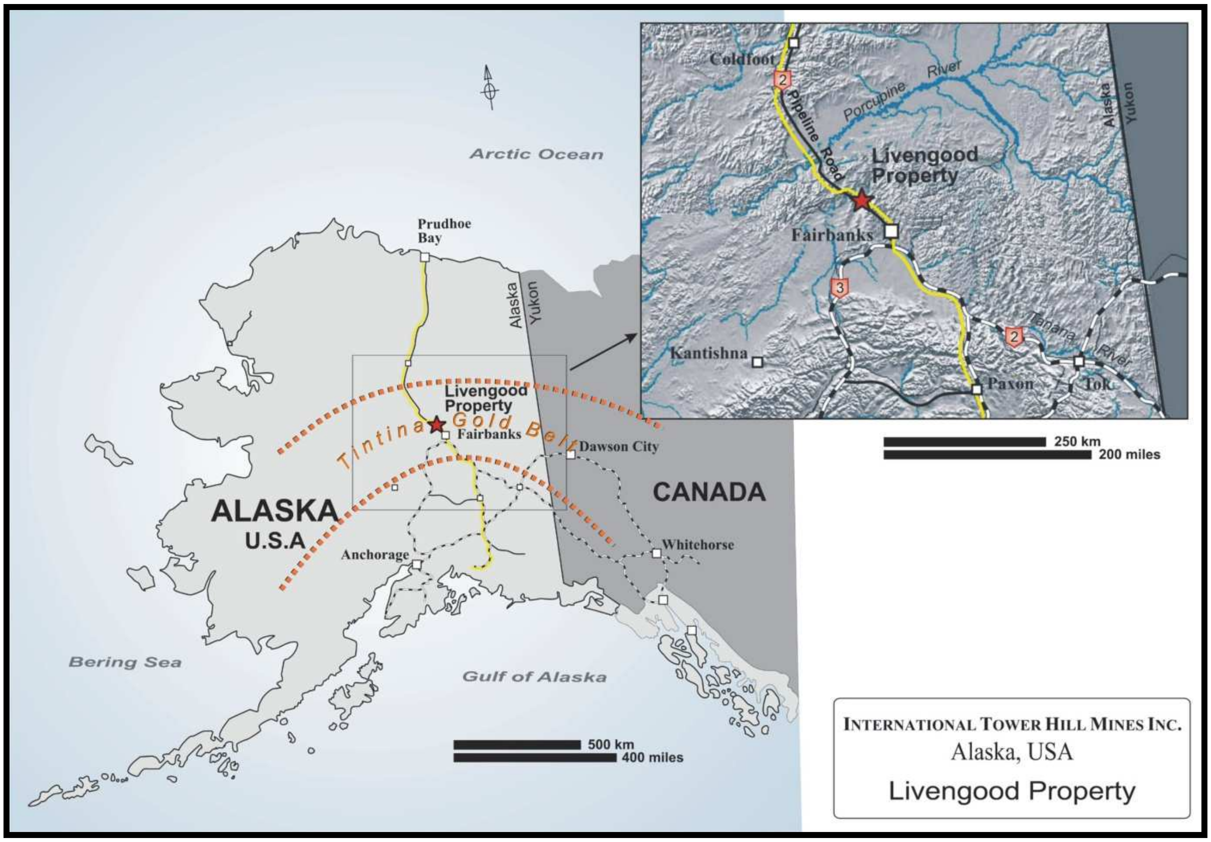

Figure 4.1. Location map showing the location of the Livengood project and the Tintina Gold Belt (orange dashed lines enclose the belt).

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

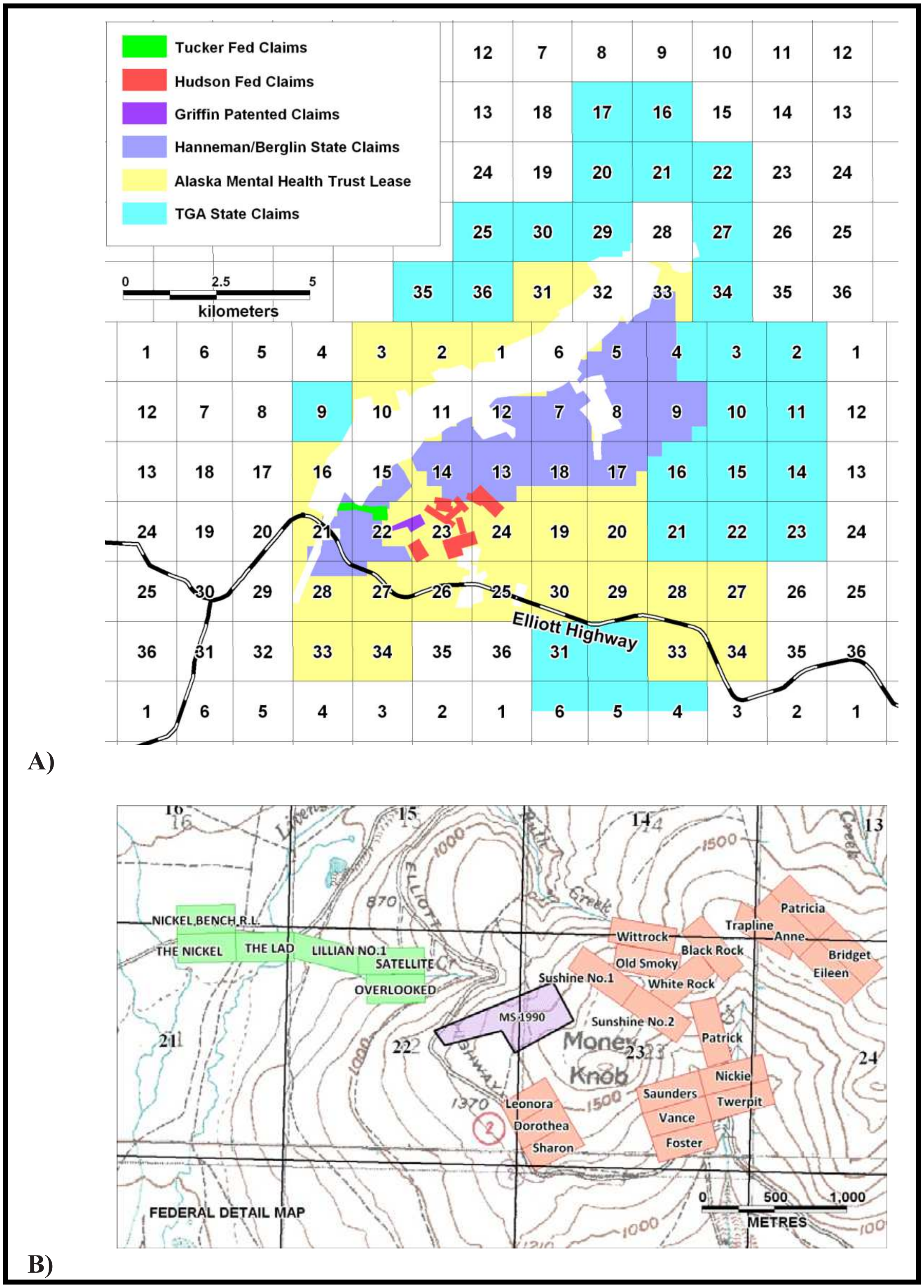

Figure 4.2. Land holding map showing the Livengood land position. **A)** The AMHL Lease holdings are shown in yellow, Alaska State Claims are shown in light blue, and holdings belonging to other parties shown in respective colors. **B)** Detailed map of the individual claims within the AMHL Lease.

TABLE 4.1
SUMMARY OF CLAIM HOLDINGS AND ANNUAL OBLIGATIONS

Holder	Type of Holding	Current Year	2010 Holding Obligation
AMHLT	State Mining Lease	6	$249,250 advance royalty; no work expenditure owing as ITH has banked work commitments to 2013
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 and claim rental fees of $28,730
Alaska State Lands	115 Alaska State mining claims	1	$17,920 claim rental paid with recording; $44,800 work commitment due by Sep. 1, 2010.

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 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 Lease with Option to Purchase Agreement 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 for claims held prior to 2010. Claims staked in 2010 will be subject to new work commitments.

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 Permits and Environmental Requirements

Project activities are required to operate within all normal Federal, State, and local environmental rules and regulations. These activities require permits from State and Federal Agencies including the United States Bureau of Land Management (BLM), United States Army Corp of Engineers (USACE), Alaska Department of Natural Resources (ADNR), Alaska Department of Fish and Game (ADFG), Alaska Department of Environmental Conservation (ADEC), and the State Historic Preservation Office (SHPO).

ITH staff and their subcontractors are conscientious in their care and diligence concerning historic features, flora and fauna, water quality, and general good stewardship toward the environment in their exploration activities. 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.

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 #2138 for ground controlled by the State of Alaska was issued on August 18, 2010 and covers calendar years 2010 thru 2014. This permit expands the area of activity to include locations which require investigation as possible sites for project infrastructure. 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.

One of the USACE permitting requirements is that wetland sites be drilled in winter to minimize surface impact to vegetation and soil. It also requires that all winter roads and pads in wetlands be fully reclaimed prior to April 15th. Some slopes are covered in a patchwork of vegetation consistent with a wetlands designation. These areas have been mapped by Three Parameters Plus, Inc., a natural resource consulting firm (**Figure 4.3**). Based on a new USACE Preliminary Jurisdictional Determination, an amendment to the original permit (dated November 13, 2008) was granted on March 15, 2010 and enables ITH to drill in areas of shrub and tundra on and around Money Knob. In support of this amended permit, the Alaska Department of Environmental Conservation (ADEC) has issued, on March 5, 2010, 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 surface waters beyond normal operational obligations which fall under operating permits issued by the state as outlined above.

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

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 artefacts in the project area. Twelve previously undocumented historic sites or artefacts were identified in 2008. No prehistoric artefacts and no previously unknown prehistoric cultural resources were located in the 2008 exploration area.

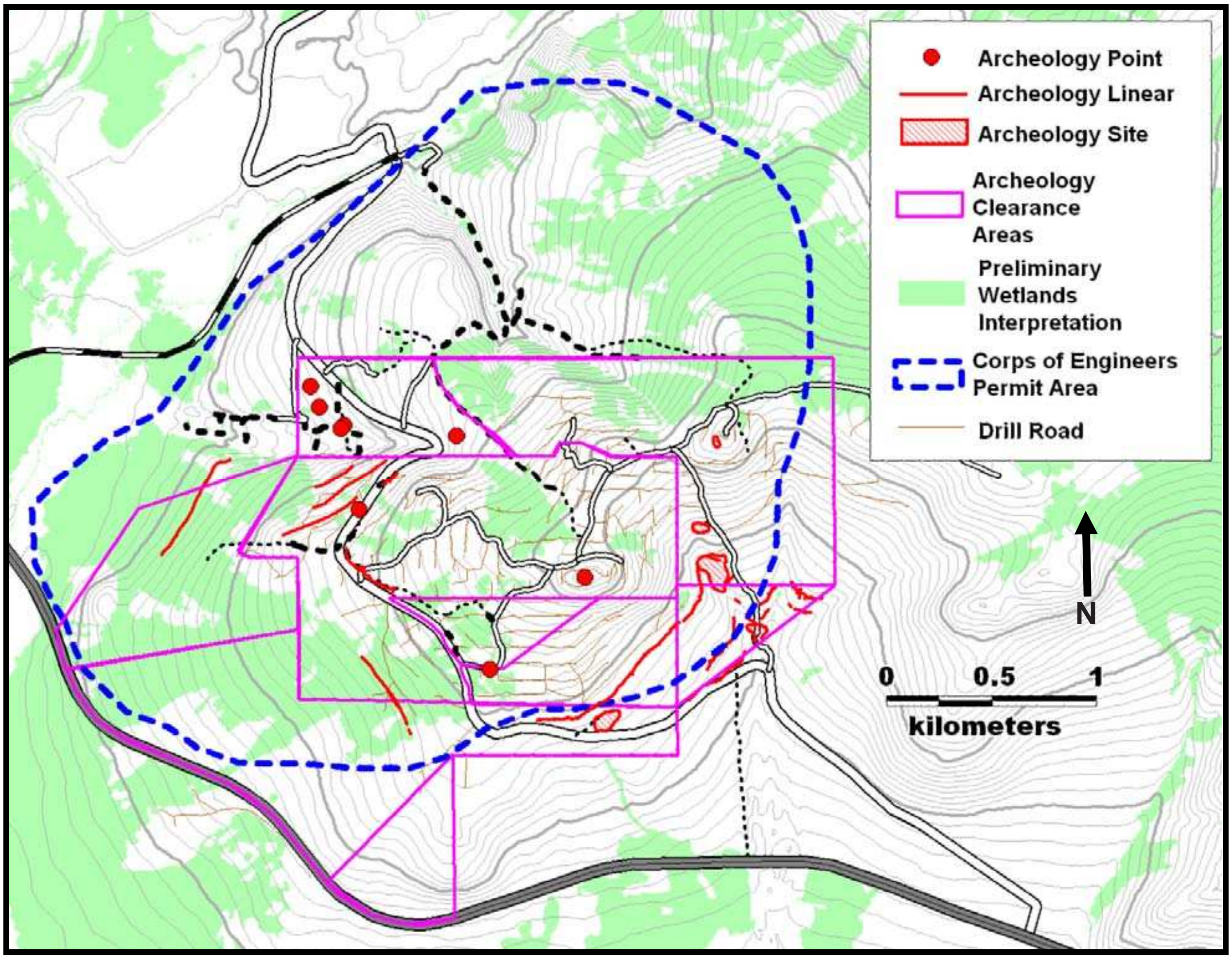

Figure 4.3. Map of the Money Knob area showing the archaeological study area. In addition, the location of cultural features identified in the survey, wetlands as currently assigned under the USACE preliminary wetlands interpretation, and the USACE permit area. The Elliott Highway runs across the southern portion of the map 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. archaeological) mining equipment, buildings and linear ditch features, and relocated a previously known prehistoric site within the expanded coverage area (**Figure 4.3**). Also, 12 select areas identified during the 2008 and 2009 programs were reviewed at a Phase II level (site documentation). NLUR has provided recommendations which include a policy of feature avoidance to prevent damage to the condition or integrity of identified features. All recommendations made by NLUR need to be made official by SHPO who will determine if any identified cultural resources require further action or isolation from disturbance.

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.

Three Parameters Plus, Inc. of Fairbanks, AK, 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 on and around ITH's land position.

Water samples have been collected from 14 sites on a near monthly basis from March through October. A 2009 report indicates apparent local and seasonal spikes among some analytes (Three Parameters Plus, Inc. 2009). These are deemed to be mostly natural and, in part, a reflection of past placer mining activity. Sampling will continue in order to develop base line trends for each sample location. One well has been established to monitor the static water table fluctuations on Money Knob and water table measurements are taken on each drill hole upon completion.

ABR Inc. of Anchorage, AK conducted a survey in 2009 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 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.

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 and is suitable for light aircraft. 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.

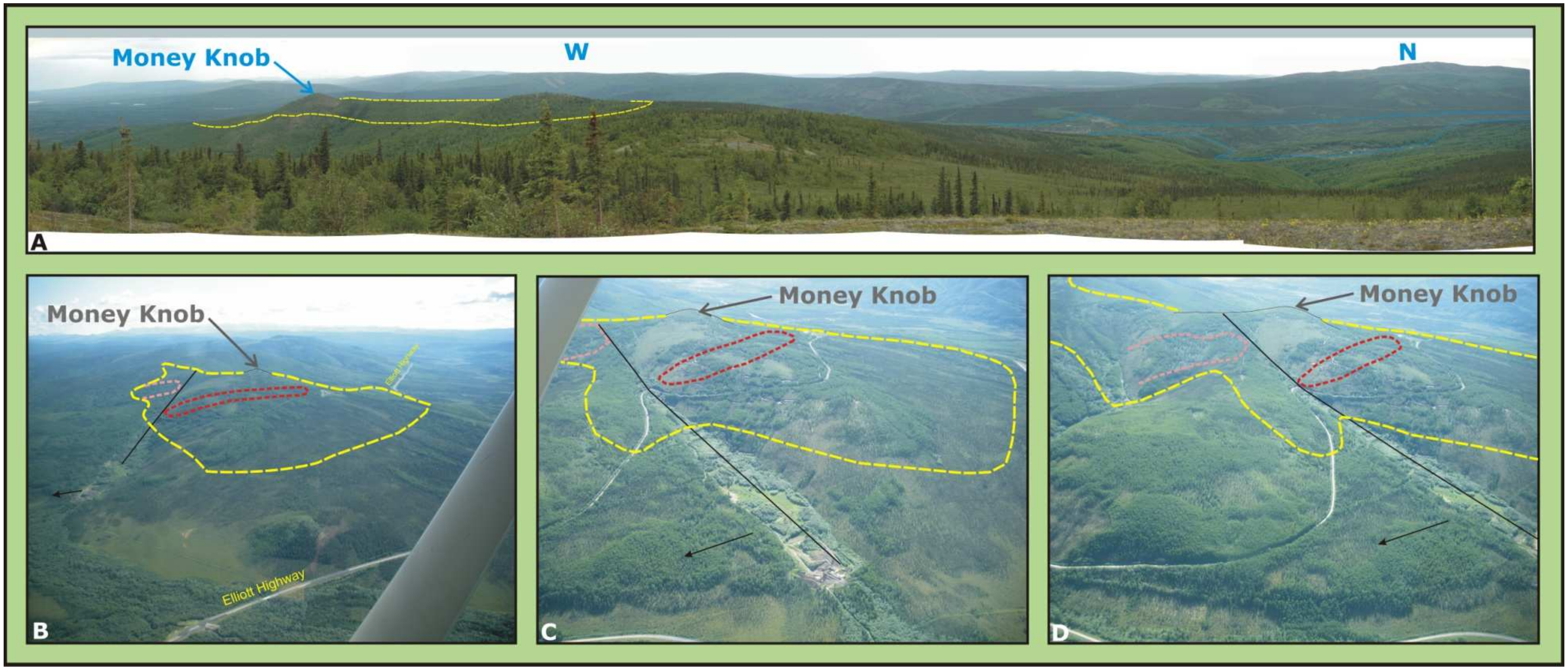

Figure 5.1 Photos of Money Knob and the project area. **A)** Panoramic view looking west and north toward Money Knob. Dashed yellow line outlines the perimeter of the area under investigation by drilling. Blue lines to the right outline placer workings to the north in Livengood Creek. **B - D)** Aerial 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 Sunshine Zone is outlined with a pink dotted line. Arrow indicates north.

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.

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, magnetic surveys, EM surveys, 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	195 reverse circulation holes; 12 diamond core holes	Infill drilling in wetland areas; discovery of Sunshine and SW Zones and other	Early results expanded the resource estimate presented in October,

Company / Year	Major Activity	Results	Comment
		areas of mineralization; expanded resource estimate	2009; later results discussed in this report
ITH 2010	50 reverse circulation holes; 6 diamond drill holes	Infill between Core and Sunshine Zones and expansion of SW Zone	Close wetland gaps

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

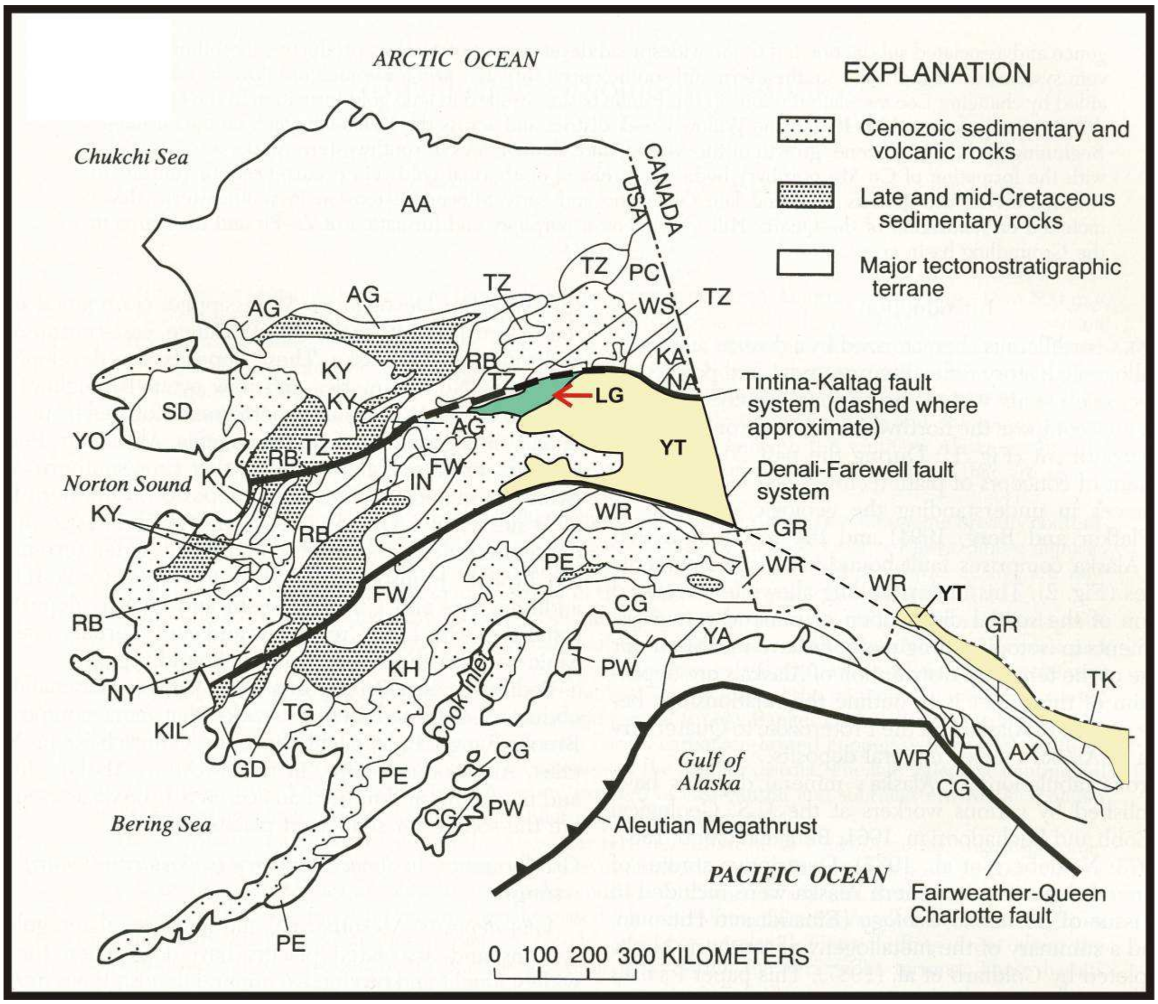

Figure 7.1. Terrane map of Alaska showing the location of the Livengood Terrane (LG; red arrow). The heavy black line north of the Livengood Terrane is the Tintina Fault. The heavy black line to the south of the Livengood and Yukon – Tanana Terrane (YT) is the Denali Fault. The Tintina Gold Belt lies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

each assemblage, is bounded by both low to possibly 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).

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 (IPzZ 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.

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.

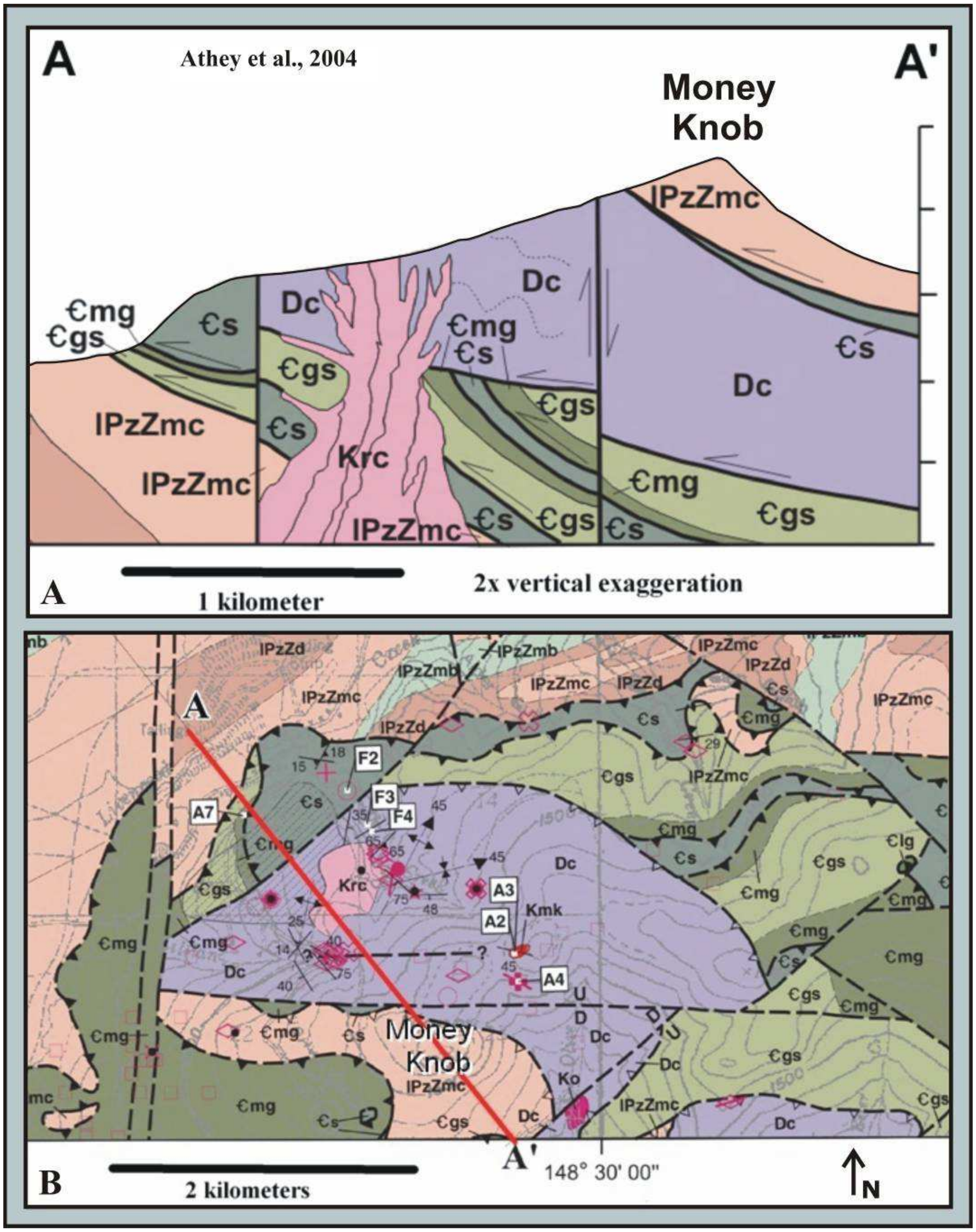

Figure 7.2. Geologic cross section and map of the Livengood project area (Athey, et al., 2004). A) Cross section through Money Knob illustrating the geological components of the Livengood District. lPZZmc are older siliceous shelf metasediments. Cs, Cgs and Cmg are Cambrian mafic and ultramafic volcanics 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.

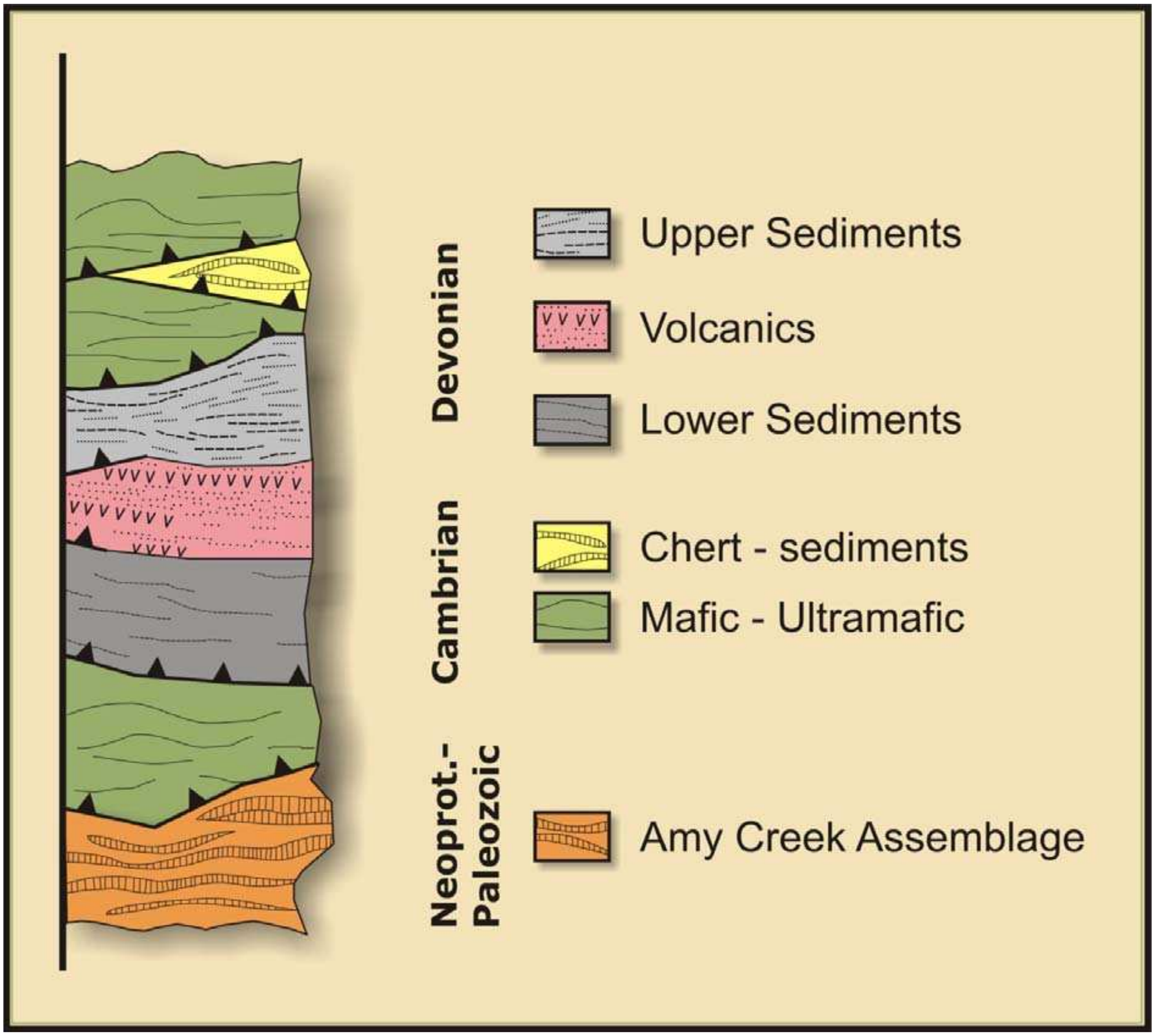

Figure 7.3 Diagrammatic lithologic column shows the tectonic stacking of rock groups in the Livengood area.

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.

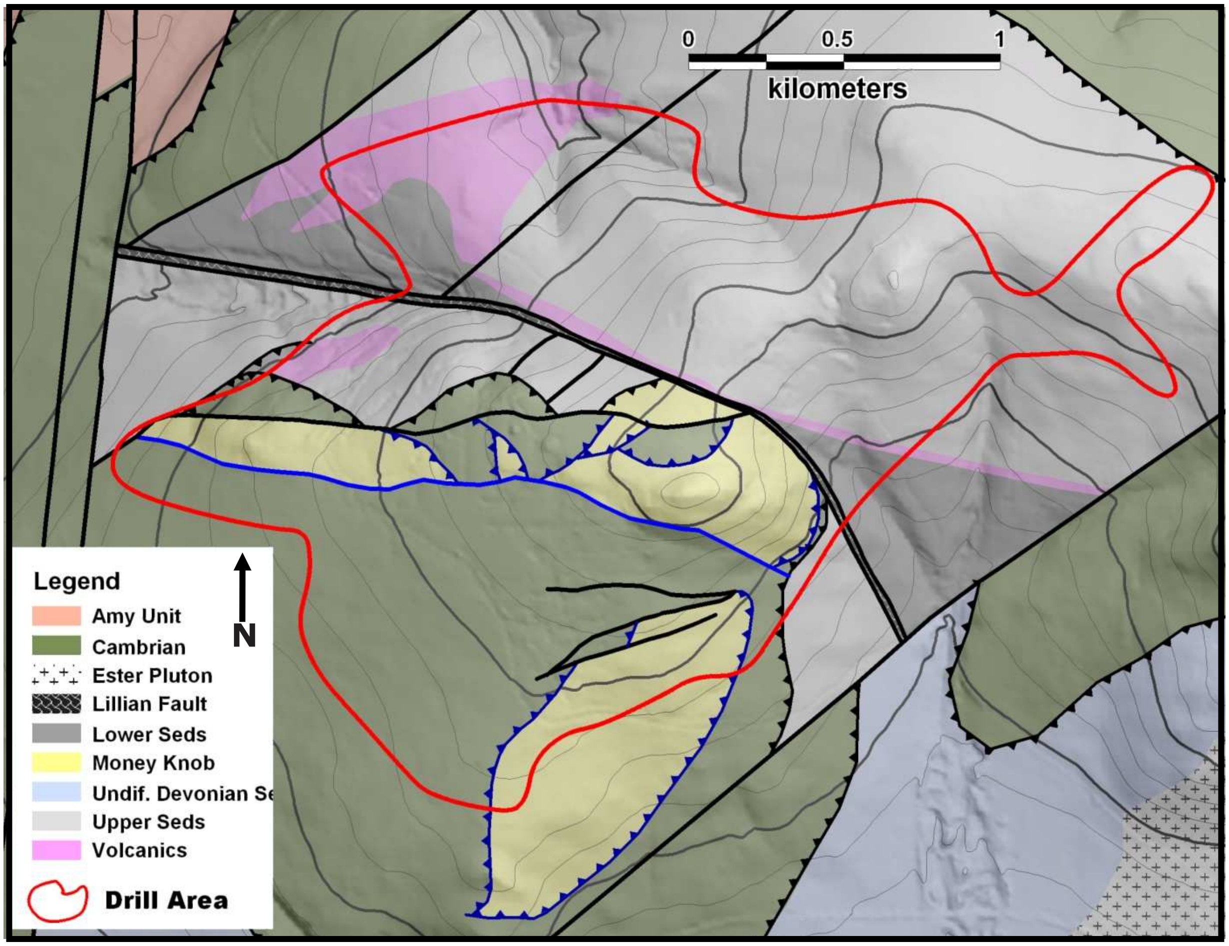

Figure 7.4. Generalized geologic map of the Money Knob area based on geologic work by ITH.

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, possibly Cretaceous, stocks and large dikes are biotite monzonite. Narrower, possibly 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.

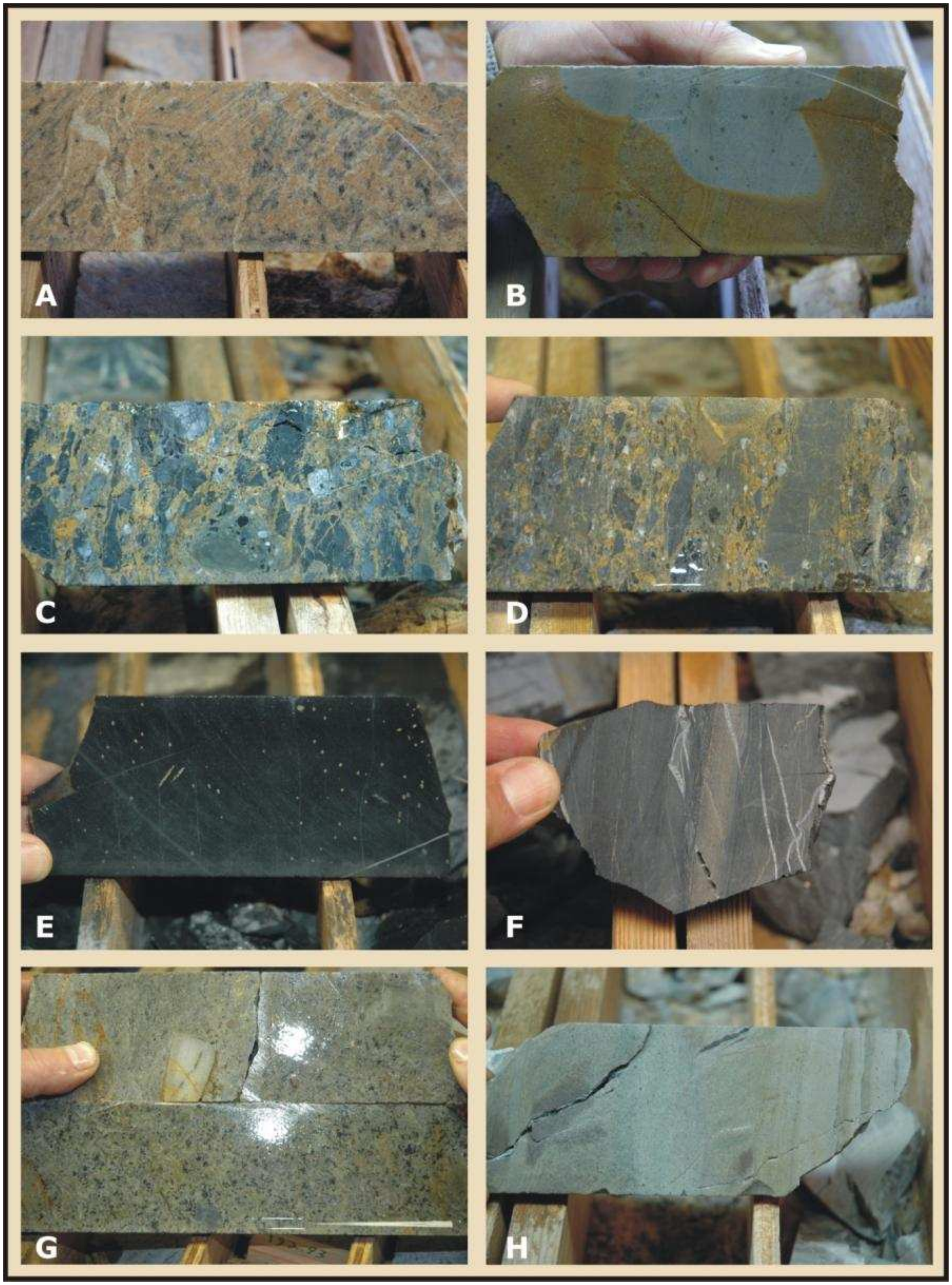

Figure 7.5. Photographs of key rock types at Livengood. **A)** ultramafic rock with carbonate alteration (yellow-brown); MK7-20, 13.5m; **B)** siltstone with carbonate and pyrite knots. Brown color is oxidation front. MK 07-18, 8.5m **C)** sedimentary conglomerate; at least some clasts appear to be rip-up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

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 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, steeply 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 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. The mineralized areas north of the Lillian fault is known as the Sunshine Zone where mineralization is related to a dike swarm in the steeply dipping sedimentary and volcanic rocks.

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

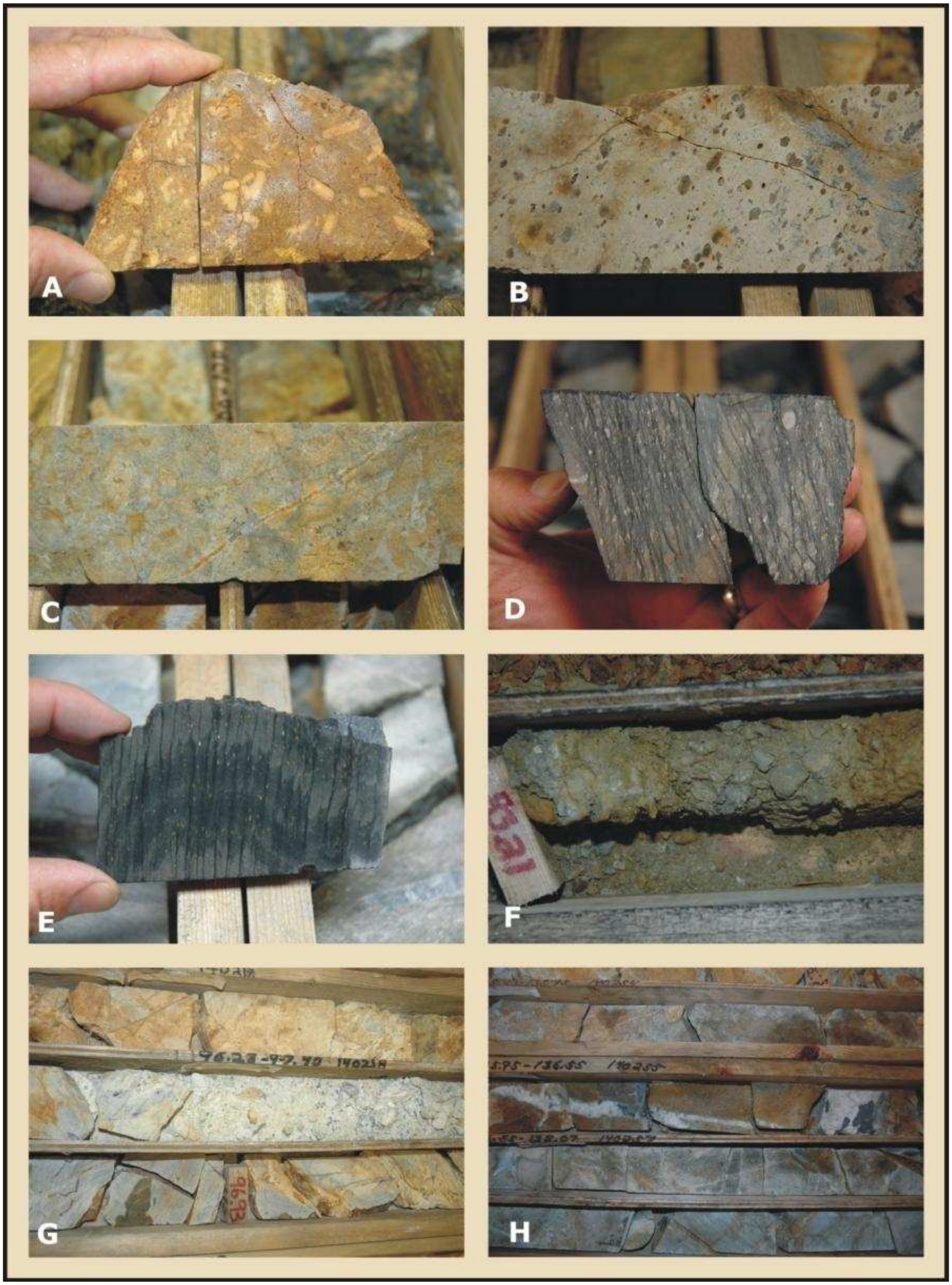

Figure 7.6. Photographs of key rock types and mineralization features. **A)** porphyry dike; MK07-18, 41.2 m; 1.01 g/t Au. **B)** amygdaloidal volcanic, presumably a flow, with possible Na alteration; MK07-18, 152-189. **C)** silicified volcanic breccia; MK07-18. **D)** argillite with more silty band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

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 (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, possibly 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.

7.3 Geological Interpretation

Geologic interpretation at Livengood depends on surface information gained through mapping and examination of outcrops, exposures in road cuts, and trenches. Subsurface information is acquired from diamond drill core and RC drill chips. Drill core provides clear macroscopic visual information on rock

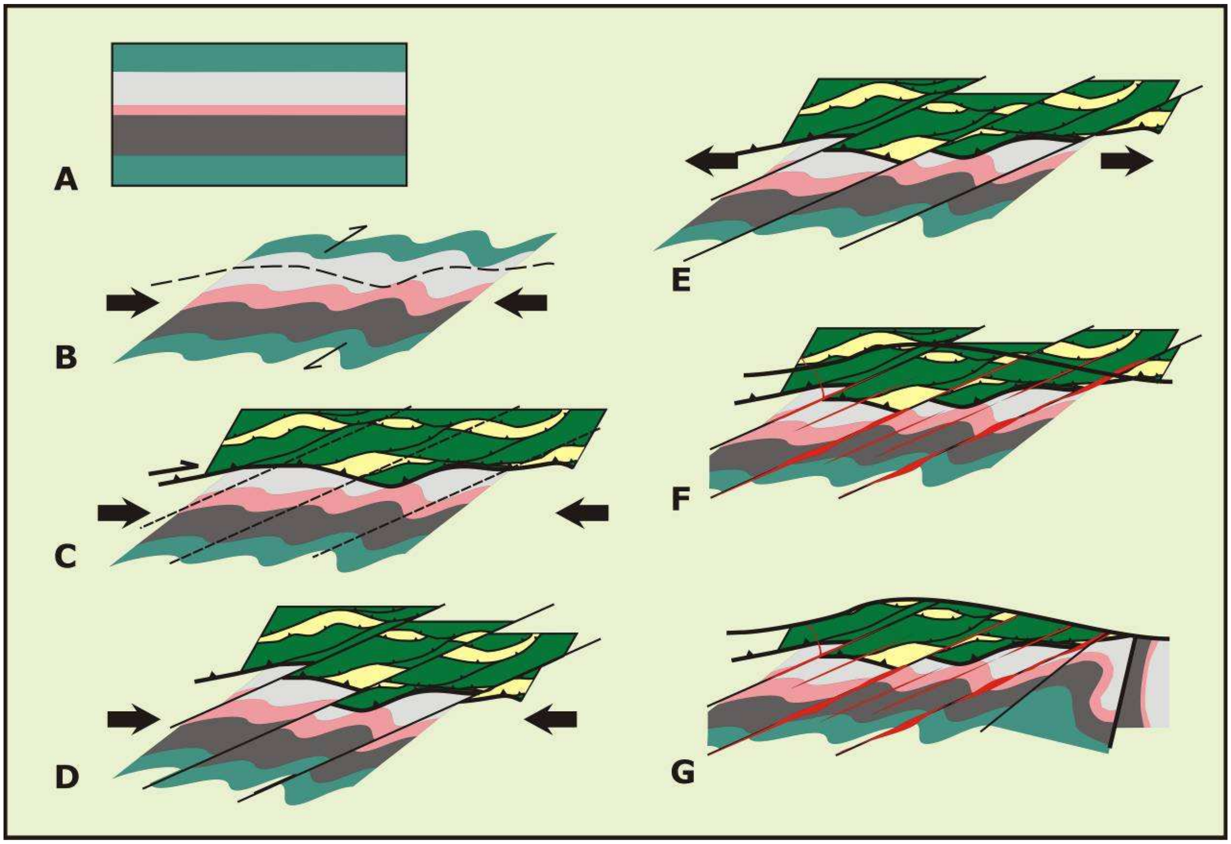

Figure 7.7. This cartoon shows an interpretive sequence of north-south sections and events to explain the structural relations observed at the surface and in drill core. The details and sequence of the events shown here are partly the interpretations of Klipfel (Klipfel, et al., 2010b). ITH staff geologists 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 to the south of Livengood.

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 is 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) Possible 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.

type and structural features. RC chips also provide visual information on rock type, but no structural information. In core, the orientation of structural elements (joints, faults, veins, contacts, etc) are measured and used to help understand the relative relations of structural components. Visual examination of core is used to assess rock type and alteration. Petrographic examination of select samples has helped determine alteration mineralogy and relative timing of successive alteration events.

In addition, rock composition is determined for RC samples through use of a portable XRF device (Thermo Fisher Scientific Niton XLT3) which provides a semi-quantitative measure of select elements, which are generally diagnostic of each rock type intersected in the drill hole. 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.

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. 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 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.

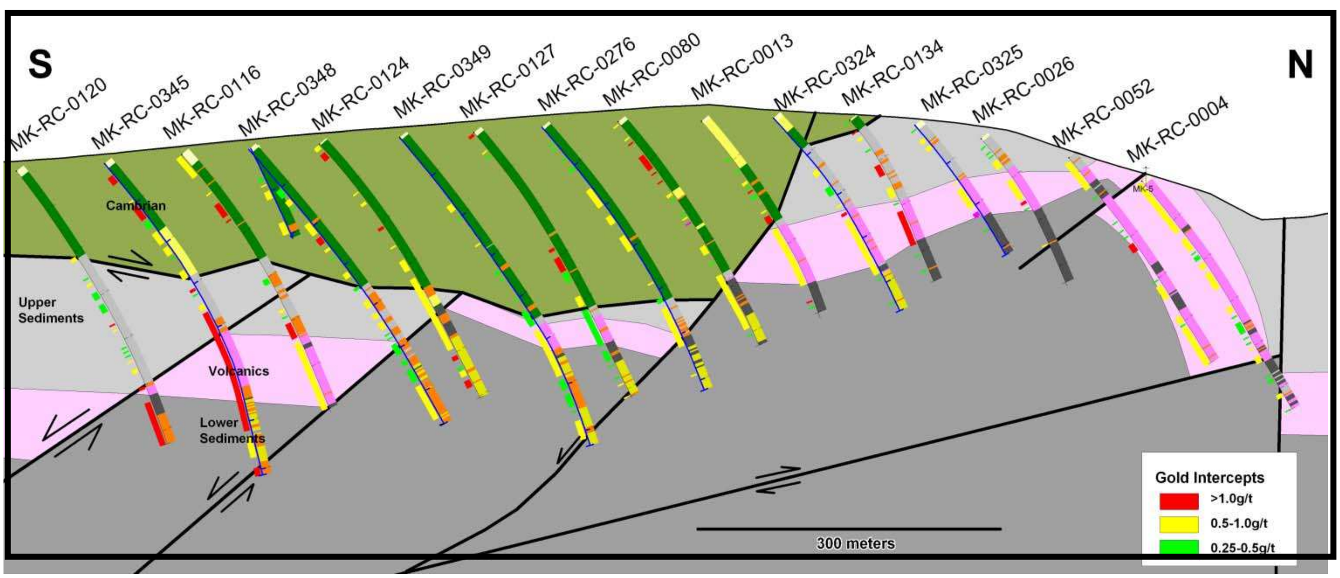

Figure 7.8. N-S Section 428625 E illustrates the complexities of thrust and normal fault interpretation and shows the southerly dip of high grade zones (red).

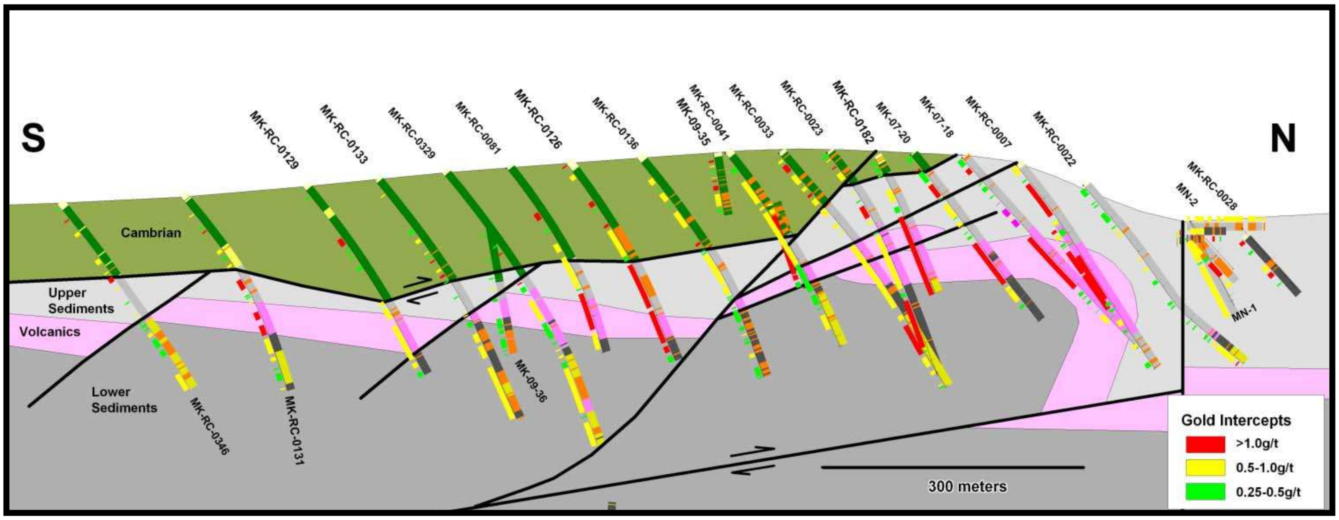

Figure 7.9. N-S Section 428850 illustrates the southerly dip of high grade zone (red) along the general stratigraphic pattern.

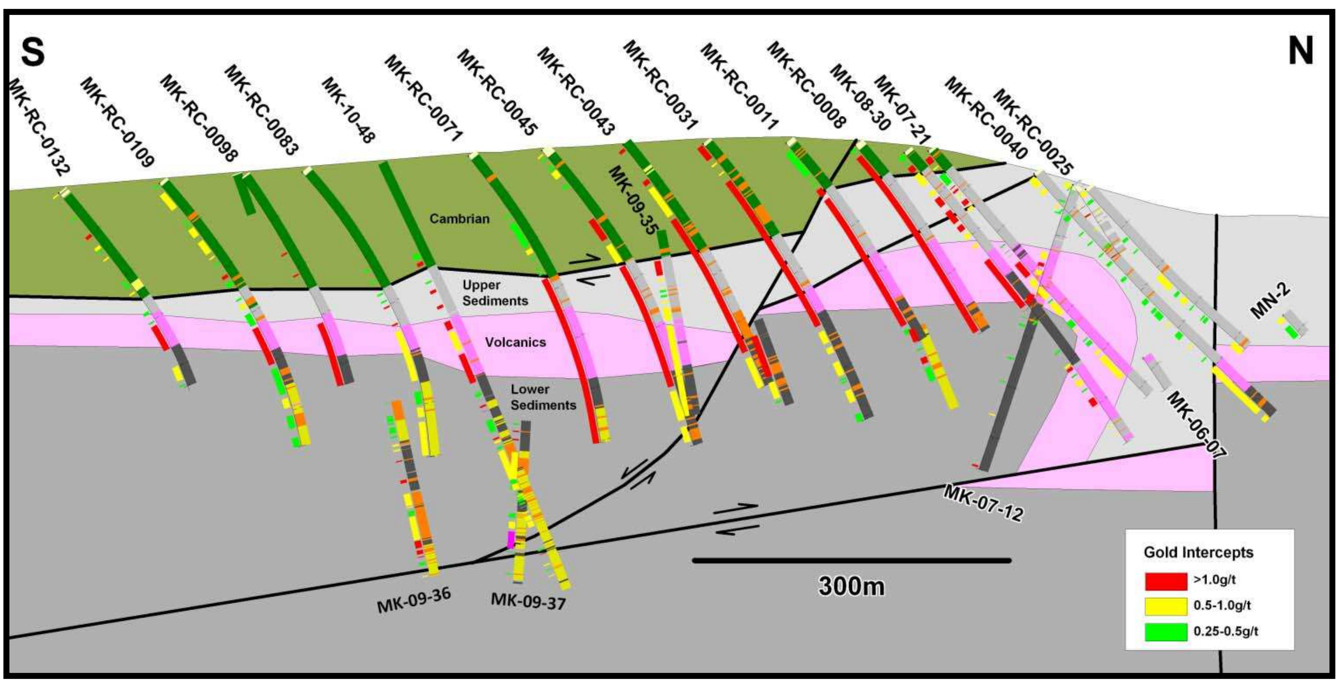

Figure 7.10. N-S Section 428925 illustrates the general southerly dip of mineralization and how it lies along the stratigraphic and structural grain.

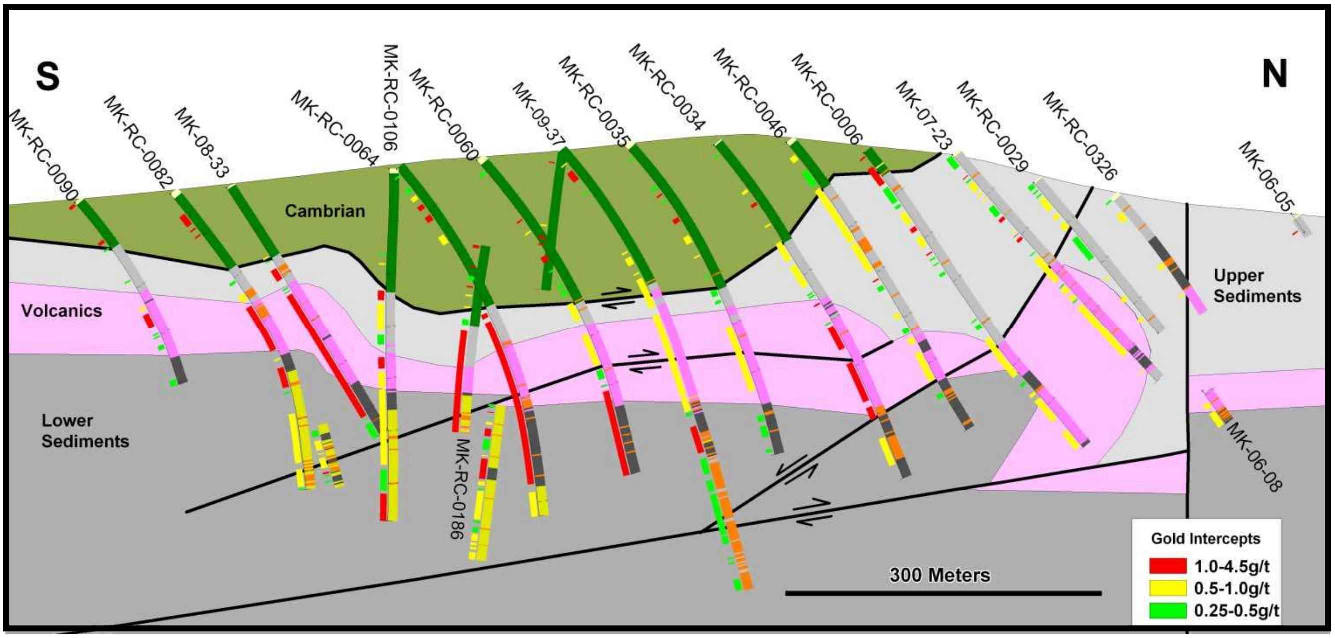

Figure 7.11. N-S Section 429075 illustrates the pattern of mineralization reflecting structural and stratigraphic controls.

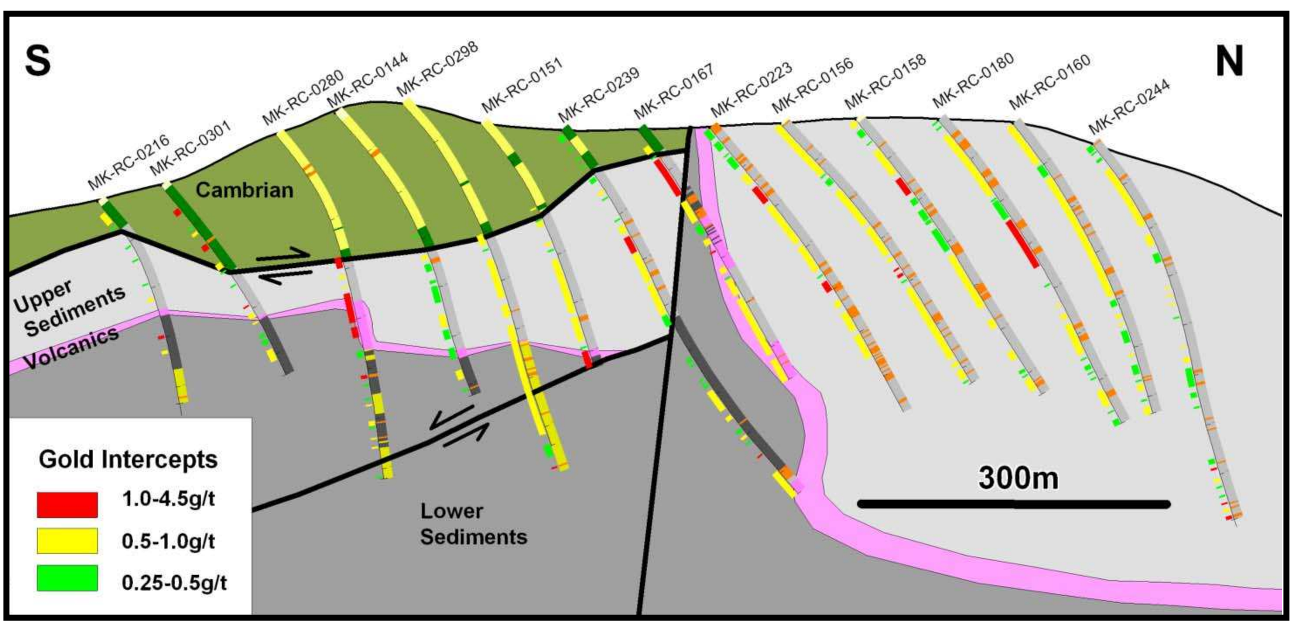

Figure 7.12. N-S Section 429675 illustrates the pattern of mineralization reflecting structural and stratigraphic controls.

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.

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. Historic to near recent prospecting in this area revealed numerous gold-bearing quartz veins, generally associated with dikes, sills and stocks of monzonite, diorite, and syenite composition. These rocks with their reduced magma type and porphyritic to brecciated textures, as well as common arsenopyrite, are characteristics common to many deposits of the Tintina Gold Belt (e.g. Brewery Creek, Donlin Creek) (McCoy, et al., 1997; Smith, 2000). However, no lode production has taken place at Money Knob.

Over the past 35 years, exploration of the area by various companies has included soil surveys by Alaska Placer Development, Cambior, AGA and ITH, and revealed 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 434 holes to May 31, 2010, 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 locally open in all directions, especially to the southwest and at depth.

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. Four contiguous principle zones of mineralization have been identified: the Core Zone, Sunshine Zone, Tower Zone, and Southwest Zone (**Figure 9.2**). Gold mineralization in the Core and Southwest Zone 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 also may be spatially associated with 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 siltstone, sandstone, and pebble conglomerate as opposed to shale. The diffuse style of mineralization is spatially associated with areas containing vein mineralization, but disseminated mineralization can be present where there is no discernable quartz veining to explain it.

In contrast to the Core Zone, mineralization north of the Lillian Fault within the Sunshine and Tower Zones is hosted dominantly in Upper Sediments. In this zone, mineralization is related spatially to swarms of dikes which appear to dip steeply to the south in a package of sediments that dips steeply to the north. Disseminated sulphides occur in the Sunshine Zone as in the Core Zone, but two things distinguish

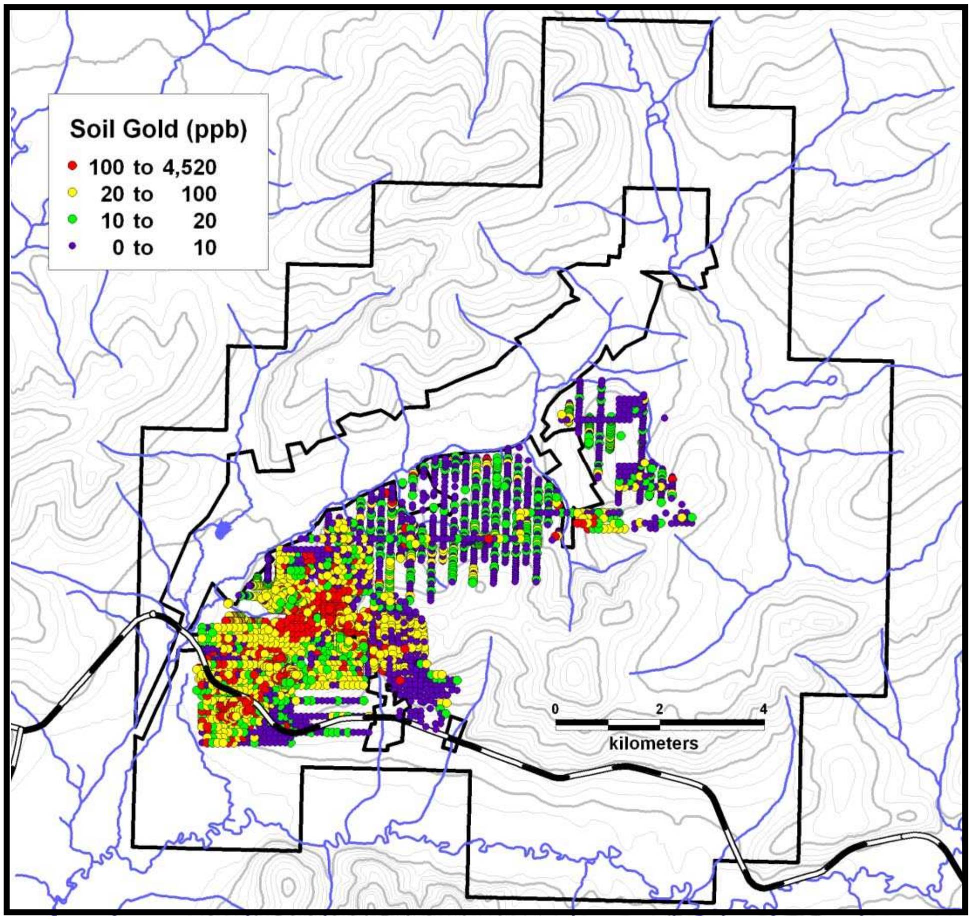

Figure 9.1. Plot of soil samples. Color coding shows relative gold content with red indicating gold ≥0.100 g/t Au.

it from other parts of the deposit. The first is the presence of many thin quartz veins (0.5 to 40 mm) with visible gold and the second is the fact that the rocks are sodium-rich. These aspects are under evaluation by ITH geologic staff.

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^2 and the 0.1 g/t grade shell averages 280m thick and ranges up to 510m 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 and orientation, but more

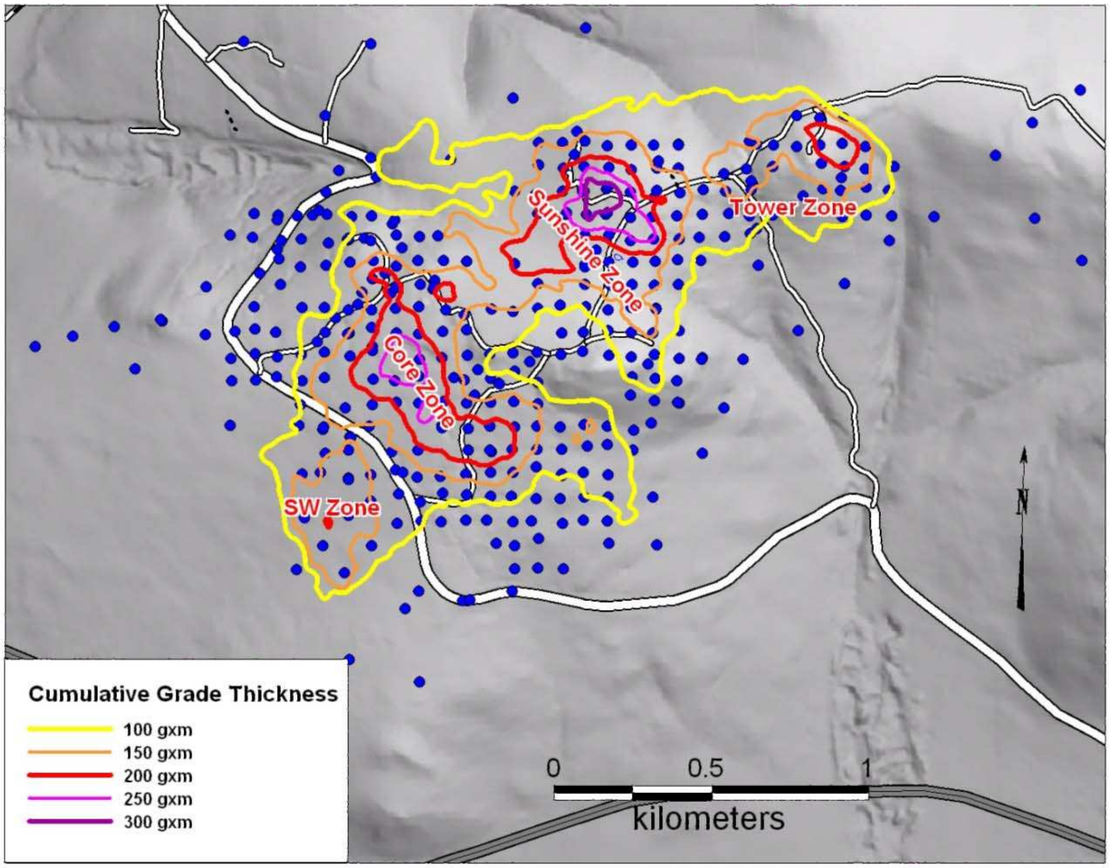

Figure 9.2. Cumulative gold grade thickness (March 2010) for the block model of the Money Knob deposit showing the four main zones within the deposit.

widespread and 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 possible minor pyrophyllite (?) is curious and may be important, but convincing identification has not been made and it is unclear at this time where and how these minerals might fit 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 9.3 and 9.4**). 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 9.3 and 9.4**). 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 locally appears to 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 disintegration 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.

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 the 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 siderite and ankerite. Arsenopyrite and pyrite are common in carbonate-quartz veins and veinlets.

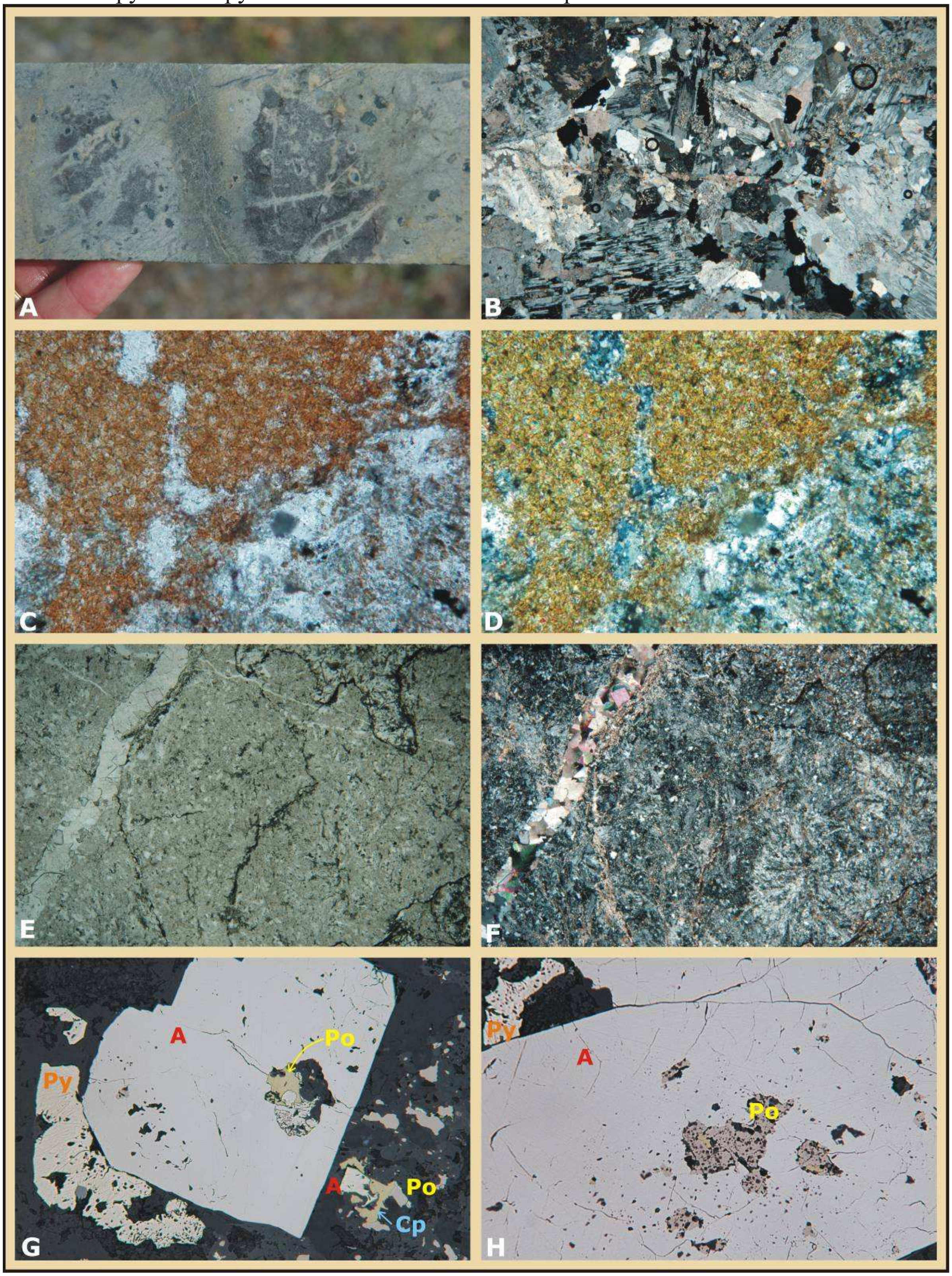

Figure 9.3. Photomicrographs of characteristic alteration among rocks at Money Knob. **A)** View of core showing relict patches of secondary biotite (dark color) cut by and overprinted by albite and sericite alteration. 08-33, 190.25. **B)** rare, relatively weakly altered Cretaceous intrusive dike with 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. **H)** Arsenopyrite grain with contained blebs of pyrrhotite (po) and adjacent pyrite (py).

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 9.5**). 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.

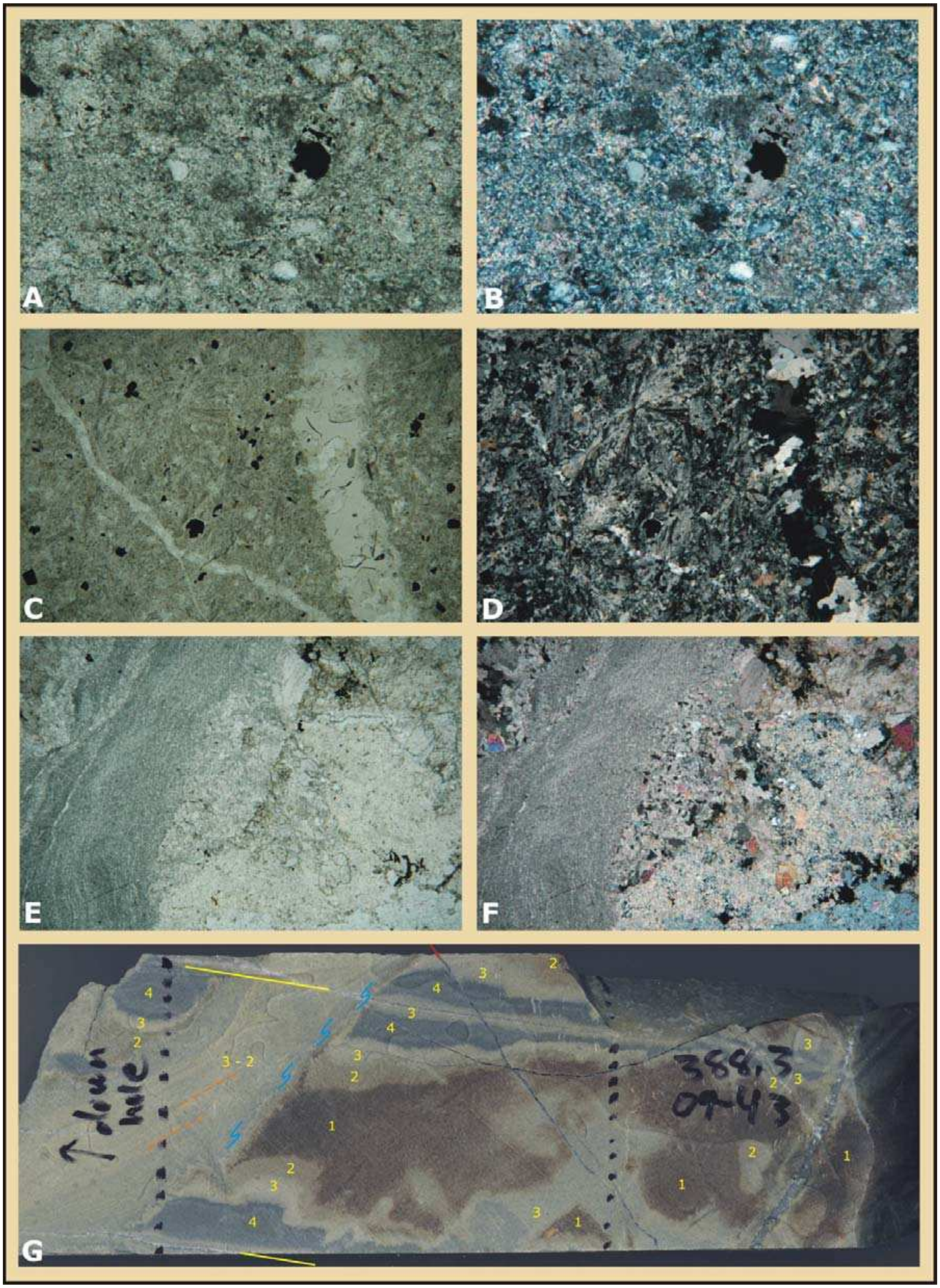

Figure 9.4. Photomicrographs of characteristic alteration among rocks at Money Knob - plain light on the left; crossed polarized light on the right. **A and B)** Sericite and carbonate replace a silty phyllite

(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)** Core showing a complex sequence of alteration types which generally mimic the larger scale assessment of alteration styles. Zone 1 = secondary biotite-carbonate±sericite. Zone 2 = Carbonate-sericite with darker color possibly owing to overprinted secondary biotite. Zone 3 = carbonate-sericite. Zone 4 = sulfide-rich sericite-carbonate. Blue symbol = shear. Orange dashed lines = bedding. The yellow lines indicate quartz-carbonate±sulfides veinlets. Red line indicates quartz-feldspar±carbonate veinlet. From MK09-43, 388.3.

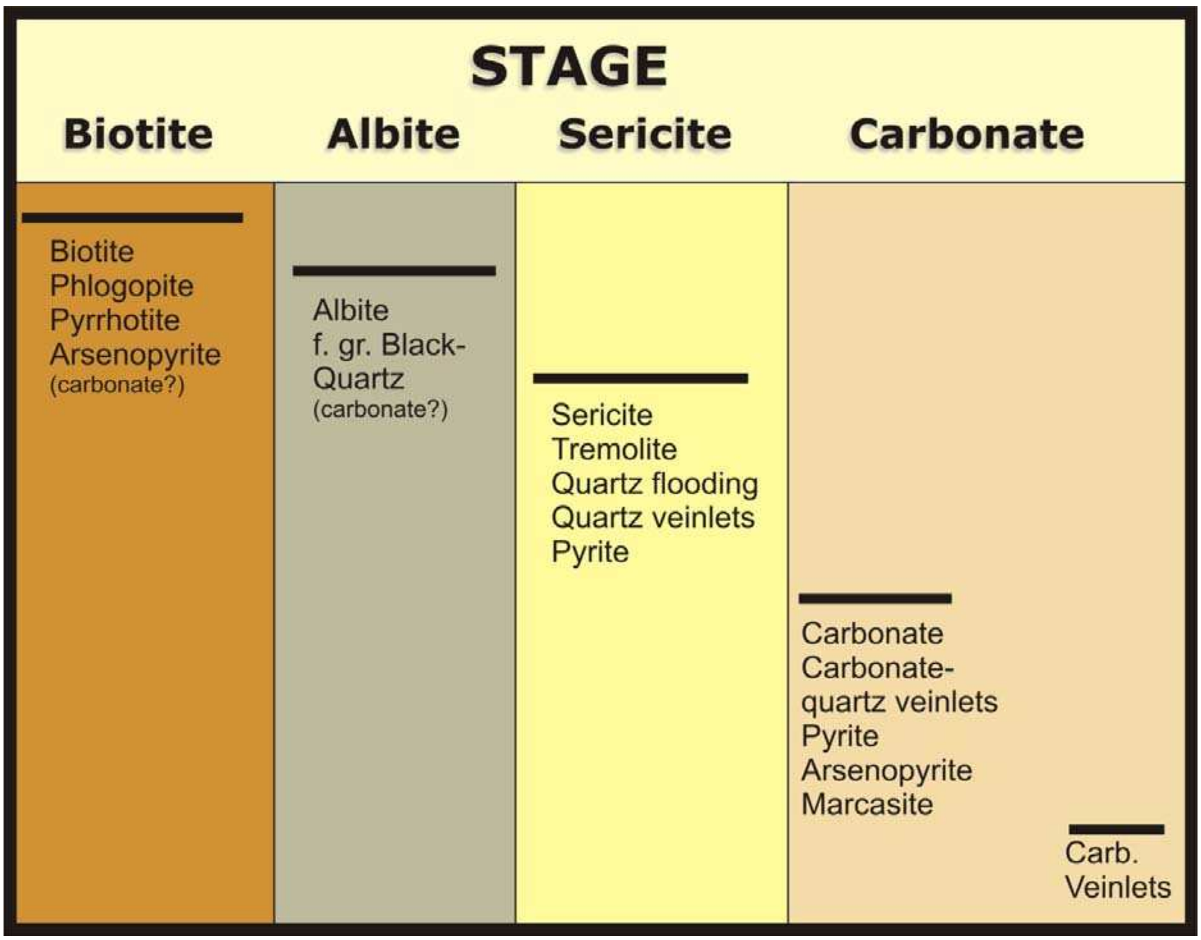

Figure 9.5. Interpreted paragenetic sequence of key alteration and mineralization stages. Gold occurs with arsenopyrite and may have been introduced during all stages or dominantly during a particular stage(s).
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10.0 Exploration

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, et al., 2009a). Data from drill holes completed through February, 2010 were used to complete a new resource estimate along with additional information on possible recovery techniques being contemplated by ITH (Klipfel, et al., 2009b, 2010). The remaining data from drilling completed in the first half of 2010 is the subject of a new resource update and reported in this document.

10.2 Current Exploration

ITH has continued to conduct step-out and infill drilling through 2009 and the first half of 2010. This report includes the results from all results for 2010 drilling as received through May 31, 2010. This data does not include results from the current summer drill program. This data will be utilized in an updated resource assessment to be completed in the first quarter of 2011. This data has been used in a 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. These results are presented in section 18.

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: 1) helped fill in gaps within the drilling grid and enabled increased continuity of information for improved resource estimation, and 2) discovered the Sunshine and Tower Zones, and grid drilled them in the process. 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.

11.2 Current Drilling

The resource estimate presented in this report is based on drilling completed by ITH through May 2010. Further drilling that has been completed this year is not incorporated in to the resource estimate or the PA.

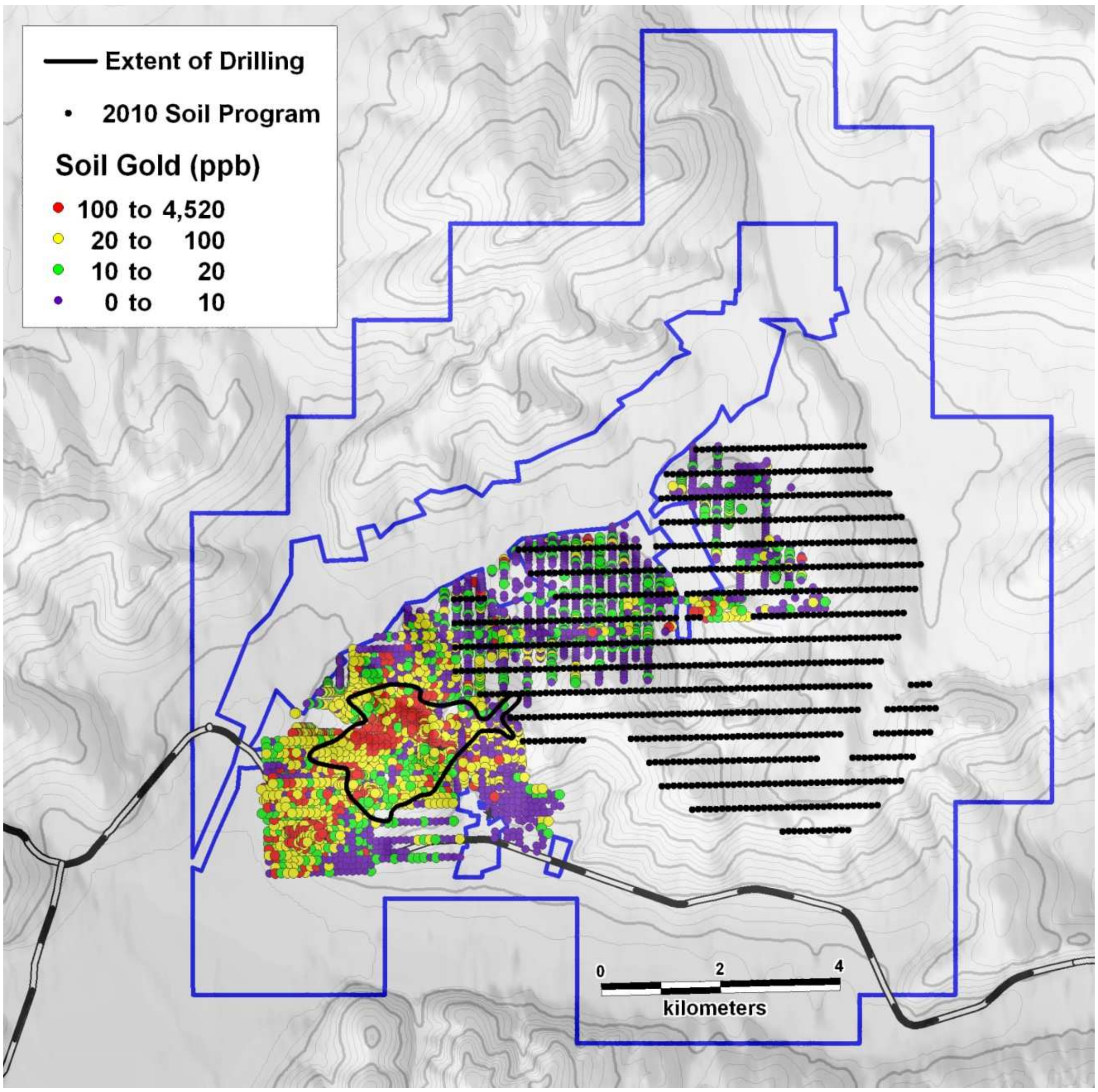

Figure 11.1. Distribution of drilling in the Money Knob area with respect to anomalous soil samples. The majority of the soil geochemical target remains untested. An expanded soil sampling program is proposed for 2010.

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
2009	12	4572	195	59,757	Expanded the extent of the mineralization to include the new Sunshine and Tower Zones and provided input for the resource evaluation reported in March 2010 (9.8 M oz Indicated and Inferred resource).
Winter 2010	5	1,998	50	15,584	Filled in between the Core and Sunshine zones, expanded SW Zone

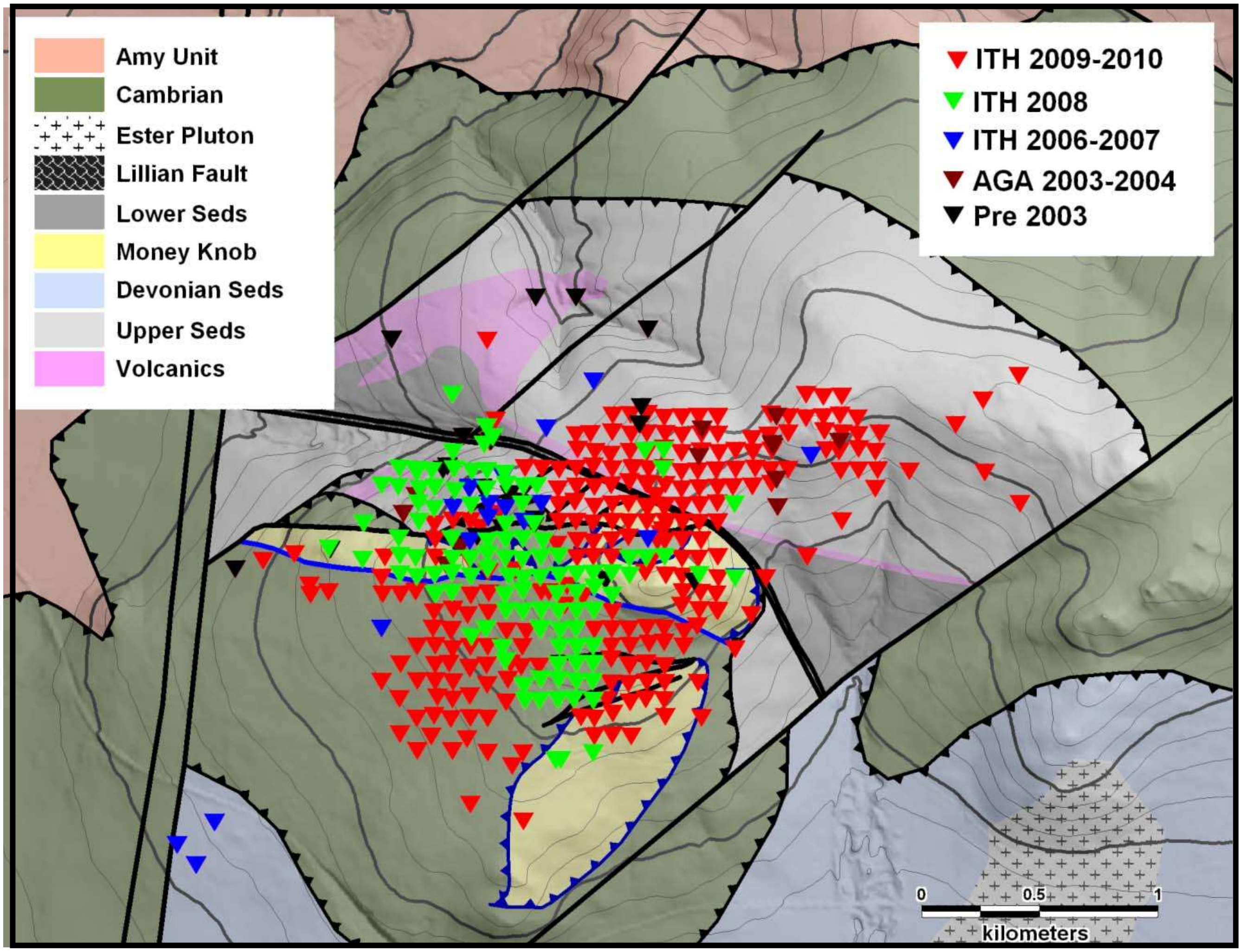

Figure 11.2. Distribution of drilling in the Money Knob area according to year and company.

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.

Currently, core is reviewed, marked for orientation, and ‘quick-logged’ at the drill rig prior to transporting it to ITH’s core shed for logging. This is a relatively new procedure and has been in place since August, 2009. In the past, core would be marked for orientation and then 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 was 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 a 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-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. Mr. Carew has reviewed the data, methodology and results of this analysis and concurs with these conclusions. Drill hole surveys were completed by Northern Associates, Inc. and were observed in the field by a Qualified Person (Klipfel, et al., 2010b).

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., Layne Christensen, and T and J Enterprises.

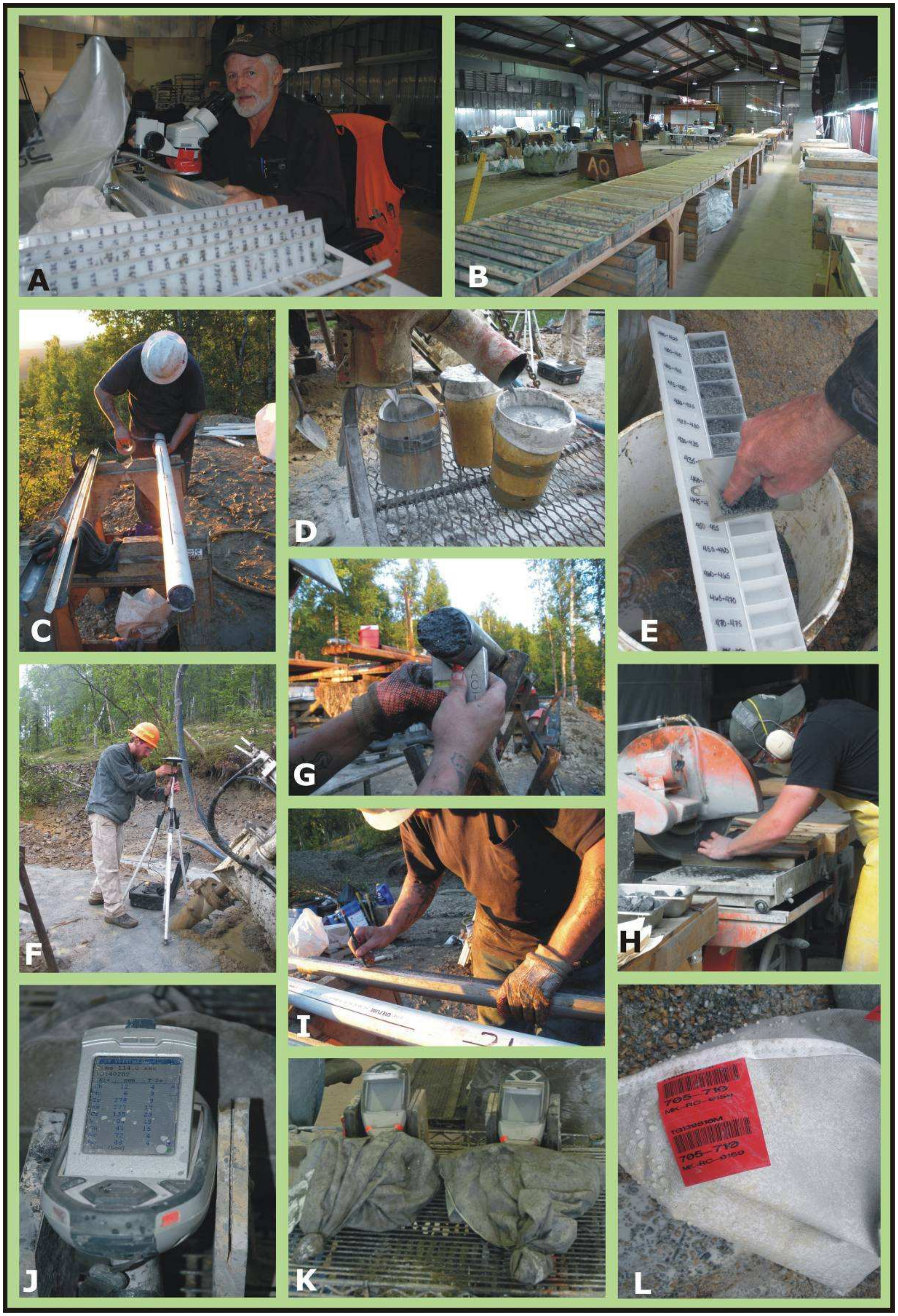
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Figure 11.3. Photos of various exploration functions.

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.

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. These protocols were reviewed in 2006 as well as AGA's security procedures and verified that they met or exceeded standard industry practices (Klipfel, et al., 2010b). 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. Examination of the core logs and core from the four 2004 holes verified the reliability of the logging (Klipfel, et al.,2010b). 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. This core was examined by a QP (Dr. Paul Klipfel) in the course of 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.

Since June of 2009, core is examined by a geologist in the original split tube, the soft structures are documented, then the core is boxed and transported to the core shed for detailed logging, mark-up and sampling. For the 2008 program, core was 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 was effective and minimized disturbance to the core, prevented unnecessary breakage, and minimized 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 his 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 (**Figure 13.1**). 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.

All core and RC samples are taken from the drill rig directly to ITH's core shed. RC and core samples are placed in super sacks, sealed, and palleted for shipment to ALS Minerals' preparation facility in Fairbanks.

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 has developed a set of decision criteria that compare the NITON-measured 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 master project database is maintained in Microsoft™ Access 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.

Results of technical studies being performed at Livengood, and which will form the basis of the pre-feasibility studies are maintained in a data hierarchy on ITH servers located at the Denver Office. Tape backup of the data is conducted nightly, with rotation of tapes into offsite storage.

The author, Mr. Carew, has reviewed these procedures and observed the data entry process at various steps during site visits. He is satisfied that ITH is diligent in their data management

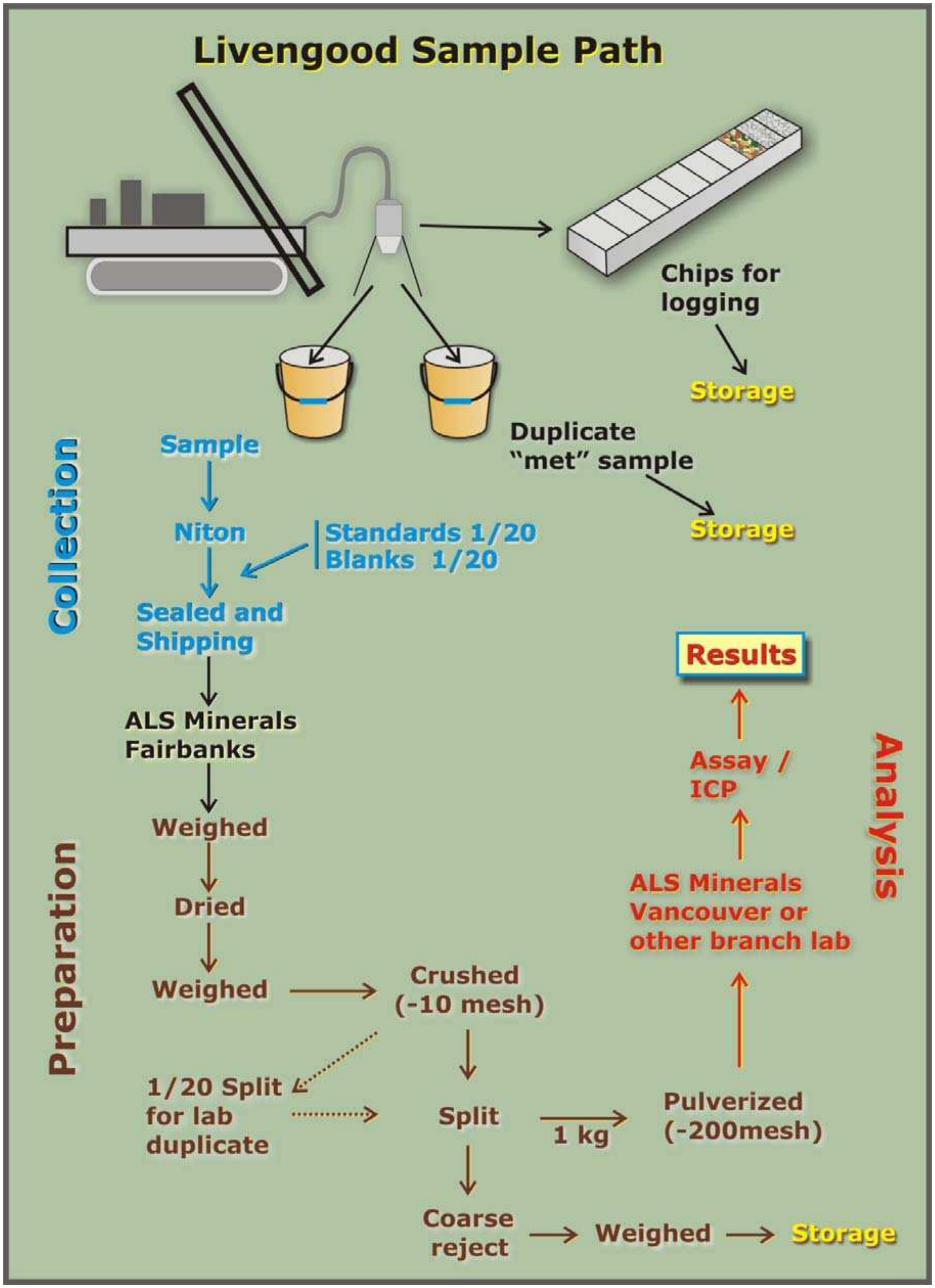


Figure 13.1. This diagram shows the flow path and steps involved for RC samples from the drill rig to analytical results.

procedures and have check procedures in place that should identify any issues. He has not completed a thorough check or validation of the master project database but is not aware of any issues. Mr. Carew has, however, conducted a data validation check on a random sample (10%) of the subset of drillhole data provided for resource modeling, as described in Section 17.1

13.4 Quality Assurance and Quality Control

The QA/QC data from ITH sampling program has been reviewed by Mr. Carew. 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 (394 from 2007 to 2010) represent equivalent samples collected at the drill rig during the original sampling process and confirm that the sampling process is representative (**Figure 13.2a**). Prep duplicates (3455 from 2007 to 2010) 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 (**Figure 13.1**). These duplicates are designed to assess sample homogeneity and confirm that no bias is created during the sample preparation process (**Figure 13.2b**). Pulp duplicates (388 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 13.2c**). 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. This issue has been evaluated carefully and it is believed to be the result of normal nugget effect where a grain of relatively coarse gold ends up in one 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 13.3**). 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 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 (3455 from 2007 to 2010), 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=0.92, 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

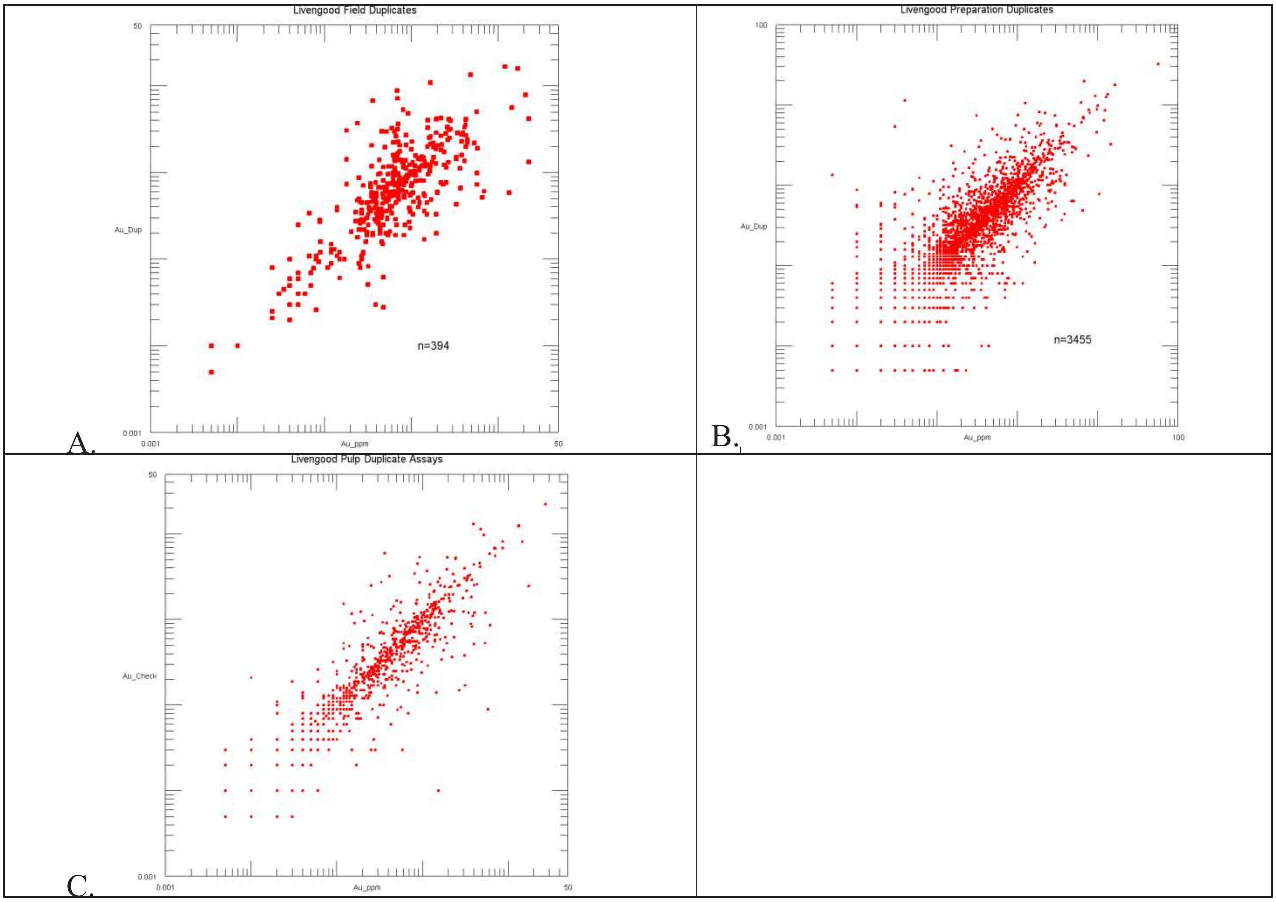


Figure 13.2. These scattergram plots show how different categories of sample duplicates compare with original sample results. The diagonal line has a slope of 1. Perfect duplication of results would plot on this line. Variation and scatter is interpreted to be the product of normal nugget effect. **A)** 2007-2010 field duplicate vs. original samples; n= 394. The envelope of points flares with increasing grade. This is typical of nugget effect which becomes more pronounced at higher grades. **B)** 2007-2010 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 these samples. **C)** 2007-2010 pulp duplicates vs. original sample. Scatter is similar to that in B.

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. Mr. Carew considers these results to be appropriate for Livengood mineralization and indicative of sound QA/QC procedures.

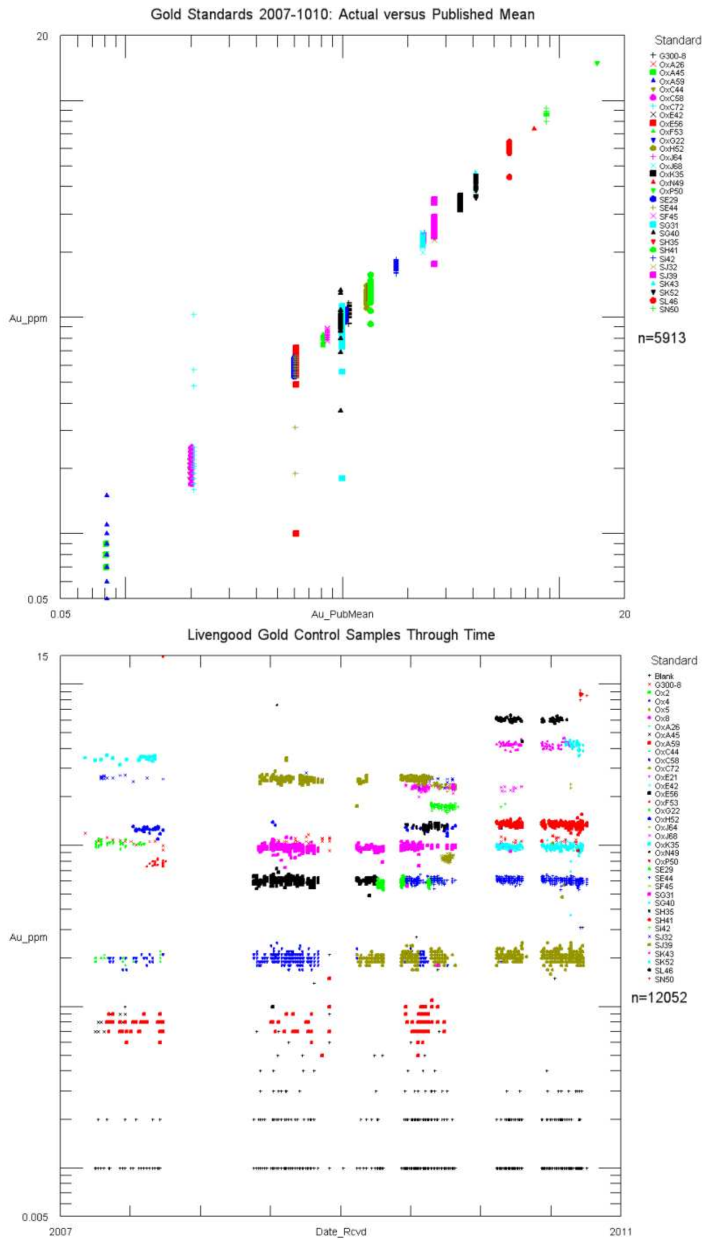


Figure 13.3. X-Y scattergrams for 2007 - 2010 showing the stated value of standards vs. the measured value by the lab: **Top)** values are plotted according to measured value vs. stated values of a standard placed in the sample stream. **Bottom)** values plotted as a function of time to check for drift in results over time. The horizontal nature of the points for each value indicates that drift is minimal.

Mr. Carew has visited the ALS Minerals preparation facility in Fairbanks to verify sample handling procedures and concludes that the lab follows sound log-in, weighing, drying, and splitting procedures. Sample crushing, splitting, and pulverization is done by modern equipment with diligent air cleaning between samples and cleaning with blank material between runs and at the beginning of the day.

Handling techniques demonstrate care in assuring that bags and samples are not mixed up. All pulps are sealed in paper envelopes and placed in boxes, packaged and sealed for transport to the Vancouver or Reno labs for analysis.

14.0 Data Verification

Field and drill core observations made by Mr. Carew during site visits are consistent with the style of mineralization and alteration interpreted and reported in ITH documents. Outcrop exposures in road cuts 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. Paul 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 addition, Dr. Klipfel witnessed the sluicing and panning of concentrated "clean up" material shovelled from a trench face. 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, (Klipfel, et al., 2010b).

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, (Klipfel, et al., 2010b).

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, (Klipfel, et al., 2010b).

As a check of the data generated during 2009, Dr. Klipfel selected two batches of samples. The first batch consisted of 28 samples selected from the duplicates collected by ITH from the winter program. The second batch consists of 13 duplicate RC chip samples randomly selected at each of the three RC drill rigs. Samples of the first batch were selected to be representative of a range of rock type and gold values from different holes.

Results for the first batch 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).

Results for the second batch show good correlation and do not display any discernible bias (**Figure 14.2**). 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.

Mr. Carew has reviewed the results of the 2009 verification sampling and agrees with the conclusions regarding accuracy, precision and lack of bias. Mr. Carew also collected a batch of samples from the later 2010 drilling for verification purposes during his site visit from October 24-27, 2010. The results of these samples are pending. Mr. Carew has not verified all sample types or material reported. To the best of his knowledge, ITH has been diligent in their sampling procedures and efforts to maintain accurate and reliable results.

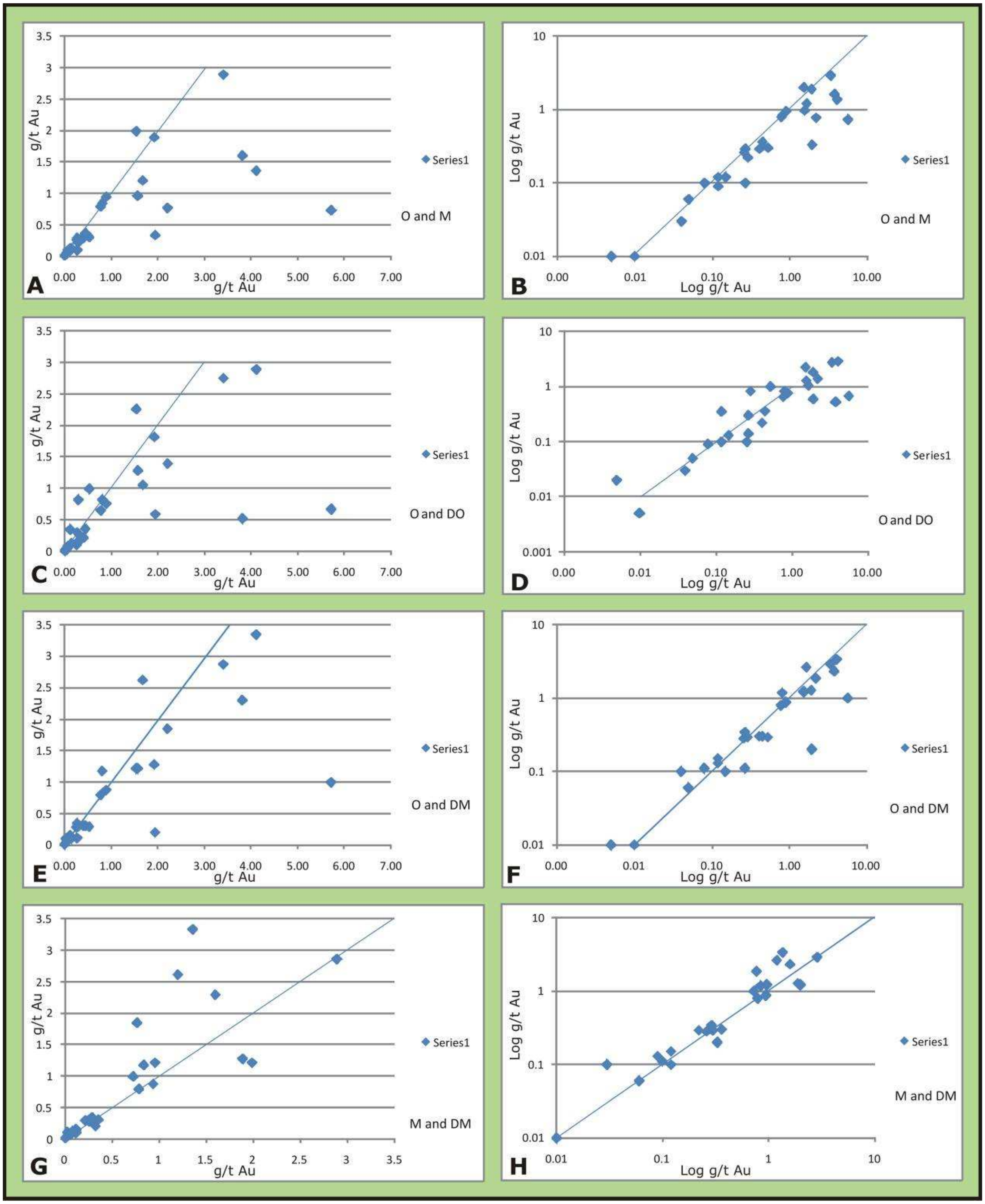


Figure 14.1. X-Y scatter plots of 2008 and early 2009 original and duplicate sample data for check samples collected by Dr. Klipfel as part of data validation procedures. The diagrams on the left are plotted with numeric scales. The diagrams on the right are plotted with log-log scales. The scatter increases 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.

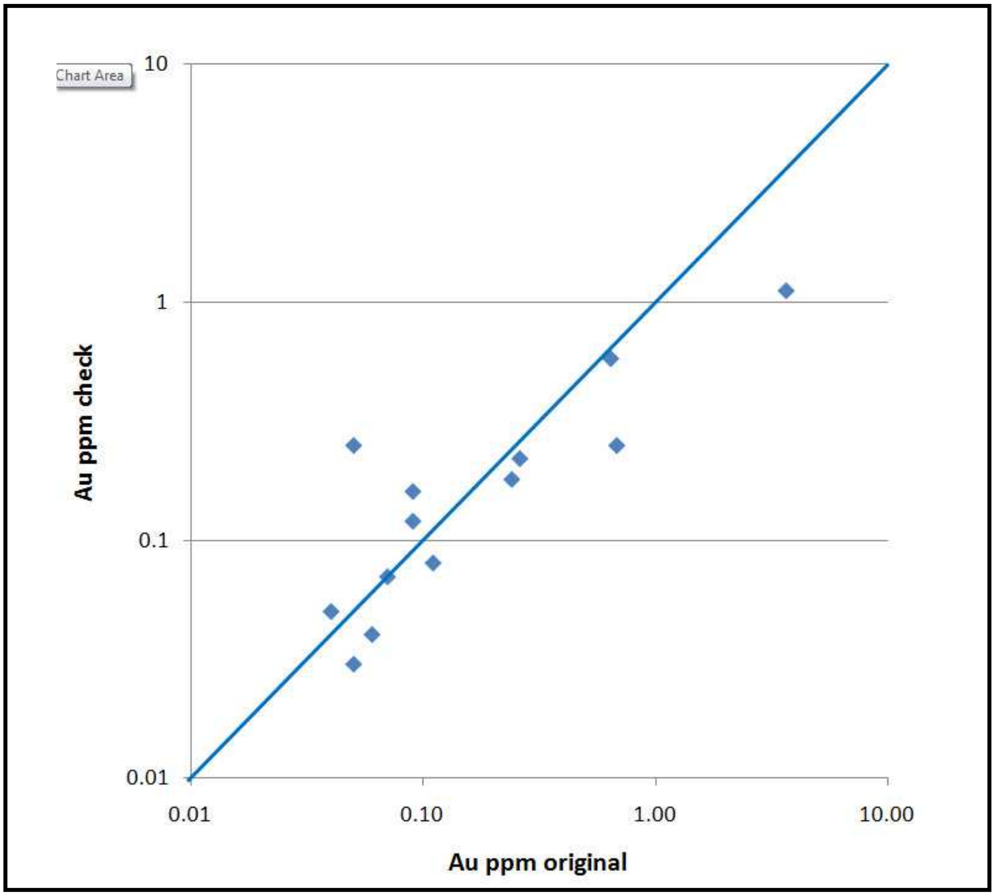


Figure 14.2. X-Y scatter plot of original and check samples for June 2009 RC drilling. The correlation line shows a slope of 1. Samples with identical results will plot on the line. Deviation of results from the line is interpreted to be the result of normal variation and nugget effect.

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 some combination of heap leach, mill with Carbon in Leach (CIL) and gravity or flotation separation techniques. Current test work focuses on determining the best means of optimizing these combined recovery methods. This work involves studies that evaluate how mineralization is characterized and how the mineralized materials vary in their physical and metallurgical response to process treatment parameters by type according to the various lithologic units that host mineralization. The characteristics under review include grindability, abrasiveness, optimal particle size for downstream treatment, and response to leach, flotation, or gravity process parameters as a function of oxidation and lithology. In addition, the right combination of these techniques for different mineralization types is being evaluated.

The information presented here derives from on-going studies which are in progress. In the previous **PA** developed in October, 2009 (Klipfel, et al., 2009b), results from leach tests were applied to mineralization that is amenable to heap leach processing. Those results have been updated with results of column leach tests received in 2010. Although ITH has envisioned that Livengood gold would be recovered through a combination of processes, test work for a mill with Carbon in Leach (CIL) and gravity or flotation techniques had not been completed at that time and was not used in the Whittle Pit estimation. Test work continues and is still in progress. Results received since then are presented here but are not final. On-going work will support future evaluations.

Specific metallurgical characteristics, identified in the testing programs to date, have shaped the processing strategies used as the basis for the PA and assumed project configurations. These important metallurgical findings are:

1) variable metallurgy (chemical and physical properties), depending upon mineralization type; degree of oxidation, amount of organic carbon, etc.;
2) identification of mineralization types that are highly amenable to simple cyanide leaching process techniques like heap leaching with a carbon in column adsorption plant (CIC), particularly oxidized and partially oxidized mineralization;
3) identification of some sediment-hosted mineralization that contains organic “preg-robbing” carbon that will require CIL techniques, gravity or flotation techniques; and
4) high recoveries for some mineralization types using gravity and flotation separation techniques.

Test work completed or currently in progress includes grind ability, abrasiveness, optimal particle size for downstream treatment, and response to flotation or gravity concentration followed by cyanide leaching of the concentrates as a function of oxidation and lithology. Power requirements and reagent consumptions for each mineralization type for each process scenario are also being developed from test work data. This information will be used as inputs into process operating costs for each mineralization type in future estimates.

16.2 Metallurgical Summary

Metallurgical test work programs on the Livengood mineralization began in 2004 and continue as of the preparation of this report. The mineralization 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 mineralization could be considered moderately soft to medium hard in hardness with an average Bond Ball Work index of 15.8. The mineralization 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 mineralization would be considered non-abrasive, with an average Abrasion index of 0.0809. The mineralization type abrasion characteristics varied significantly from 0.0023 to 0.2872.
- All of the Livengood mineralization types respond to cyanide leaching to some degree.
- Some of the unoxidized mineralization with organic carbon has “active” or “preg-robbing” carbon.
- The effect of leach times on gold recovery and gravity concentration results indicate some of the mineralization contains coarse gold.
- Gold recovery at 10 mesh particle sizes on some of the mineralization types exceeded 90 percent.
- Gold recovery on some of the mineralization 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.
- Initial flotation and gravity concentration tests indicate the combined processes exceed 90% gold recovery to the concentrates.
- The degree of oxidation of the mineralization, as observed by the geologists, 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 mineralization is active to varying degrees in some of the mineralization types, particularly the un-oxidized version of those mineralization types.
- The gold is often associated with sulfides, but this mineralization would not be classified as a sulfide refractory type.

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

It became evident early in the test work that the oxidized to partially oxidized mineralization responded well to cyanide leaching while other un-oxidized mineralization 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 mineralization types do respond to cyanidation to some degree.

The most significant metallurgical parameter for Livengood un-oxidized mineralization 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 mineralization. The simplest of these methods, the Carbon-in-Leach (CIL) process, has been the focus of test work since October, 2009, and is currently being used in test work performed in the latest round of tests. The CIL test work, although incomplete, is showing positive results in counteracting the effects of preg-robbing carbon, providing an average increase of gold recovery compared to standard cyanide leaching for all mineralization types of approximately 18 percent and as high as a 49.5 percent increase in gold recovery for the more difficult un-oxidized mineralization.

In addition, gravity concentration testing of the Livengood mineralization shows encouraging results with 58% of the gold reporting to the gravity concentrates. The results show a 69-1 concentration ratio (gravity concentrate weight percent of 1.43%) provides an average concentrate grade of 46.1 g/t Au. Test work on the gravity concentrate to establish the viability of ultra-fine grinding and high intensity cyanide leaching of the concentrate indicate that total recoveries of gold in the main rock units range between 83% and 92%. Lower total recoveries were observed in the unoxidized KINT units (59%-62%), however, these rocks form a minor portion of the total mineralization.

From the data currently at hand the oxide and partially oxidized mineralization types will respond well to heap leaching. Ongoing test work indicates higher gold recoveries can be obtained from all mineralization types and particularly the weakly to un-oxidized types 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 weakly-oxidized to un-oxidized mineralization has the potential to significantly improve the Livengood project in both its size and economic performance.

Metallurgical test work currently underway and / or 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.

Initial batch flotation test work has been performed to determine the potential for concentrating the gold and depressing gold preg-robbers prior to downstream cyanidation. In these tests, flotation was followed by gravity recoverable gold tests. The test results indicated that flotation would recover between 57.7% and 97.9% of the total gold, and that gravity recovery on the flotation tails would recover an additional 7.5%-93.3% of the gold reporting to the gravity circuit. Total gold recovery to the combined concentrates was relatively high and ranged between 76.4%-99.9% with an overall average of 90.1%. Further testing is ongoing to evaluate the CIL / high intensity cyanide leach recovery from the flotation and gravity concentrates.

Column leach test work was performed by McClelland Laboratories of Reno Nevada in 2010, on a ½ inch crush of mineralization types that do not show preg- robbing tendencies in order to establish the effectiveness of heap leaching as a process option. **Table 16.1** provides the results of these tests.

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

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

TABLE 16.1
SUMMARY METALLURGICAL RESULTS, COLUMN PERCOLATION LEACH TESTS, LIVENGOOD DRILL CORE COMPOSITES, 80%-12.5MM FEED SIZE

ML1 Composite	Test No.	Leach/Rinse Time, days	Au Rec. %	Tail Screen Assay	Calc'd. Head	Average Head	NaCN Consumed, kg/mt feed	Lime Added kg/mt feed
Comp #1 Cambrian-Partial Ox	P1	114	79.1	0.39	1.87	1.98	4.52	3
Comp #1 Cambrian-Partial Ox	P2	114	96.8	0.06	1.87	1.98	5.16	3
Comp #2 Cambrian-Trace Ox	P3	114	50.4	0.58	1.17	1.99	3.42	1.5
Comp #2 Cambrian-Trace Ox	P4	111	40.4	0.81	1.36	1.99	3.96	1.5
Comp #3 Cambrian-No Ox	P5	81	42.4	0.34	0.59	0.64	2.52	3.5
Comp #3 Cambrian-No Ox	P6	80	60.6	0.13	0.33	0.64	2.38	3.5
Comp #4 Upper Seds-Partial Ox	P7	116	72.3	0.13	0.47	0.4	3.25	2
Comp #4 Upper Seds-Partial Ox	P8	116	81.1	0.1	0.53	0.4	3.1	2
Comp #5 Upper Seds-Trace Ox	P9	139	58.7	0.43	1.04	1.14	2.74	3
Comp #5 Upper Seds-Trace Ox	P10	116	58.6	0.53	1.28	1.14	3.49	3
Comp #6 Upper Seds-No Ox	P11	116	56.3	0.45	1.03	1.08	3.28	2
Comp #6 Upper Seds-No Ox	P12	113	50	0.57	1.14	1.08	2.99	2
Comp #7 Main Volcanics-Partial Ox	P13	113	44.2	0.48	0.86	0.86	3.92	5
Comp #7 Main Volcanics-Partial Ox	P14	113	44.8	0.48	0.87	0.86	3.8	5

ML1 Composite	Test No.	Leach/Rinse Time, days	Au Rec. %	Tail Screen Assay	Calc'd. Head	Average Head	NaCN Consumed, kg/mt feed	Lime Added kg/mt feed
Comp #8 Main Volcanics-Trace Ox	P15	112	33.3	0.8	1.2	1.3	3.52	1.5
Comp #8 Main Volcanics-Trace Ox	P16	113	31.9	0.79	1.16	1.3	2.84	1.5
Comp #9 Main Volcanics-No Ox	P17	112	28.8	0.57	0.8	1.36	3.75	3.5
Comp #9 Main Volcanics-No Ox	P18	112	19.8	0.69	0.86	1.36	4.05	3.5

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. letter report dated February 7, 2007, Subject: Characterization of Livengood Gold Ore, Hazen Project 10504).

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 mineralization 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 16.2,** 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 16.2**) and somewhat less in the sulphide material. Two of the sulphide samples (**Table 16.2**, samples 3 and 1A) were from rock with albitic alteration and they each returned 60% cyanide recovery. The 3rd sulphide sample (**Table 16.2**, 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.

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 16.3**). This was undertaken as a separate study from a previous one with Chemex. Results indicate that overall average cyanide

TABLE 16.2
GOLD RECOVERY FROM 2007 CYANIDE EXTRACTION TESTS

Sample #	Mineralization 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.*

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 16.4**.

TABLE 16.3
GOLD RECOVERY FROM 2008 HAZEN CYANIDE EXTRACTION TESTS
(-10 MESH)

Sample ID	Mineralization 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%

Sample ID	Mineralization 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
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%
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 16.4
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 on NaCN, kg/t	Addition Ca(OH)2, kg/t
100112113	0.459	0.39	0.073	84.10%	3	1.1	2.75
100123124	0.609	0.47	0.144	76.40%	3	0.45	1
100588589	1.686	1.23	0.461	72.70%	3	0.53	2

Sample ID	Calculated Head, Au g/t	Extracted, Au g/t	Avg. Tails, Au g/t	Au Extracted, %	Leach Time, days	Consumption on NaCN, kg/t	Addition Ca(OH)2, kg/t
100772773	0.728	0.51	0.221	69.60%	3	2.01	2.75
100829830	1.278	1.06	0.221	82.70%	3	0.55	2.5
101024026	0.62	0.54	0.077	87.60%	3	0.66	2.25
101273274	0.787	0.68	0.105	86.70%	3	0.51	1.5
101291292	1.333	1.21	0.125	90.60%	3	0.81	1
101437438	0.819	0.57	0.247	69.80%	3	0.48	1.5
101548549	2.67	2.51	0.162	93.90%	3	0.22	1.5
101604605	0.992	0.83	0.166	83.20%	3	0.37	1.5
101618619	1.434	1.15	0.28	80.50%	3	0.82	2.5
101774775	1.069	1	0.068	93.70%	3	0.56	1.5
101827829	2.733	2.67	0.063	97.70%	3	0.66	1.5
101847849	1.279	0.75	0.525	59.00%	3	0.48	1.5
101896897	1.269	0.52	0.747	41.10%	3	0.79	1.5
101925926	1.552	1	0.555	64.20%	3	0.12	1.5
102070071	0.594	0.52	0.077	87.00%	3	0.72	2
102096097	1.074	0.96	0.117	89.10%	3	0.57	1.5
102536537	0.875	0.84	0.034	96.10%	3	0.69	2
102575576	0.927	0.87	0.053	94.30%	3	0.71	1.5
102642643	0.596	0.48	0.12	79.90%	3	2.49	4
102886887	0.873	0.36	0.51	41.60%	3	1.28	4
103110111	0.711	0.6	0.11	84.60%	3	0.94	2.5
Average	**1.124**	**0.9**	**0.219**	**79.40%**	**--**	**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 mineralization 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 16.4.**

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 chemical and physical characteristics of the mineralization, and the effectiveness of gravity and cyanidation for gold recovery.

Other test work currently in progress 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 (February 2010) was performed at Kappes Cassidy and Associates (KCA) in Reno, Nevada, on Livengood mineralized samples. KCA has also contracted with ALS Minerals to perform ICP analyses of the composites, and Phillips Enterprises LLC to perform grinding and abrasion studies. Results from this program have been compiled and are included in this report.

Initially, thirty-five test composites were sorted and provided to the laboratory for testing. The samples represent eight different stratigraphic units with distinct silicate mineral assemblages. Samples from each stratigraphic unit were selected to represent variations in grade and degree of surficial oxidation. Samples that make up the composites were sorted on site into 35 bins with an average weight of 200 kilograms. These bins were shipped directly to the KCA laboratory in Reno, NV.

More recently, an additional 8 composites were sorted from recent drilling of the Sunshine Zone. These composites were similar to two of the stratigraphic units previously supplied to KCA, Upper Sediments and Kint, but were from a new mineralized zone.

When the samples arrived at the lab, they were identified by composite, 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 mineralization 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 for Livengood mineralization follows the diagram.

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

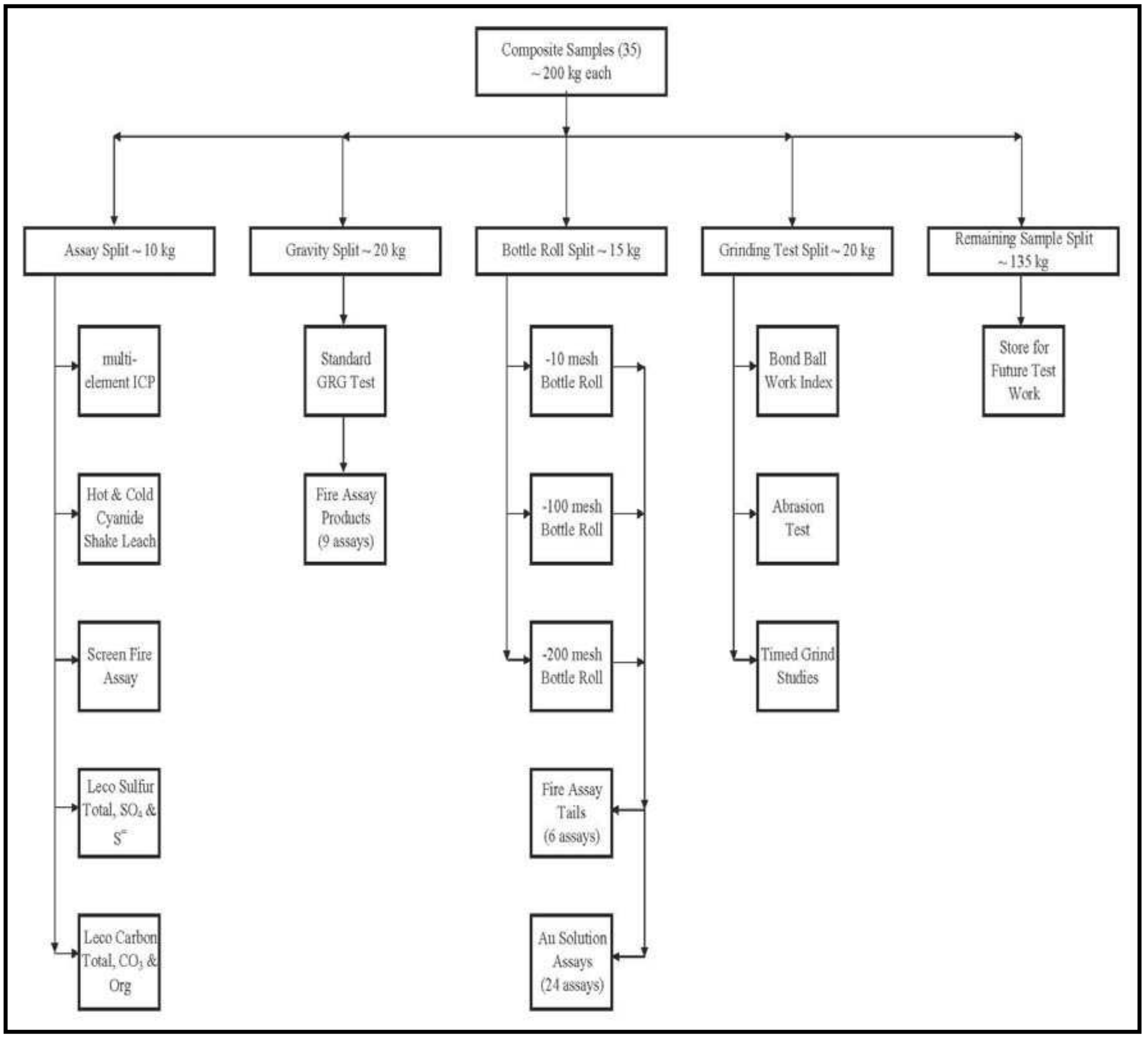


Figure 16.1. Flow chart and breakdown of Livengood composite sample test work.

Using this methodology the total number of composite samples comes to 54. However, some of the composites selected were volumetrically insignificant in the deposit and therefore the total number of composites submitted totaled 41.

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 Minerals (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 conditions listed in **Table 16.5**.

After leaching the samples they were centrifuged and the solution removed for Au assay by atomic absorption spectrometry. Assays were performed in triplicate.

As indicated by the test results, (**Table 16.6 and 16.7)** the response of the Livengood mineralization to the CN shake leach test procedure for determining gold leachability was poor. The poor results were later found to be linked to "active" organic carbon in some of the mineralization, slow leaching gold mineralization, 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.

TABLE 16.5
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 16.6
LIVENGOOD PROJECT - MAIN ZONE SUMMARY OF CYANIDE SHAKE TESTS (5 GPL NACN)

Description	Average Head Assay, Au g/t	Average Met Screen, Au g/t	Overall Average Head, Au g/t	Avg. Cyanide Sol.(22 °C), Au g/t	Avg. Cyanide Sol.(60 °C), Au g/t
Overburden: Partial Ox (L)	0.82	0.59	0.71	0.27	0.3
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)	0.8	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	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	1.1	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	0.1
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	0.2
Upper Seds: Partial Ox (H)	1.3	1.4	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	0.77	1.14	0.95	0.05	0.01

Description	Average Head Assay, Au g/t	Average Met Screen, Au g/t	Overall Average Head, Au g/t	Avg. Cyanide Sol.(22 °C), Au g/t	Avg. Cyanide Sol.(60 °C), Au g/t
(L)					
Upper Seds: No Ox (H)	2.77	0.99	1.88	0.06	0.03
Lower Sand: Partial Ox (L)	0.8	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	0.7	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	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
Average			**1.12**	**0.1**	**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

TABLE 16.7
LIVENGOOD PROJECT – SUNSHINE ZONE SUMMARY OF CYANIDE SHAKE TESTS (5 GPL NACN)

Description	Average Head Assay, Au g/t	Average Met Screen, Au g/t	Overall Average Head, Au g/t	Average Cyanide Sol. (22 °C), Au g/t	Average Cyanide Sol. (60 °C), Au g/t
Kint: Ox_high	2.25	1.51	1.88	0.35	0.51
Kint: Ox_Low	0.59	1.03	0.81	0.17	0.27
Kint:TraceOx_High	1.34	1.44	1.39	0.22	0.21
Kint: TraceOx_Low	1.24	0.81	1.02	0.13	0.22
Upper Seds: Ox_High	0.77	1.50	1.13	0.24	0.41
Upper Seds: Ox_Low	2.38	0.99	1.68	0.15	0.25
Upper Seds: Trace_High	1.32	1.60	1.46	0.18	0.25
Upper Seds: Trace_Low	0.63	0.84	0.74	0.09	0.19
Average			**1.26**	**0.19**	**0.29**

A total of 43 composites were tested to achieve a work index for each of the mineralization types. **Tables 16.8 and 16.9** provide the results of the Bond Ball Work Index tests for rock from the Main Zone and the Sunshine Zone respectively. 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.

TABLE 16.8
LIVENGOOD PROJECT – MAIN ZONE 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	12.12	13.2	14.55
Kint: Partial Ox (L)	11.25	12.41	13.5	14.89
Kint: Partial Ox (H)	11.8	13.01	14.16	15.61
Kint: Trace Ox (L)	13.2	14.55	15.83	17.46
Kint: Trace Ox (H)	13.06	14.4	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	14.7	16	17.64
Lower Seds: Trace Ox (H)	13.09	14.43	15.7	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)	13.2	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	13.8	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 16.9
LIVENGOOD PROJECT – SUNSHINE ZONE 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
Kint_Ox_high	11.89	13.11	14.26	15.73
Kint_Ox_Low	12.12	13.36	14.54	16.03
Kint_TraceOx_High	12.92	14.24	15.50	17.09
Kint_TraceOx_Low	12.69	13.99	15.23	16.79
US_Ox_High	12.05	13.28	14.46	15.94
US_Ox_Low	12.67	13.97	15.20	16.76
US_Trace_High	12.96	14.29	15.55	17.15
US_Trace_Low	13.00	14.33	15.59	17.19
Average	**12.54**	**13.82**	**15.04**	**16.59**

Fifteen core samples from the Livengood property were obtained for abrasion tests. The results are shown in **Table 16.10.** The abrasion data indicates that the Livengood mineralized material varies from being medium abrasive (Ai of 0.30) to relatively non-abrasive (Ai less than 0.10).

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 16.11** summarizes the test procedure.

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.

Results from this test work for the Main and Sunshine Zones is shown in **Table 16.12** and **16.13** respectively. The gold in the Livengood mineralization appears to respond well to gravity separation.

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

TABLE 16.10
LIVENGOOD PROJECT REPORT SUMMARY OF RESULTS ABRASION TEST RESULTS
PHILLIPS REPORT 093029_15 OCTOBER 2009

Description	Rock Type	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	0.012
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	0.204
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
Cambrian: Trace Ox	Gabbro	Clay Mica	0.0118

TABLE 16.11
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.*

TABLE 16.12
LIVENGOOD PROJECT – MAIN ZONE 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, Au g/t	Conc + Mid Rec % Ag
Overburden: Partial Ox (L)	0.55	1.30%	20.79	49.60%	14.3	7.30%
Cambrian: Partial Ox (L)	0.62	1.50%	28.16	66.40%	13.8	9.10%
Cambrian: Partial Ox (H)	1.34	1.30%	76.95	76.20%	15.6	10.90%
Cambrian: Trace Ox (L)	0.63	1.50%	29.88	69.80%	9.1	6.20%
Cambrian: Trace Ox (H)	1.59	1.50%	89.01	82.00%	13.8	10.70%
Kint: Partial Ox (L)	0.8	1.50%	19.07	35.60%	5.5	3.40%
Kint: Partial Ox (H)	1.67	1.50%	36.9	33.10%	10.2	5.40%
Kint: Trace Ox (L)	0.96	1.50%	25.72	40.60%	7.6	4.10%
Kint: Trace Ox (H)	1.41	1.50%	41.93	45.10%	7.2	5.10%
Kint: No Ox (L)	0.77	1.50%	15.77	31.10%	4.5	2.80%
Kint: No Ox (H)	1.4	1.50%	37.31	40.90%	5.6	2.80%
Lower Seds: Trace Ox (L)	1.12	1.50%	38.88	52.30%	5.8	3.20%
Lower Seds: Trace Ox (H)	1.21	1.50%	40.88	51.70%	12	8.30%
Lower Seds: No Ox (L)	0.75	1.50%	32.79	63.90%	6.3	4.40%
Lower Seds: No Ox (H)	1.21	1.50%	55.36	67.20%	9.5	6.50%
Main Volcanics: Partial Ox (L)	0.79	1.40%	22.37	40.10%	7.2	4.80%
Main Volcanics: Partial Ox (H)	1.8	1.40%	87.75	70.20%	12.9	9.90%
Main Volcanics: Trace Ox (L)	0.9	1.50%	23.83	38.70%	5.5	2.90%
Main Volcanics: Trace Ox (H)	1.65	1.50%	54.1	48.30%	7.9	4.20%
Main Volcanics: No Ox (L)	0.86	1.50%	23.2	40.10%	4.1	3.00%
Main Volcanics: No Ox (H)	1.84	1.50%	52.81	43.50%	6.2	3.40%
Upper Seds: Partial Ox (L)	0.84	1.40%	30.7	50.40%	6.6	6.30%
Upper Seds: Partial Ox (H)	1.42	1.40%	57.79	58.90%	8.9	7.00%
Upper Seds: Trace Ox (L)	0.8	1.40%	36.33	63.50%	8.2	4.60%
Upper Seds: Trace Ox (H)	1.42	1.40%	73.57	72.90%	10.1	6.60%
Upper Seds: No Ox (L)	0.84	1.40%	39.56	65.30%	8.3	4.70%
Upper Seds: No Ox (H)	1.11	1.40%	58.55	73.80%	8.1	5.30%
Lower Sand: Partial Ox (L)	1.09	1.50%	42.34	57.60%	8	6.60%
Lower Sand: Partial Ox (H)	1.42	1.40%	63.66	65.00%	11.8	6.00%
Lower Sand: Trace Ox (L)	0.99	1.40%	44.22	63.70%	9.2	5.30%
Lower Sand: Trace Ox (H)	1.34	1.50%	58.08	64.60%	9.4	7.60%
Lower Sand: No Ox (L)	0.72	1.40%	28.13	56.50%	6	3.10%

Description	Calculated Head, Au g/t	Conc + Mid Wt. %	Conc + Mid Assay, Au g/t	Conc + Mid Rec % Au	Conc + Mid Assay, Au g/t	Conc + Mid Rec % Ag
Lower Sand: No Ox (H)	1.48	1.40%	74.67	71.80%	11.5	5.80%
Amy Sequence: Partial Ox (L)	0.4	1.30%	15	49.30%	4.2	2.60%
Amy Sequence: No Ox (L)	0.57	1.40%	24.19	60.20%	7.1	3.60%
Averages		**1.45%**	**42.9**	**56.00%**	**8.6**	**5.50%**

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

TABLE 16.13
LIVENGOOD PROJECT – SUNSHINE ZONE 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
Kint: Partial Ox (L)	1.87	1.27%	92.21	62.70%	8.3	5.90%
Kint: Partial Ox (H)	0.94	1.37%	39.2	57.50%	6.7	5.10%
Kint: Trace Ox (L)	1.43	1.27%	79.5	70.90%	14.5	8.30%
Kint: Trace Ox (H)	0.96	1.41%	43.32	63.50%	8.7	5.70%
Upper Seds: Partial Ox (L)	1.07	1.37%	57.54	73.60%	8.1	6.10%
Upper Seds: Partial Ox (H)	0.84	1.34%	42.32	67.60%	7.8	7.00%
Upper Seds: Trace Ox (L)	1.72	1.45%	95.31	80.50%	10.9	7.80%
Upper Seds: Trace Ox (H)	0.69	1.39%	34.94	70.60%	6.1	4.40%
Averages		**1.36%**	**60.54**	**68.40%**	**8.9**	**6.30%**

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

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. Results from this test work for the Main and Sunshine Zones is shown in **Table 16.12** and **16.13** respectively. The gold in the Livengood mineralization appears to respond well to gravity separation.

Table 16.14 provides a summary of the bottle roll test results. They indicate that most of the Livengood mineralized materials 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 mineralization type. The degree of oxidation also appears to have an effect on the gold cyanide leachability.

TABLE 16.14
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	2.8
Cambrian: Partial Ox (L)	0.8	0.75	80%	0.26	3
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
Cambrian: Trace Ox (H)	1.48	1.82	90%	0.33	2.5
Kint: Partial Ox (L)	0.69	0.73	54%	0.51	4.5
Kint: Partial Ox (H)	1.67	1.55	60%	0.99	3.33
Kint: Trace Ox (L)	0.82	0.76	24%	0.38	2.5
Kint: Trace Ox (H)	1.24	1.23	22%	0.41	2.5
Kint: No Ox (L)	0.84	0.76	32%	0.29	2.8
Kint: No Ox (H)	2.42	1.51	32%	0.81	3
Lower Seds: Trace Ox (L)	1.05	0.88	0%	0.28	2.33
Lower Seds: Trace Ox (H)	1.2	1.36	1%	0.38	2
Lower Seds: No Ox (L)	0.62	0.65	0%	1.78	2
Lower Seds: No Ox (H)	1.36	1.28	0%	0.35	2
Main Volcanics: Partial Ox (L)	0.75	0.68	56%	0.3	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
Main Volcanics: Trace Ox (H)	1.45	1.42	42%	2.21	2
Main Volcanics: No Ox (L)	0.97	0.91	49%	0.2	2
Main Volcanics: No Ox (H)	1.66	2.1	39%	2.13	2
Upper Seds: Partial Ox (L)	0.71	1.05	64%	0.35	2
Upper Seds: Partial Ox (H)	1.45	1.5	80%	0.37	2
Upper Seds: Trace Ox (L)	0.89	0.97	37%	0.22	2
Upper Seds: Trace Ox (H)	1.67	1.58	73%	0.42	2
Upper Seds: No Ox (L)	0.76	0.85	26%	0.31	2
Upper Seds: No Ox (H)	1.28	1.68	55%	0.32	2
Lower Sand: Partial Ox (L)	0.91	0.86	49%	1.98	2
Lower Sand: Partial Ox (H)	1.1	1.52	61%	0.63	2
Lower Sand: Trace Ox (L)	1.09	0.94	48%	0.39	2.5
Lower Sand: Trace Ox (H)	1.35	1.32	67%	0.4	2.33
Lower Sand: No Ox (L)	0.68	0.75	21%	0.45	2
Lower Sand: No Ox (H)	1.32	1.28	55%	0.55	2.33
Amy Sequence: Partial Ox (L)	0.39	0.7	49%	0.24	2.6
Amy Sequence: No Ox (L)	0.52	0.51	4%	0.22	2.5
Average	**1.09**	**1.13**	**47%**	**0.59**	**2.38**

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 mineralized material types was due to "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.

Recoveries improved significantly, with some mineralization types showing as high as a 49.5% increase in overall gold recovery, with the addition of carbon in the cyanide leach process. It appears that some of the mineralization types have "preg- robbing" characteristics, which explain 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 mineralization.

Similar tests were run on Sunshine Zone mineralized materials. **Tables 16.15** and **16.16** illustrate the results of these tests from the Main and Sunshine Zone respectively.

16.5.5 Flotation Concentration Tests

In order to understand how Livengood mineralization responds to sulfide flotation, a test program was developed for KCA to perform on their existing Livengood mineralized material composites. Knowing that the Livengood mineralization has a substantial amount of coarse gold, a test protocol was developed that would first subject the material to sulfide flotation followed by performing a GRG test on the flotation tailings. From this test scenario, a better understanding is gained of the ability to float the coarse gold while understanding the ability to collect gold in a pre- or post-flotation gravity circuit.

Batch flotation tests were performed by Kappes, Cassidy and Associates on samples drawn for the composites prepared for metallurgical testing from the reverse circulation drilling samples as described earlier in this report (**Section 16.5**). Duplicate tests were conducted for each of the samples, with the sample material being ground to nominally 80% passing 0.075 mm. The samples were then conditioned for 5 minutes with 5 g/t of $CuSO4$ and 25 g/t of PAX. A float concentrate was then produced in 20 minutes with rougher flotation parameters of 25% solids and AF 70-20 g/t. The flotation tails were then run through a Knelson Concentrator to collect the remaining gravity recoverable gold. The middlings portion was recovered by hand panning the gravity concentrate. All concentrate fractions and the gravity tails were assayed for gold and silver.

Results of the duplicate tests have been averaged and the proportion of total gold recovered by flotation and gravity are listed in **Table 16.17**.

16.5.6 CIL Recovery on Gravity Concentrates

Carbon in Leach (CIL) bottle roll tests (BRT) were performed on samples used to produce a gravity recoverable gold concentrate. Twenty kilogram (20 Kg) samples were split from the composites

TABLE 16.15
LIVENGOOD PROJECT – MAIN ZONE
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%**

Table 16.16
LIVENGOOD PROJECT – SUNSHINE ZONE SUMMARY OF CIL CYANIDE BOTTLE ROLL TESTS

Description	Calculated Head, Au g/t	Overall Average Assay, Au g/t	CIL Au Rec, %
Kint_Ox_(H)	1.97	2.25	86.50%
Kint_Ox_(L)	0.69	0.59	73.00%
Kint_TraceOx_(H)	0.87	1.34	79.10%
Kint_TraceOx_(L)	0.73	1.24	72.40%
Upper Seds_Ox_(H)	0.98	0.77	90.90%
Upper Seds_Ox_(L)	1.02	2.38	94.50%
Upper Seds_Trace_(H)	1.72	1.32	87.60%
Upper Seds_Trace_(L)	0.54	0.63	88.60%
Averages		**1.31**	**84.10%**

TABLE 16.17
SUMMARY OF BATCH FLOTATION TEST RESULTS MARCH 2010

Test Sample	% Gold Recovered by Flotation	% Gold Recovered by Gravity from Flotation Tails	Total Gold Recovered (%)
Volcanics, Partial Ox - Low Grade	76%	8%	84%
Volcanics, Partial Ox - High Grade	72%	15%	87%
Volcanics, Trace Ox - Low Grade	47%	49%	96%
Volcanics, Trace Ox - High Grade	66%	25%	91%
Volcanics, No Ox - Low Grade	79%	8%	87%
Volcanics, No Ox - High Grade	74%	20%	94%
Average	**69%**	**21%**	**90%**
Upper Seds, Core Z, Part Ox - Low Grade	78%	4%	82%
Upper Seds, Core Z, Part Ox - High Grade	63%	19%	81%
Upper Seds, Core Z, Trace Ox - Low Grade	39%	39%	78%
Upper Seds, Core Z, Trace Ox - High Grade	27%	69%	96%
Upper Seds, Core Z, No Ox - Low Grade	49%	32%	81%
Upper Seds, Core Z, No Ox - High Grade	53%	42%	95%
Upper Seds, Sunshine, Part Ox - Low Grade	61%	26%	87%
Upper Seds, Sunshine, Trace Ox - High Grade	73%	15%	88%
Upper Seds, Sunshine, Partial Ox - High Grade	69%	21%	90%
Upper Seds, Sunshine, Trace Ox - Low Grade	84%	7%	91%
Average	**60%**	**27%**	**87%**
Lower Seds, Trace Ox - Low Grade	34%	57%	91%
Lower Seds, Trace Ox - High Grade	24%	59%	83%
Lower Seds, No Ox - Low Grade	23%	54%	77%

Test Sample	% Gold Recovered by Flotation	% Gold Recovered by Gravity from Flotation Tails	Total Gold Recovered (%)
Lower Seds, No Ox - High Grade	19%	66%	85%
Average	**25%**	**59%**	**84%**
Cambrian, Partial Ox - Low Grade	54%	35%	90%
Cambrian, Partial Ox - High Grade	41%	51%	92%
Cambrian, Trace Ox - Low Grade	74%	22%	95%
Cambrian, Trace Ox - High Grade	52%	44%	96%
Average	**55%**	**38%**	**93%**
Lower Sand, Partial Ox - Low Grade	78%	14%	93%
Lower Sand, Partial Ox - High Grade	80%	14%	94%
Lower Sand, Trace Ox - Low Grade	83%	11%	93%
Lower Sand, Trace Ox - High Grade	69%	26%	95%
Lower Sand, No Ox - Low Grade	73%	14%	86%
Lower Sand, No Ox - High Grade	61%	35%	95%
Average	**74%**	**19%**	**93%**
Kint, Partial Ox - Low Grade	74%	4%	79%
Kint, Partial Ox - High Grade	59%	21%	80%
Kint, Trace Ox - Low Grade	64%	27%	91%
Kint, Trace Ox - High Grade	72%	17%	89%
Kint, No Ox - Low Grade	78%	9%	87%
Kint, No Ox - High Grade	76%	20%	96%
Kint, Trace Ox - Low Grade	66%	22%	88%
Kint, No Ox - Low Grade	91%	3%	93%
Average	**73%**	**15%**	**88%**
Amy Sequence, Partial Ox - Low Grade	52%	30%	83%
Average	**52%**	**30%**	**83%**

discussed earlier in this section of the report, and then ground to 90% passing 0.85 mm. The material was slurried in water and then fed into a Knelson Concentrator in 3 stages:

- Stage 1: A gravity concentrate and tails was produced for the 90% passing 0.85 mm;
- Stage 2: The tails from Stage 1 were milled to 50% passing 0.075mm and fed into the Knelson Concentrator, producing a Stage 2 concentrate and Stage 2 tails; and
- Stage 3: the tails from Stage 2 were milled to 80% passing 0.075mm and fed into the Knelson Concentrator, producing a Stage 3 concentrate and Stage 3 tails.

At each of the three stages, middlings were separated by hand panning the concentrate. The middlings products and concentrate products were combined for each of the 3 stages, and CIL bottle roll tests were performed for the Stage 3 tails, the combined middlings, and combined concentrates.

The results of the CIL bottle roll tests on gravity recoverable gold concentrates and tails are summarized in **Table 16.18**.

TABLE 16.18
RESULTS OF CIL BOTTLE ROLL TESTS IN GRAVITY CONCENTRATION TESTS MARCH 2010

Description	Product	% Gold Recovery in Product	% Total Gold Recovered	NaCN Consumption, (kg/MT)	$Ca(OH)_2$ Addition, (kg/MT)
Cambrian: Partial Ox (L)	Con	97%	58%	9.04	2.54
	Mid	91%	4%	6.08	0.72
	Tail	77%	26%	2.21	0.50
	Overall	90%	89%	2.27	0.51
Cambrian: Trace Ox (H)	Con	97%	67%	9.78	2.10
	Mid	93%	6%	7.18	0.72
	Tail	82%	19%	1.96	0.50
	Overall	93%	92%	2.05	0.51
Kint: Partial Ox (H)	Con	89%	31%	9.33	0.88
	Mid	76%	1%	7.86	0.79
	Tail	48%	29%	2.21	0.50
	Overall	63%	62%	2.31	0.51
Kint: No Ox (H)	Con	92%	45%	7.81	0.86
	Mid	81%	1%	5.79	0.79
	Tail	26%	12%	2.23	0.50
	Overall	59%	59%	2.30	0.51
Main Volcanics: Partial Ox (H)	Con	98%	71%	13.12	0.95
	Mid	95%	2%	7.05	0.74
	Tail	68%	16%	2.38	0.50
	Overall	90%	89%	2.49	0.51
Upper Seds: Trace Ox (L)	Con	78%	50%	12.34	1.97
	Mid	71%	2%	6.30	0.75
	Tail	36%	11%	1.82	0.50
	Overall	64%	63%	1.90	0.51
Upper Seds: Trace Ox (H)	Con	96%	74%	9.63	0.95
	Mid	88%	3%	7.03	0.81
	Tail	48%	9%	1.80	0.50
	Overall	87%	86%	1.89	0.51
Lower Sand: No Ox (H)	Con	95%	71%	5.85	1.43
	Mid	87%	2%	6.91	0.80
	Tail	44%	10%	2.10	0.50
	Overall	83%	83%	2.17	0.51

Description	Product	% Gold Recovery in Product	% Total Gold Recovered	NaCN Consumption, (kg/MT)	$Ca(OH)_2$ Addition, (kg/MT)
Kint_Ox_high	Con	97%	66%	9.32	1.43
	Mid	93%	2%	7.47	0.81
	Tail	59%	17%	2.36	0.50
	Overall	86%	85%	2.45	0.51
Kint_TraceOx_High	Con	97%	68%	15.13	1.47
	Mid	64%	2%	6.15	0.77
	Tail	54%	14%	1.98	0.50
	Overall	84%	84%	2.09	0.51
US_Ox_Low-Sunshine	Con	96%	61%	17.21	2.01
	Mid	92%	4%	8.33	0.77
	Tail	69%	21%	2.04	0.50
	Overall	87%	86%	2.16	0.51
US_Trace_Low – Sunshine	Con	96%	70%	12.31	1.72
	Mid	93%	4%	7.90	0.78
	Tail	69%	16%	1.86	0.50
	Overall	90%	89%	1.96	0.51

16.5.7 Ongoing Flotation and Gravity Concentration Tests

Test work on flotation and gravity concentration on the Livengood mineralization types is on-going. The test work focuses on potential concentration methods prior to cyanide leaching. The methods include both flotation and gravity and a combination of both. A program was developed and awarded to Resource Development Inc. (RDI) in Wheat Ridge, Colorado. The test work was performed on a unit flotation cell (1 cubic foot) utilizing 10 kg of mineralized material as feed. These larger scale tests provided more material for both concentrate leaching tests and for gravity separation tests. Initial test work indicates that on average more than 90 percent of the gold will report to the concentrates in a combined gravity/flotation concentrating scenario. **Table 16.19** provides the latest results from the flotation and gravity test work.

Test work is also underway to investigate cyanide leaching of the concentrates such that a doré product can be produced directly from the concentrates. Initial leach tests vary, but first run indications imply that on average greater than 81% of the gold can be leached from the combined concentrates. However, leach recoveries on the Main Volcanic and Lower Sed units, which are important components of the potential mill feed require further work to improve the leach recovery of gold from the concentrate. Leaching parameters have yet to be optimized, and it is very likely that higher leach recoveries on the concentrates will be attainable.

Table 16.19
RDI FLOTATION AND GRAVITY TEST RESULTS JUNE 2010

Sample Description			Assay Head, Au gm/t	Calc Head, Au gm/t	Flotation Recovery	Gravity Recovery	Au Recovery Reporting to Flotation Conc. and Gravity Conc.	Flotation Concentrate Au Leach Recovery (%)
Livengood Main Volcanics	Partial Oxide - High Grade	Comp 1	0.7	0.6	74.00%	9.40%	76.40%	74.3
		Comp 1	0.8	0.58	74.00%	17.90%	78.70%	90.1
Livengood Main Volcanics	Partial Oxide - High Grade	Comp 1	0.7	0.6	74.00%	9.40%	76.40%	71.1
		Comp 1	0.8	0.58	74.00%	17.90%	78.70%	67.3
Livengood Main Volcanics	No Oxide - High Grade	Comp 3	2.26	2.23	95.40%	7.50%	95.70%	56.1
		Comp 3	1.1	3.16	97.10%	67.60%	99.10%	63.5
Livengood Upper Seds	Partial Oxide - High Grade	Comp 4	1.56	0.89	68.70%	9.20%	71.60%	59.3
		Comp 4	2.21	0.72	65.30%	44.50%	80.70%	84.3
Livengood Upper Seds	Trace Oxide - High Grade	Comp 5	1.57	1.06	92.60%	42.80%	95.80%	85.7
		Comp 5	0.75	0.51	81.80%	66.30%	93.90%	83.1
Livengood Upper Seds	No Oxide - High Grade	Comp 14	0.59	0.66	91.10%	89.70%	99.10%	82.9
		Comp 14	0.47	0.5	91.90%	68.00%	97.40%	80.9
Livengood Lower Seds	Trace Oxide - High Grade	Comp 6	0.7	0.86	74.40%	40.30%	84.70%	90.6
		Comp 6	0.76	0.71	57.70%	52.60%	79.90%	69.2
Livengood Lower Seds	No Oxide - High Grade	Comp 7	2.04	0.35	70.90%	59.10%	88.10%	62.2
		Comp 7	0.49	2.22	95.70%	17.90%	96.50%	79.1
Livengood Cambrian	Partial Oxide - High Grade	Comp 8	0.96	0.63	80.10%	71.60%	94.30%	97.0
		Comp 8	0.65	0.81	87.70%	10.10%	88.90%	97.1
Livengood Cambrian	Trace Oxide -	Comp 9	1.09	1.52	78.30%	92.60%	98.40%	89.0

Sample Description			Assay Head, Au gm/t	Calc Head, Au gm/t	Flotation Recovery	Gravity Recovery	Au Recovery Reporting to Flotation Conc. and Gravity Conc.	Flotation Concentrate Au Leach Recovery (%)
	High Grade	Comp 9	5.91	2.78	97.90%	93.30%	99.90%	98.3
Sunshine Upper Seds	Partial Oxide - High Grade	Comp 10	0.49	0.59	76.80%	11.10%	79.40%	93.5
		Comp 10	0.58	1.05	88.00%	35.00%	92.20%	91.6
Sunshine Upper Seds	Trace Oxide - High Grade	Comp 11	0.64	1.01	91.40%	20.50%	93.20%	89.0
		Comp 11	0.47	1.59	96.60%	18.90%	97.20%	84.0
Livengood Lower Sand	Partial Oxide - High Grade	Comp 12	0.94	1.43	93.40%	18.10%	94.60%	81.1
		Comp 12	1.07	1.02	94.30%	12.40%	95.00%	75.8
Livengood Lower Sand	Trace Oxide - High Grade	Comp 13	0.66	0.72	87.40%	46.80%	93.30%	81.0
		Comp 13	0.97	0.37	79.40%	18.70%	83.30%	81.5
Average			**1.16**	**1.12**	**84.60%**	**39.20%**	**90.60%**	**80.3**

16.6 Future Metallurgical Test Work

ITH has undertaken or is planning further metallurgical test work. The following list is a brief outline of the test work currently envisioned for the Livengood project.

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
- Additional Flotation Concentrate and Tails Leach Tests

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

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

16.7 Mineral Processing

Based on the test work discussed previously and on the current estimated resource, process options were investigated. The process envisioned for this document involves crushing the run of mine production in a three stage crushing circuit to less than ¾ inch, and placing the crushed mineralized material on a lined heap leach pad and utilizing conventional heap leaching technologies for the first three years of operation. **Figure 16.2** presents a simple block flow diagram of the proposed circuit. The placement of material 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.

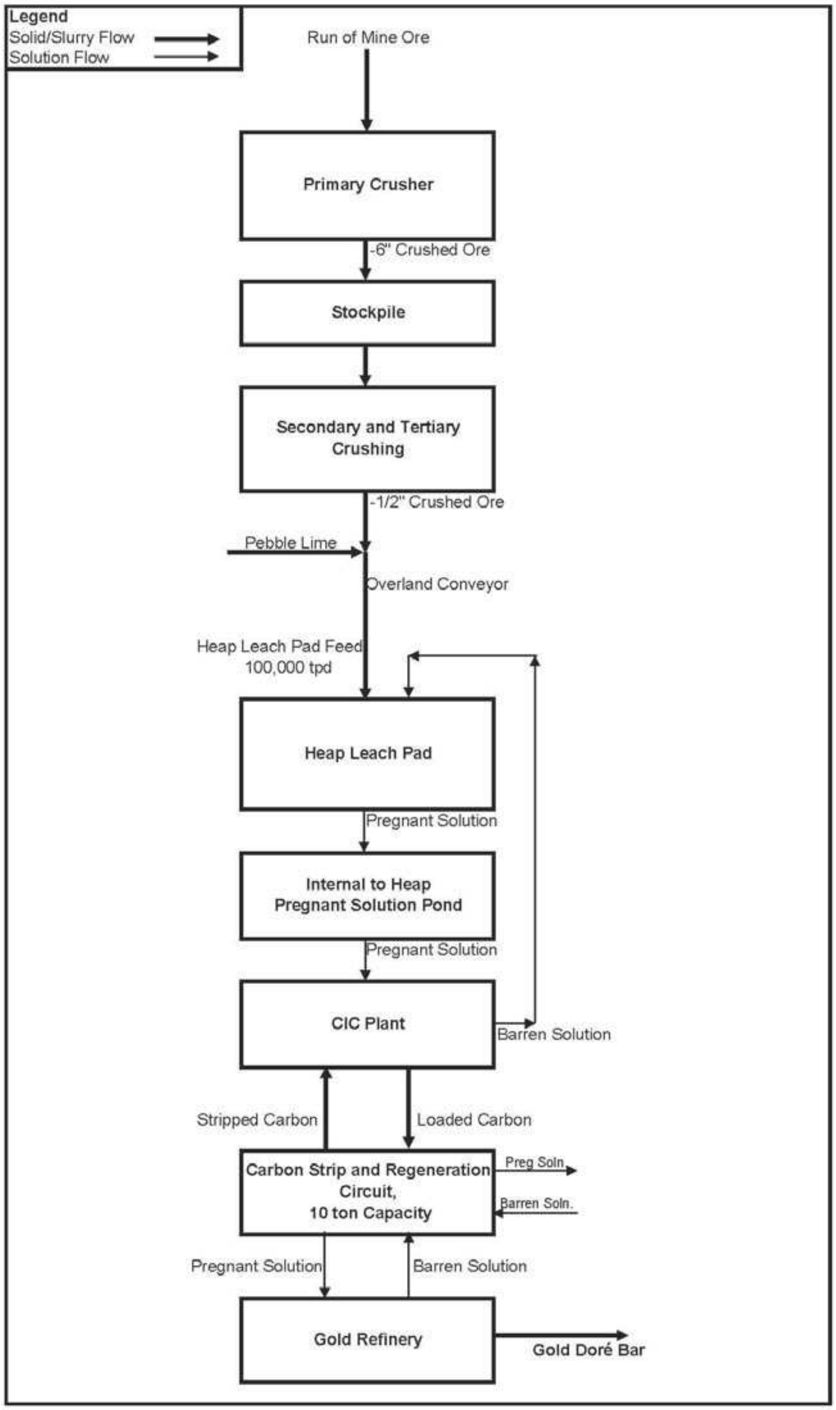


Figure 16.2. Proposed Livengood process block flow diagram showing heap leach process streams.

After the first three years of operation, utilizing only the heap leach process, a milling facility utilizing gravity separation in the grinding circuit, followed by flotation and fine grinding of the flotation concentrate, which would then be leached in a CIL circuit, is envisioned for start-up in year four. The base case scenario is for a heap leach of 100,000 tpd for the first three years reducing to 50, 000 tpd when the mill comes on line and the mill would process 53,400 tpd to 100,000 tpd.

Future mineral processing investigations will likely focus on this 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.

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 mineralization type has been performed. **Table 16.20** provides the gold recoveries as currently estimated. As shown in the table, the Lower Sediments are carrying a zero percent gold recovery for heap leaching and, for the purposes of the heap leach study, both the Lower Sediments and Main Volcanics were excluded from the heap leach. These mineralization types would be stockpiled for milling and concentrating where their preg-robbing characteristics can be better managed.

The mill recoveries utilized in this study are from preliminary test data on the various mineralization types. The tests completed include column tests, bottle roll tests using both straight cyanide leaching and cyanide leaching with activated carbon added, and flotation and gravity concentration schemes. Minimal testing on flotation concentrate leaching has been performed to date and is a focus for on-going test work. Initial test work has been performed on samples from the main volcanics, upper sediments, lower sediments, Cambrian and lower sand units. Recoveries from these preliminary tests gave gold recoveries between 56.1% and 98.3%. The average flotation concentrate leach recovery for the composite samples tested was 80.3% (Table 16.18).

The flotation leach recoveries do not include the gravity concentrates, which are primarily free gold. Leach Au recovery from the gravity concentrate is projected to be greater than 95%.

Since the concentrate leach test work has yet to be optimized and since the gravity concentrates have not been leached, an 85% combined Au leach recovery from concentrates has been assumed to estimate the overall gold recoveries from the mill process.

Current test work that may have a significant impact on the reported gold recoveries are those involving gravity concentration. The estimated Mill/Gravity/Flotation with CIL recoveries may be improved with extraction of coarse gold in a gravity circuit followed by Flotation with CIL processing. The gravity circuit could address the two key recovery factors affecting the un-oxidized mineralized materials, namely coarse gold and preg-robbing carbon. In addition, with the positive results that are being received from the flotation/gravity tests, a mill scenario that includes flotation and leaching of the gravity and flotation concentrates may be a more economic milling scenario than the conventional whole feed CIL milling scenario.

16.9 Process Operating Costs

ITH developed and presented operating costs for a 100,000 tpd heap leach operation in October, 2009 (Klipfel, et al., 2009). Since then, further work has been done on cost estimation and assessing multiple recovery methods. In particular, gravity and flotation concentration with leaching of the flotation concentrates is currently being evaluated in the laboratory. Initial test work results indicate that the recoveries shown in **Table 16.20** can be obtained.

Process operating costs have been developed for each of the identified Livengood mineralized stratigraphic units (**Table 16.21**). The test work performed on each of the mineralization types has provided preliminary data for calculating reagent and grinding media consumptions, and power consumptions by mineralization type. The size of the facility envisioned and the unit processes involved also enabled preliminary maintenance and manpower requirements to be generated for a 100,000 tpd heap leach process facility at Livengood as a stand-alone operation, and as a component in a processing system that included both a heap leach facility and a milling facility based on gravity and flotation pre-concentration with CIL leaching of the concentrate. 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.

These operating costs were input into the mine block model by mineralized material type and the block model returned an overall weighted average operating cost of $3.11 per tonne for stand-alone heap leaching at 100.000 tpd, $7.69 per tonne for milling costs at 50,000 tpd, and $2.85 per tonne for add on heap leach costs at 50,000 tpd. These costs were then used as inputs into the financial modeling exercise.

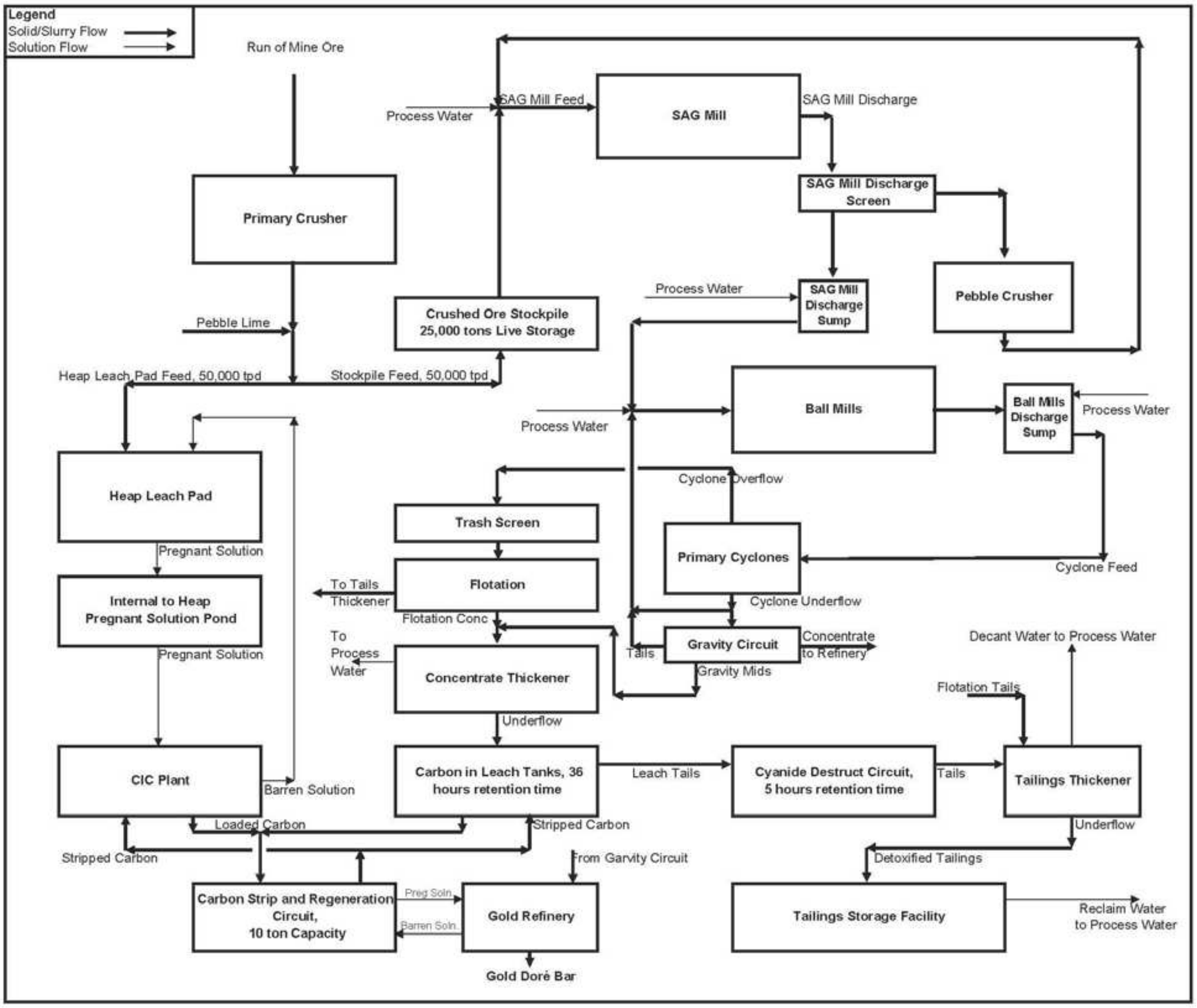


Figure 16.3 Alternate Livengood process block flow diagram showing both Heap Leach and Mill utilizing Gravity/Flotation/Concentrate CIL process streams.

TABLE 16.20
GOLD RECOVERY ESTIMATES BY MINERALIZATION TYPE FOR HEAP LEACH AND MILL (gravity/flotation w/CIL) PROCESS SCENARIOS

Mineralization Type – Main Zone	Heap Leach % Au Rec	Mill - gravity/flotation with CIL % Au Rec
Overburden	75.0%	NS
Cambrian Oxidized	86.0%	85.3%
Cambrian Trace	65.0%	88.8%
Cambrian Unoxidized	61.0%	NS
Upper Seds Oxidized	76.0%	74.5%
Upper Seds Trace	55.0%	82.8%
Upper Seds Unoxidized	50.0%	82.0%
Kint Oxidized	50.0%	72.4%
Kint Trace	22.0%	82.4%
Kint Unoxidized	22.0%	83.5%
Main Volcanics Oxidized	53.0%	78.1%
Main Volcanics Trace	35.0%	87.2%
Main Volcanics Unoxidized	33.0%	82.5%
Lower Seds Oxidized	0.0%	NS%
Lower Seds Trace	0.0%	83.0%
Lower Seds Unoxidized	0.0%	77.8%
Lower Sand Oxidized	50.0%	85.2%
Lower Sand Trace	50.0%	86.1%
Lower Sand Unoxidized	15.0%	83.6%
Amy Sequence Oxidized	35.0%	76.7%
Amy Sequence Trace	35.0%	NS
Amy Sequence Unoxidized	27.0%	NS
Ore Type – Sunshine Zone		
Sunshine Upper Seds Oxidized	78%	81.7%
Sunshine Upper Seds Trace	75%	81.6%
Sunshine Kint Oxidized	68%	72.4%
Sunshine Kint Trace	64%	81.3%

NS indicates No Sample was available for testing.

TABLE 16.21
PRIMARY MINERALIZATION TYPES
ESTIMATED PROCESS OPERATING COSTS
FOR HEAP LEACH AT 100,000 TPD, MILLING AT 50,000 TPD, AND HEAP LEACHING AS AN ADD ON TO THE MILL AT 50,000 TPD

Mineralization **Type**	**100 ktpd Stand Alone Heap Leach Op. Cost $/t**	**50 ktpd Mill / Gravity / Flotation / Conc. CIL Op. Cost $/t**	**50 ktpd Add On Heap Leach Op. Cost $/t**
Core Zone Deposit			
Overburden	$2.79	$5.92	$2.44
Cambrian Oxidized	$3.05	$7.55	$2.71
Cambrian Trace	$3.38	$7.68	$3.03
Cambrian Unoxidized	$3.38	$7.69	$3.03
Upper Seds Oxidized	$3.23	$7.73	$2.89
Upper Seds Trace	$3.36	$7.89	$3.01
Upper Seds Unoxidized	$3.36	$7.89	$3.01
Kint Oxidized	$3.97	$7.47	$3.63
Kint Trace	$3.58	$7.26	$3.26
Kint Unoxidized	$3.58	$7.26	$3.26
Main Volcanics Oxidized	$3.62	$8.24	$3.28
Main Volcanics Trace	$3.17	$7.69	$2.82
Main Volcanics Unoxidized	$2.70	$7.19	$2.35
Lower Seds Oxidized	$3.23	$7.73	$2.89
Lower Seds Trace	$3.36	$7.89	$3.01
Lower Seds Unoxidized	$3.36	$7.89	$3.01
Lower Sand Oxidized	$3.23	$7.73	$2.89
Lower Sand Trace	$3.36	$7.89	$3.01
Lower Sand Unoxidized	$3.36	$7.89	$3.01
Amy Sequence Oxidized	$3.05	$7.55	$2.71
Amy Sequence Trace	$3.38	$7.69	$3.03
Amy Sequence Unoxidized	$3.58	$7.26	$3.26
Sunshine Zone Deposit			
Sunshine Upper Seds Oxidized	$3.23	$7.73	$2.89
Sunshine Upper Seds Trace	$3.36	$7.89	$3.01
Sunshine Kint Oxidized	$3.97	$7.47	$3.63
Sunshine Kint Trace	$3.58	$7.26	$3.26

17.0 Mineral Resource Estimate

The March 2010 mineral resource estimate for the Livengood deposit was updated using information available through May 31st, 2010. The drill data was maintained in a Gemcom® GEMS database, and the basic statistical and geostatistical analysis was performed using SAGE2001® and WinGSLib®. The resource model was constructed using Gemcom GEMS® and the Stanford GSLIB (Geostatistical Software Library) MIK post processing routine. 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, based on interpretations by ITH geologists, was used to code the rock type block model. A three-dimensionally defined probability grade shell (0.1 g/t) was used to constrain the gold estimation. A summary mineral resource at cutoff grades of 0.3, 0.5, and 0.7 g/t gold is shown in **Table 17.1.** The results are presented as in-situ.

TABLE 17.1
SUMMARY IN-SITU MINERAL RESOURCE

Classification	Au Cutoff (g/t)	Tonnes (millions)	Au (g/t)	Million Ounces Au
Indicated	0.30	789	0.62	15.7
Inferred	0.30	229	0.55	4.9
Indicated	**0.50**	**409**	**0.83**	**10.9**
Inferred	**0.50**	**94**	**0.79**	**2.4**
Indicated	0.70	202	1.07	6.9
Inferred	0.70	40	1.06	1.4

Compared to the March 2010 resource estimate, the tonnage and total ounces estimated has increased in the Indicated category and has decreased in the Inferred category for cutoff grades of 0.30, 0.50, and 0.70 g/t Au. This change is due, in part, to addition of newly defined tonnes in the southwestern area of the deposit, and to the addition of a number on infill holes in the central area of Money Knob that have improved the classification of material, with a consequent conversion to Indicated classification. Other model validation activities are discussed in Section 17.7, including external review of the estimation methodology used for the Livengood resource.

> Based on the study herein reported, delineated mineralization of the Livengood Deposit is classified as a resource according to the following definitions from National Instrument 43-101 and from CIM (2005):
>
> *"In this Instrument, the terms "mineral resource", "inferred mineral resource", "indicated mineral resource" and "measured mineral resource" have the*

meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as those definitions may be amended."

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

17.1 Data Used

17.1.1 Sample Data

The data available for this model comprised 123,034 meters of core and RC drilling, plus trench data. Historical drilling and sampling is shown in **Table 17.2**. Drilling performed by TGA is shown in **Table 17.3**. It can be seen that the historical data represents about 4% of the total information used. The use of historic data is based on its statistical consistency with current data and the small portion of the total data represented as shown in past technical reports (Klipfel and Giroux, 2008a and b, 2009; Klipfel et al., 2009, and 2009a). For data validation purposes, Mr. Carew checked the assay data for a sample of drill holes (10%) used for the resource estimate in GEMS against the original assay certificates (Secure PDF). The error rate of less than 1% is well within acceptable standards. These minor errors arose exclusively from mismatches with samples re-assayed for QA QC purposes, and were corrected by revising the GEMS database update procedure.

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 and RC chip samples and are shown in **Table 17.4**.

TABLE 17.2
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 17.3
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	Core	12	4,572
2009	RC	195	59,757
2010	Core	6	1,998
2010	RC	50	15,584
Total		**404**	**118,288**

TABLE 17.4
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			

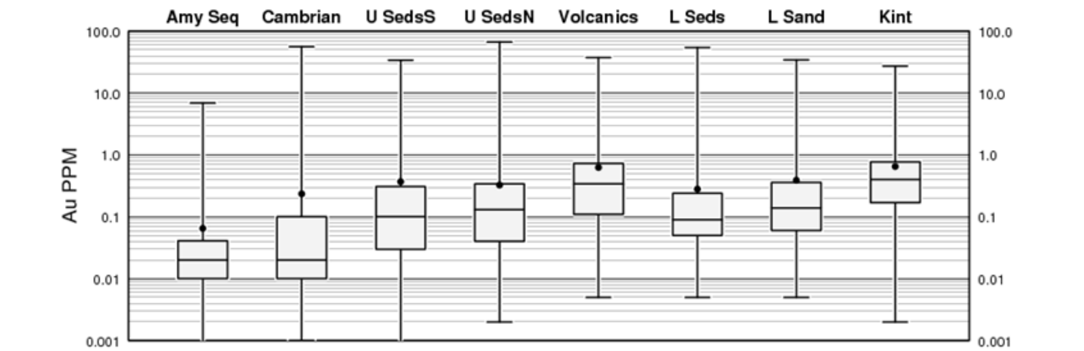
17.2 Data Analysis

Multi-element assay information is available for nearly 50% of the samples. A statistical summary of this data from a previous report (July 09) is shown in **Table 17.5**. 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 17.5
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.



	Amy Seq	Cambrian	U SedsS	U SedsN	Volcanics	L Seds	L Sand	Kint	
Number of data	4222	14039	12669	19584	10342	8093	4403	7505	Number of data
Mean	0.065	0.235	0.37	0.326	0.628	0.281	0.39	0.651	Mean
Maximum	6.92	56.2	34.2	67.3	37.4	54.5	34.4	27.3	Maximum
Upper quartile	0.041	0.1	0.31	0.34	0.73	0.24	0.36	0.77	Upper quartile
Median	0.02	0.02	0.1	0.13	0.34	0.09	0.14	0.4	Median
Lower quartile	0.01	0.01	0.03	0.04	0.11	0.05	0.06	0.17	Lower quartile
Minimum	0.0	0.001	0.0	0.002	0.005	0.005	0.005	0.002	Minimum
Variance	0.0487	1.5762	1.2582	0.6464	1.552	0.8469	1.0754	1.0235	Variance
CV	3.402	5.335	3.03	2.467	1.984	3.27	2.658	1.555	CV
Skewness	14.488	20.238	11.124	33.47	9.852	29.847	14.556	8.694	Skewness

Figure 17.1. Gold distribution by lithology unit.

17.3 Geologic Model

ITH geologists provided sectional interpretations of the major lithologic units – these were used to generate a three dimensional wire framed geological model of these 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 of the material is undifferentiated Upper Sediments, with a small amount of Volcanics and Lower Sediments modeled. These represent the major lithologic units that host the mineralization. No other geologic features with possible 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 rock type block model developed from the defined geological three-dimensional wire frames.

The composite data was declustered by estimating a nearest-neighbor value into each block. The declustered composite statistics are tabulated below, (**Table 17.6**).

TABLE 17.6
GOLD COMPOSITE STATISTICS

Mean:	0.36
Variance:	0.22
C. of V.:	1.30
Min:	0.005
Q1:	0.11
Median:	0.23
Q3:	0.45
Max:	10.99

17.4.1 Gold Indicator Statistics

The declustered 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 10 to 11% of the metal per class (**Table 17.7**). With MIK, top cutting of the assays is not necessary. In this case all composite values greater than 1.8 g/t Au (the highest threshold) are treated the same as "high grade" and the mean value of 3.14 g/t Au is used to evaluate the highest class.

TABLE 17.7
GOLD INDICATOR STATISTICS

		Data		Metal		
	Threshold	%	Cum%	%	Cum%	Mean
1	0.09	20.8	20.8	3	3	0.052
2	0.18	20.4	41.2	7.5	10.5	0.132
3	0.28	16.9	58.1	10.7	21.2	0.226
4	0.4	12.8	70.9	11.9	33.1	0.333
5	0.6	12.5	83.4	17.1	50.2	0.491
6	0.75	5.3	88.7	9.7	59.9	0.662
7	0.9	4	92.7	9.1	69	0.819
8	1.2	3.7	96.4	10.8	79.8	1.034
9	1.8	2.4	98.8	9.5	89.3	1.418
Max	**10.99**	**1.2**	**100**	**10.7**	**100**	**3.027**

17.4.2 Contact Analysis

Because significant grade contrasts were noted between the different rock types from the assay statistics, contact analysis was performed in the previous study (October 2009) 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. The additional data available for this update did not appear to alter these relationships, and the contact analysis was not repeated.

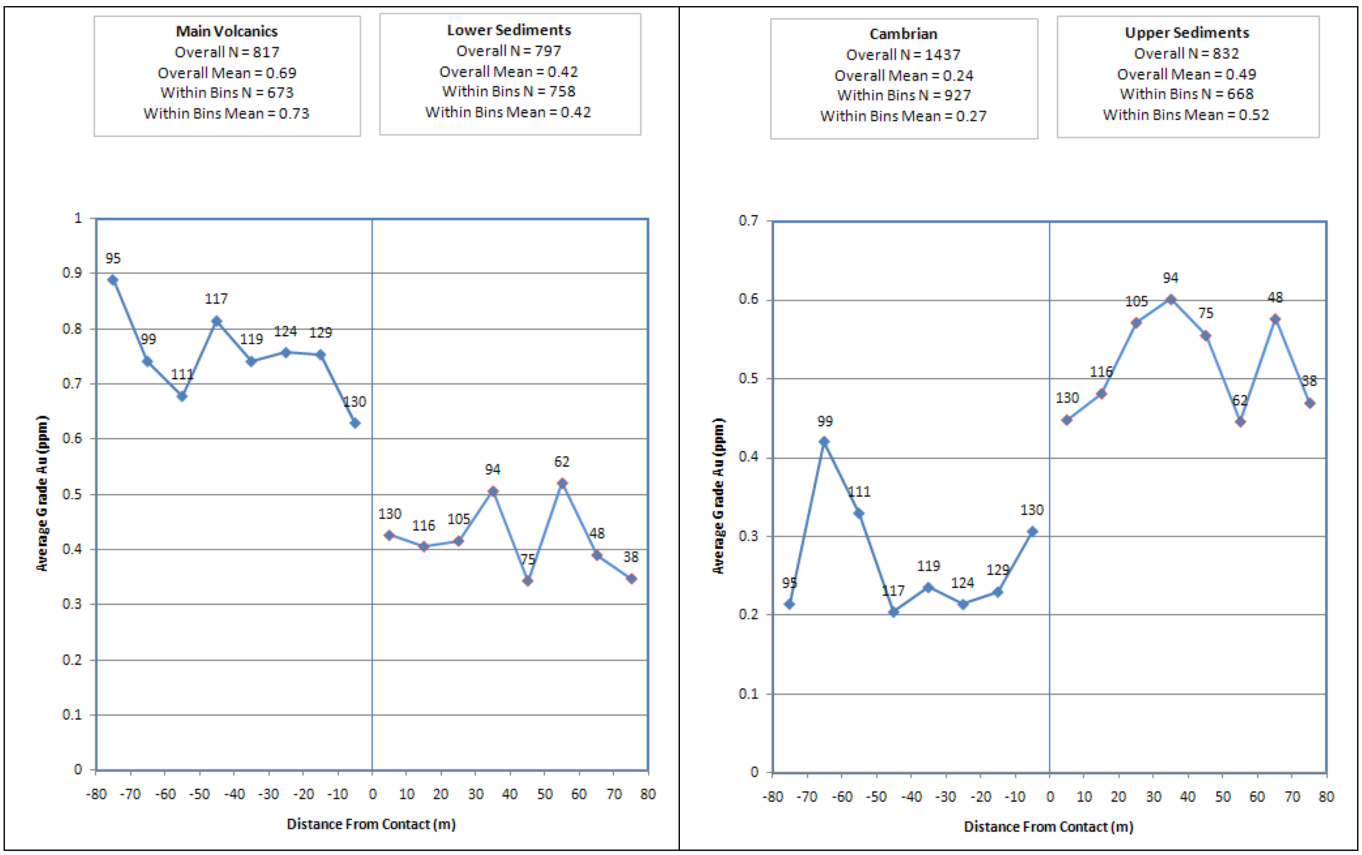


Figure 17.2. Contact plots.

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

Analysis of the additional data available for the update indicated that there were no significant changes in the spatial statistics, and the variography from the March 2010 update was therefore retained for gold, oxidation, and the minor rock types.

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 17.8**.

17.5.2 Oxide Indicator Variograms

The oxidation model was estimated using two oxide indicators, one for oxidized and one for trace (**Table 17.9**). Both the oxidized indicator variogram and the trace indicator variogram are essentially horizontal.

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 17.10**). The variogram in the north dips steeply to the south. The variogram in the south was rotated with the horizontal plane dipping to the south-west.

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 17.11**). 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

TABLE 17.8
AVERAGE GOLD INDICATOR VARIOGRAMS

Indicator	Sill	Range X	Range Y	Range Z
1	0.57			
	0.30	96	33	59
	0.13	176	360	251
2	0.57			
	0.27	179	38	82
	0.16	217	391	404
3	0.59			
	0.24	134	48	88
	0.17	217	460	364
4	0.56			
	0.28	105	33	82
	0.16	197	398	382
5	0.65			
	0.21	104	26	92
	0.14	380	216	393
6	0.66			
	0.24	159	31	56
	0.10	211	499	411
7	0.74			
	0.19	136	30	39
	0.07	198	358	600
8 & 9	0.83			
	0.12	130	79	22
	0.05	168	539	227

TABLE 17.9
OXIDE INDICATOR VARIOGRAMS

Indicator	Sill	Range X	Range Y	Range Z
Oxidized	0.19			
	0.40	134	73	115
	0.41	2317	2553	273
Trace	0.03			
	0.52	155	47	144
	0.45	2867	1117	320

TABLE 17.10
KINT DIKE VARIOGRAMS

Domain	Sill	Range X	Range Y	Range Z
North	0.30			
	0.51	64	54	616
	0.19	119	552	696
South	0.23			
	0.65	259	19	33
	0.12	368	254	431

TABLE 17.11
LOWER SANDS, SHALE & AMY SEQ. VARIOGRAMS

Domain	Sill	Range X	Range Y	Range Z
L. Sand	0.22			
	0.46	63	189	233
	0.32	633	2570	2
Amy Seq.	0.15			
	0.25	579	115	114
	0.60	774	614	211
N. Shale	0.21			
	0.67	91	48	110
	0.12	95	812	399
S. Shale	0.11			
	0.63	46	40	177
	0.26	1000	1205	167

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.

The Amy Sequence variogram dips shallowly to the East, while the Lower Sand variogram is essentially horizontal. The North Shale variogram dips shallowly to the North-West, and the South Shale variogram is horizontal.

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 17.12**.

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.

TABLE 17.12
MODEL EXTENTS

	Minimum (m)	Maximum (m)	Extent (m)	Block Size (m)	No. of Blocks
East	427,500	430,800	3,300	15	220
North	7,264,300	7,266,700	2,400	15	160
Elevation	50	560	510	10	51

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. There was no potentially economic mineralization outside of the geological model and it was not estimated. The gold kriging plan is shown in **Table 17.13** for all units.

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.

TABLE 17.13
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 (Maj.), 250 (Semi-Maj.), 200 (Min.)
Search Rotation	Maj. -30º →170º, Semi-Maj. 100º

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 by 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 17.14 and Table 17.15**.

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 un-oxidized 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 un-oxidized material is not encountered except in the lower sediments.

TABLE 17.14
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 (Maj.), 150 (Semi-Maj.), 100 (Min.)
Search Rotation	None

TABLE 17.15
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 (Maj.), 150 (Semi-Maj.), 100 (Min.)
Search Rotation	None

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 17.16** and **Table 17.17**.

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.

TABLE 17.16
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)	300 (Maj.), 250 (Semi-Maj.), 150 (Min.)
Search Rotation	Maj. -55º →248º, Semi-Maj. 80º

TABLE 17.17
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)	300 (Maj.), 250 (Semi-Maj.), 50 (Min.)
Search Rotation	Maj. -80º → 191º, Semi-Maj. 352º

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 17.19.** Note that the Amy Sequence occurs only in the Cambrian, and that the Lower sands occur only in the Lower Sediments.

TABLE 17.18
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 104º
Search Distance (m) – South Shale	Major 300, Int. 300, Minor 75
Search Rotation – South Shale	Major 0º → Azimuth 342º
Search Distance (m) – North Shale	Major 300, Int. 230, Minor 50
Search Rotation – North Shale	Major -7º → Azimuth 293º

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 0.358 g/t Au and the corresponding average block value (E-Type estimate, or block average calculated from MIK bins) is 0.365 g/t. The estimated block values are within 2.0 % of 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.

17.7.3 Swath Plots

Swaths were taken through the model and the averaged block values (e-type MIK estimates) and the averaged declustered composite values (nearest-neighbor estimates) were compared on E-W, N-S and vertical swaths (**Figure 17.3**). The kriged values have a small amount of spatial smoothing, but generally compare quite favorable to the composite values, with areas of some divergence corresponding to swaths with a low number of samples.

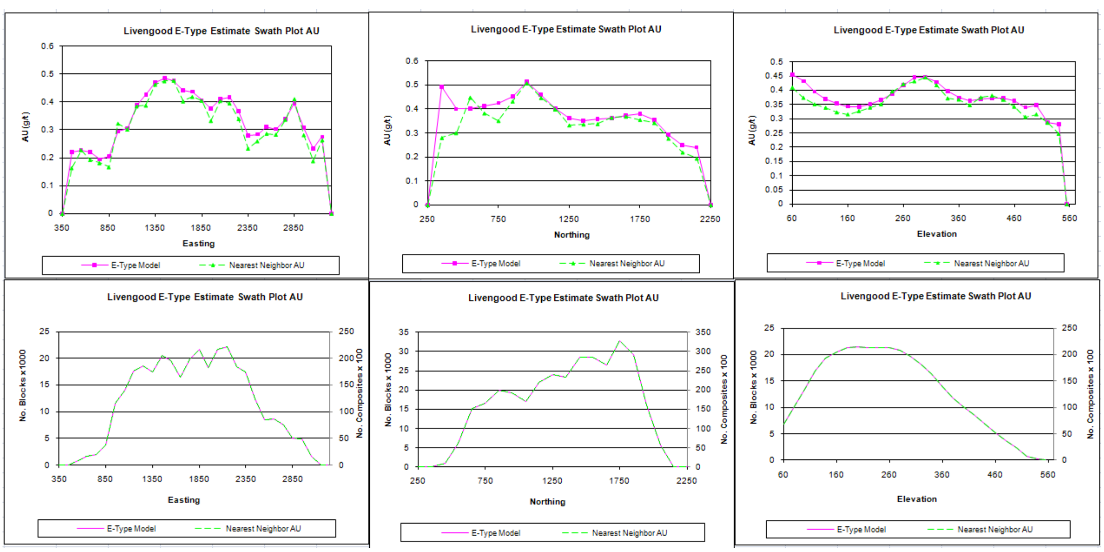


Figure 17.3. Swath plots of E-type estimate vs. nearest neighbor.

17.7.4 Review of Resource Estimation Methodology

ITH has commissioned an independent review of the resource estimation methodology as part of its Quality Assurance program (Schofield, 2010). The review concluded that Multiple Indicator Kriging (MIK) was the appropriate estimation method for the deposit. The MIK approach to recoverable resource estimation has been found to be more useful than Ordinary Kriging (OK) where the size of the ore selection unit is small compared to the spacing of the drill holes, and/or when sensitivity to extreme sample grades exists.

The review suggested that the block/panel size and SMU size should be larger due the generally 75 m drill hole spacing, and that a composite length of 3 m would be more appropriate than the 10 m composite currently selected for the Livengood model. Based on spatial analysis that places more emphasis on short range variability, and sample spacing, the review also recommended reducing the size of the search neighborhood selected for the estimation.

The impact on the resource estimation of the different assumptions was evaluated by generating an alternative estimate using the Livengood data. The comparison between alternate calculation and the Livengood resource estimate is summarized below:

- The current Livengood resource estimate is larger than would be produced using the alternate assumptions, with the main difference relating to material that is projected below the drill holes when using the larger neighborhood search parameters. The location of this material is illustrated by the cross sections showing drill hole data and model blocks in **Figures 20.1** to **20.4**., where resource blocks are extrapolated beyond the base of the drill data due to the larger search neighborhood used in the current Livengood resource estimate. This material, and similar material extrapolated laterally are predominantly classified as Inferred resource in the current Livengood resource estimate.
- The tonnage, grade and contained metal of the volumes common to both calculations are quite similar. The common volumes are constrained to close proximity of the drilling data due to the reduced search radius in the alternate method. This was evaluated by comparing calculations of recoverable resource above 0.5 g/t within the pit shell used in the heap leach analysis reported by Klipfel et.al., 2009b.
- Although the distribution of classifications was different, both the alternative calculation and the Livengood resource estimate predominantly assigned the material in the volumes common to both calculations an Indicated or above.

ITH believes that extrapolation beyond the current drilling data, due to the larger search radius, is appropriate and supported by a limited number of holes that extend beyond the current typical drill depth and provide support for geologic and grade projection. This portion of the Livengood resource estimate is predominantly classified as Inferred resource, which does not have verified geological and grade continuity.

17.8 Post-processing of MIK Model

The post-processing of the indicator kriging was done with the GSLIB post processing routine (postik). It is necessary to provide a maximum grade of the distribution. This grade can be calculated as:

$$Z_{max} = Z_{cn} + 3(Z_n - Z_{cn})$$

Where Z_{cn} is the uppermost indicator threshold, and Z_n is the mean of values > Z_{cn}. From the data in **Table 17.9**, the maximum grade used in the post-processing was calculated to be 5.49 ppm.

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 40m by 40m (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. Note that since the variography from the March 2010 update was retained for this update, that the change of support parameters are unchanged from the earlier study.

$$\bar{\gamma}(.,b_{40,40}) = 0.716$$

$$\bar{\gamma}(.,b_{5,5}) = 0.590$$

$$RF(5,5) = \sqrt{\frac{\bar{\gamma}(.,b_{40,40}) - \bar{\gamma}(.,b_{5,5})}{\bar{\gamma}(.,b_{40,40})}}$$

$$= 0.42$$

This correction is applied on a block-by-block basis with a global reduction target of 0.42. This is done on a trial and error basis to find the block reduction factor that will achieve the target global variance reduction of 0.42. A reduction factor of 0.28 was used by block.

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 plots of distance to the nearest sample versus variance (**Figure 17.4**), 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 0.33m and with a minimum of 4 octants informed, should be considered Indicated. Blocks with an estimation variance less than 0.43, and a minimum of 4 octants informed, should be considered Inferred. Blocks with an estimation variance greater than 0.43 were considered to be too unreliable for further consideration.

On average, Indicated blocks are within 34m of the nearest composite, and are informed by 27 composites from at least 8 drill holes. On average, Inferred blocks are within 84m of the nearest composite, and are informed by 20 composites from at least 6 drill holes.

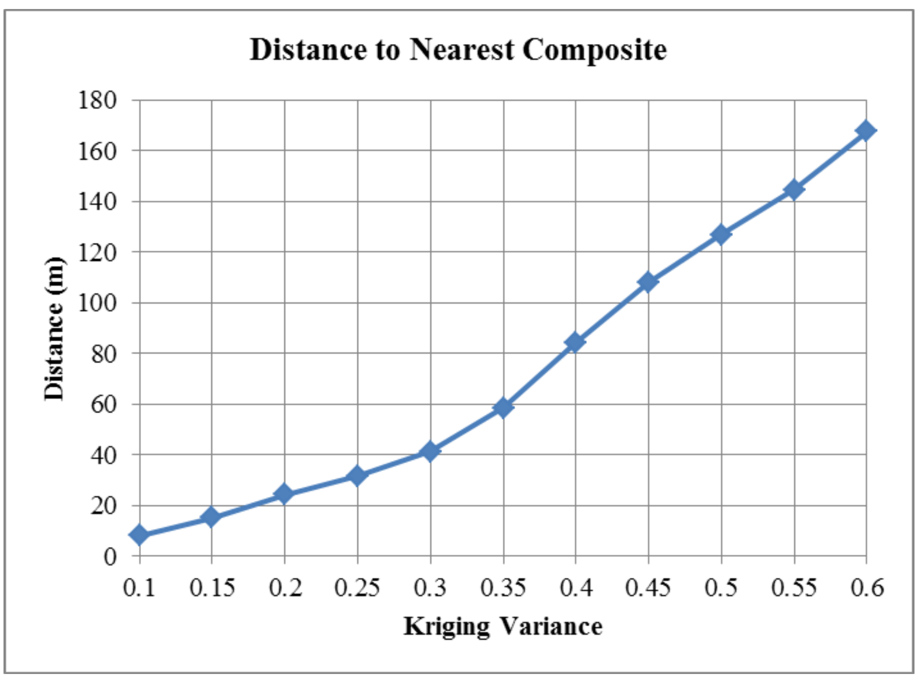


Figure 17.4. Distance to the nearest composite vs. kriging variance.

18.0 Potential Project Development Scenarios and Preliminary Assessments

This section of the technical report summarizes the results of a PA prepared by William Pennstrom (Metallurgical Engineer) of PCI, Denver, Colorado, R. John Bell (Civil Engineer) of MTB Denver, Colorado and Quinton de Klerk (mining engineer) of Cube Consulting Pty Ltd, Perth, Australia, using information from the June 2010 resource model. The **PA** describes two alternatives for processing, among several possibilities which are currently being evaluated and considered by ITH. These alternatives included processing and recovery of gold through a heap leach, and a combined heap leach and mill. Optimal use of these processing alternatives depends on numerous factors including mineralization type, relative gold recovery for the different process methods, estimated operating costs, timing and magnitude of capital expenditure project configurations, production scheduling, material processing, and financial analysis. All cost and financial data are based on constant US dollars at the time of analysis (Q3 2010). No escalations have been included.

Two development alternatives have been considered for preliminary economic assessment:

(1) Staged Construction Development beginning with heap leaching of the oxide portion of the Money Knob. The heap leach could be followed by the eventual construction and operation of a mill to process the deeper sulphide mineralized materials.

(2) Concurrent Development of heap leaching and milling operations.

The alternatives were developed using the following approach. Metallurgical testing data, presented in Section 16, suggest that mineralization at Money Knob is a candidate for two primary processing alternatives; heap leaching of near-surface oxide mineralization from the Main and Sunshine Zones, and milling using gravity and flotation pre-concentration followed by CIL leaching of the concentrates to treat sulphide mineralization from the deeper areas. Preliminary flow sheets for these two processes were developed and used to generate projected operating costs. Cost and recovery assumptions were then used in conjunction with the in-situ resource model, discussed in Section 17.0, to perform incremental revenue pit optimization of open pit mining volumes based on a range of gold prices between $300-1,500 per Au ounce. Two pit shells were selected for further analysis, a $775 shell constrained to the oxide portion of the resource, and an $850 shell which would extract both the oxide and sulphide portions of the resource. Pit designs and production schedules were then developed, and used as the basis for projections of mining operating (Opex) and capital (Capex) costs. The cost and production data were then input into financial models to estimate a range of project financial returns.

A group of project concepts were developed to examine the two project configurations and are listed in **Table 18.1**.

TABLE 18.1
SUMMARY OF PROJECT CONCEPTS EXAMINED IN THE PA

Recovery Technique	Scale Heap/Mill (ktpd)
Heap Leach Only	100/0
Heap and Mill	100/50
Heap and Mill	100/100

The heap leach only alternative considers production that is limited to the oxide portion of the deposit with a nominal heap leach placement rate of 100 ktpd. It is considered to be the optimal development path for ITH by:

- Focusing engineering and permitting on the oxide pit and heap leach operations, allowing the shortest schedule to initial gold production;
- Reducing the initial capital investment required to achieve substantial gold production; and
- Reducing the complexity of the project during the early stages of ITH's organizational growth.

The project configuration that includes both a heap leach and mill processing system assume the initial construction and operation of a heap leach pad for the first 3 years of operating life. This 3 year period is required to allow the pit to reach a depth where production of the deeper, sulphide zones can reach a volume sufficient to sustain the mill. The mill construction would be complete at the end of year +3, and operations would begin in year +4. The combination heap leach and mill (100/50) has production assumptions nominally similar to the Fort Knox mine operated by Kinross Gold Corporation near Fairbanks, AK. A heap leach with an initial production rate of 100 ktpd, would be followed by construction of a 50 ktpd mill with mill production beginning in year +4. Beyond year +4, the milling throughput is the controlling factor for the mining schedule, with a variable placement rate of the produced heap leach tonnage. A combined heap leach and 100 ktpd mill operation was also evaluated.

Subsequent portions of Section 18.0 describe the open pit mining optimization process used to define the production volumes and production schedules that form the basis of the preliminary economic assessment. The process to develop the estimates of operating expense (Opex) and capital expense (Capex) follows, with a discussion of the projected financial performance of each processing alternatives.

18.1 Processing Evaluation

Processing information was presented in Section 16.0, along with the metallurgical testing data and the assumptions used to estimate the process recoveries. PCI was responsible for development of the process assumptions used in this analysis.

18.2 Mining Evaluation

Cube Consulting Inc (Cube) of Perth Australia was retained by ITH to develop pit optimizations, pit designs, and life of mine production schedules for the Livengood project (Mader, 2010). ITH supplied Cube with the Livengood June 2010 in-situ resource block model, topography and all input parameters for pit optimization, pit design and production scheduling. Incremental revenue pit optimization runs using Whittle software were created to evaluate pit options for the two paths of heap leach and mill, and heap leach only options. Pushback designs were developed for each of the options, and production schedules targeting material to the leach pad and the mill were produced using MineSight Strategic Planner.

Livengood resource modeling uses the Multiple Indicator Kriging method to create a block model with an estimate of the mining recoverable resource within each block at cut-off grades of 0.3, 0.5 and 0.7 g/t. Individual in-situ resource blocks were assigned an economic value based on process recovery and contained gold above the 0.3 g/t cut-off grade, and the economic value is used for the revenue optimization process. Within the pit shell, the blocks were assigned to one of the heap leach, mill or waste dump destinations based on the economic value. For blocks assigned to the heap leach or mill destination, the individual block grade-tonnage data developed in the Multiple Indicator Kriging in-situ resource model was used to calculate the mining recoverable tonnage above the 0.3 g/t cut-off grade. The mining recoverable resource tonnage was scheduled to the appropriate process circuit (mill or heap leach) and the remaining material below the 0.3 g/t mining recovery cut-off was scheduled to the waste dump.

18.2.1 Heap Leach Only

This section describes the pit optimization process and production schedule for the heap leach only operation.

18.2.2 Heap Leach Only Pit Optimization

Mining and G&A operating cost assumptions from the October 2009 technical report (Klipfel, Carew and Pennstrom, 2009b) were used in the pit optimization work. Those costs were nominally 10% greater than the final cost assumptions used in the financial analysis. The difference is not considered to be significant at this stage of project definition.

A mining cost of $1.80/ton was assumed for all optimization runs. Processing costs and gold recoveries varied by rock type and oxidation state and are shown in **Table 18.2**. An additional $0.68 was added to the processing costs listed in Table 18.1 to account for administration ($0.60) and transport and refining ($0.08) costs. A royalty of 2.5% of the gold price was accounted for as a selling cost in the Whittle setup. An overall slope of 45 degrees in all rock types was used for the pit optimization runs. No geotechnical basis for pit slope specifications is currently available.

Incremental revenue optimization runs for gold prices from $300/oz to $1500/oz in $25 increments were created to evaluate the leach only scenario for the Livengood project. **Figure 18.1** displays the projected feed and waste tonnage along with the undiscounted net value of the pit shells at an $850/oz gold price. The undiscounted net value for the pit shells does not take into account capital costs but is the net value given the leach tonnes and gold grade, waste tonnes and input mining cost, processing costs, royalty cost and gold recoveries. The pit shell generated at an $850/oz gold price contained 288 Mt of heap leach material at a 0.61 g/t gold grade and a total of 332 Mt of Waste. A total of 4.05 Moz of gold are recovered through the Leach process for an undiscounted net value of approximately 1,101 M$ at an $850/oz gold price.

A plateau in the undiscounted net value begins to form between the $775-$950 gold prices. The pit shell that resulted from a $775/oz gold price was selected as the basis for the final limit pit design in the heap leach only case. The pit shell generated at an $775/oz gold price contained 259.5 Mt of Leach material at a 0.62 g/t gold grade and a total of 264 Mt of Waste. A total of 3.69 Moz of gold are recovered through the Leach process for an undiscounted net value of approximately 1,086 M$ at an $850/oz gold price.

TABLE 18.2

METALLURGICAL PROCESSING COSTS AND AU RECOVERY ASSUMPTIONS FOR THE HEAP LEACH ONLY PIT OPTIMIZATION.

Rock Type	Oxidation	Pad Process $/ton	Pad Recovery %
	Oxide	3.05	86.0%
Cambrian	Trace	3.38	65.0%
	Unoxidized	3.38	61.0%
	Oxide	3.23	78.0%
Upper Seds Sunshine	Trace	3.36	75.0%
	Unoxidized	3.36	50.0%
	Oxide	3.23	76.0%
Upper Seds Main	Trace	3.36	55.0%
	Unoxidized	3.36	50.0%
Volcanics Sunshine	Oxide	3.62	53.0%
Volcanics Main	Oxide	3.62	53.0%

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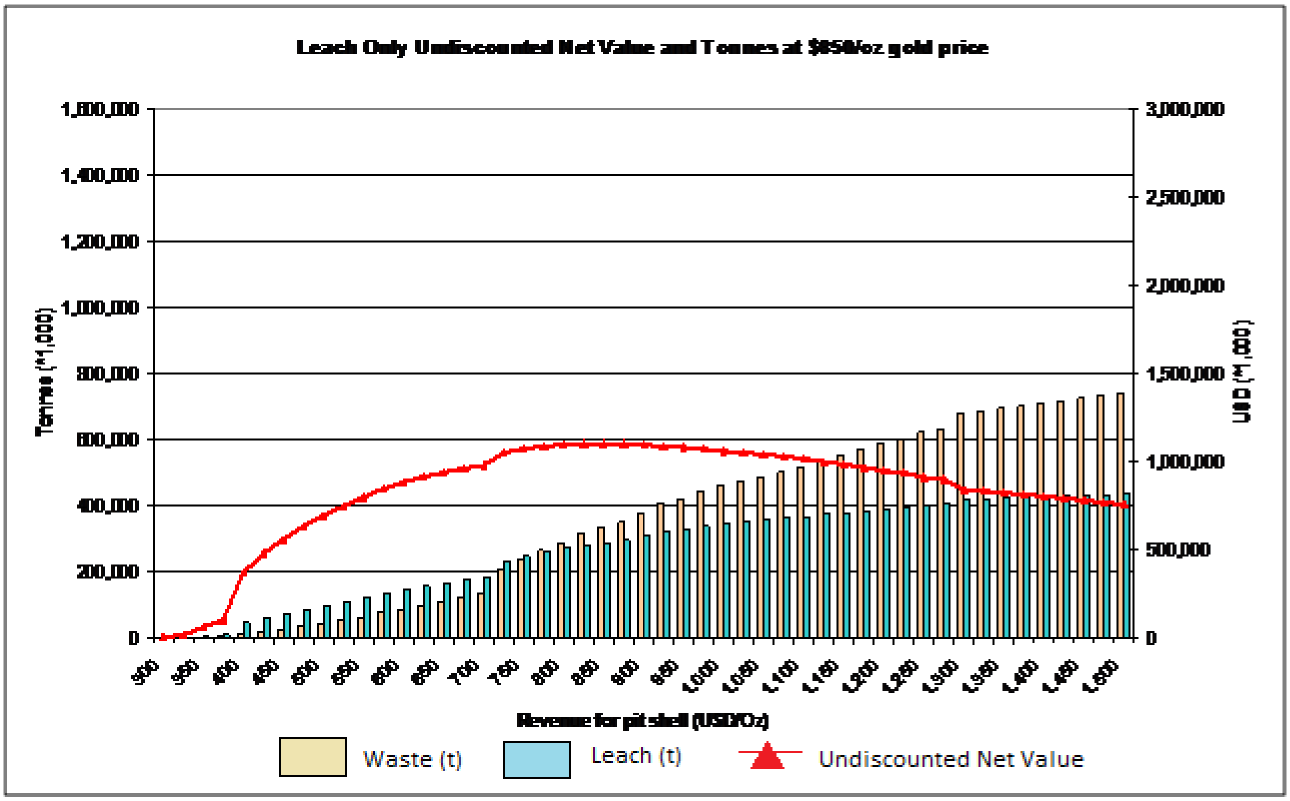

Figure 18.1. Leach tonnes, waste tonnes and undiscounted net value for heap leach only pit shells versus gold price.

The slopes used in the pit optimization do not include any allowance for ramps and therefore the pit designs based on the selected pit shell will have an increase in waste tonnes and a loss of leach tonnes. Average slope of the designed pit was approximately 40 degrees.

18.2.3 Heap Leach Only Pit Design

The final limit pit design for the Heap Leach Only is shown superposed on a geo-referenced photograph of the topography in **Figure 18.2**. The pushbacks used to develop the schedule are displayed in **Figure 18.3** and the resulting production schedule is listed in **Table 18.3.** The leach material production rate was fixed at 100 ktpd.

The Heap Leach Only pit is very well supported by the existing drilling data. Ninety seven (97) percent of the gold produced was indicated resource, with only 3% classified as inferred.

18.2.4 Mill and Heap Leach Operation

This section describes the pit optimization process and the production schedule for the heap leach and mill operations, with mill throughputs of 50 ktpd and 100kptd Mill and Heap Leach Pit Optimization.

Input parameters for the Mill and Heap Leach operation were identical to those used for the Heap Leach Only analysis listed in Section 18.2.1, with the exception that Mill processing costs were included and Heap Leach Pad processing costs were adjusted to reflect the synergies of the dual process operations. Those process cost assumptions and Au recovery assumptions by rock type and oxidation are listed **in Table 18.4**

Incremental revenue optimization runs for gold prices from \$300/oz to \$1500/oz in \$25 increments were created to evaluate the mill and heap leach scenario for the Livengood project. **Figure 18.4** displays the resulting process feed and waste tonnage along with the undiscounted net value of the pit shells at an \$850/oz gold price. The undiscounted net value for the pit shells does not take into account capital costs but is the net value given the process tonnes and gold grade, waste tonnes and input mining cost, processing costs, royalty cost and gold recoveries. A plateau of value forms around the \$850 price, which was chosen as the shell to be used in the design. The pit shell generated at an \$850/oz gold price contains 288 Mt of Leach material at a 0.57 g/t gold grade and 427 Mt of Mill material at a 0.70 g/t gold grade with a total of 730 Mt of Waste. A total of 11.8 Moz of gold are recovered through the Leach and Mill processes for an undiscounted net value of approximately 2,856 M\$ at an \$850/oz gold price.

Pit designs were created for the mill and heap leach option and used in evaluating two cases with different mill throughputs. A 100 Ktpd heap leach and 50 Ktpd mill case was created as a base line similar to the production scenario at the Fort Knox gold mine near Fairbanks AK. A mill throughput was assumed for a 100 Ktpd heap leach with 100 Ktpd mill case to examine the impact of scale and mine life.

The final limit pit design for the mill and heap leach was the same, and is shown with the local topography in the geo-referenced photograph in **Figure 18.5**. The pushbacks are displayed in **Figure 18.6,** and were used to produce two different production schedules which are listed in **Tables 18.5** and **18.6** for the two mill throughputs, respectively.

Figure 18.2. Heap leach only pit design shown in a geo-referenced photograph of the Money Knob area.

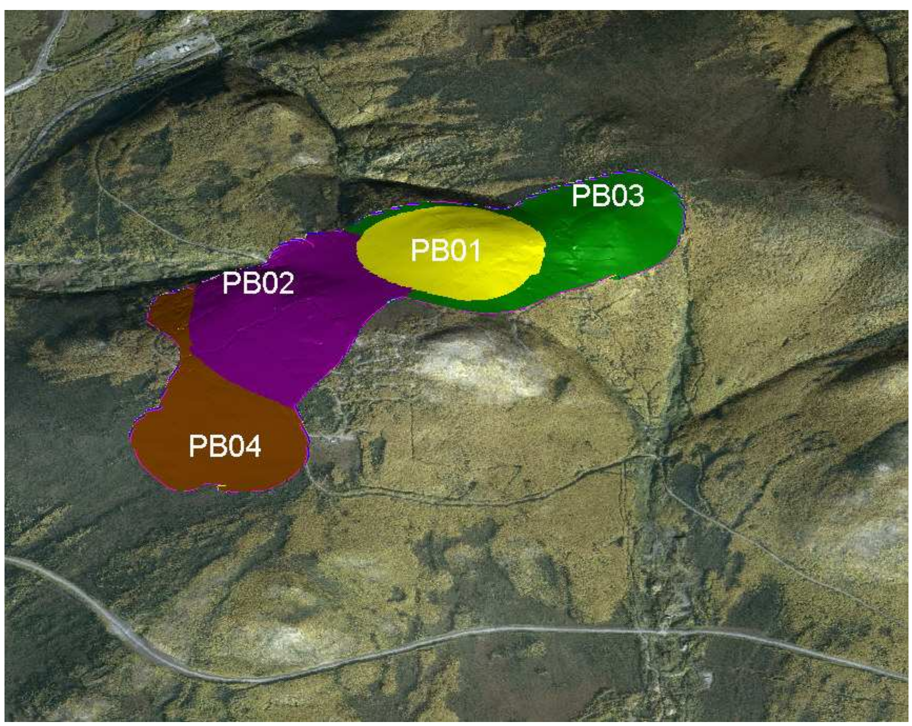

Figure 18.3. Pushbacks used for heap leach only production scheduling.

TABLE 18.3
HEAP LEACH ONLY PRODUCTION SCHEDULE

	Leach					Waste	Total
Year	Ktonnes	Contained Au (g/t)	Recovered Au (g/t)	Contained Au (Koz)	Recovered Au (Koz)	Ktonnes	Ktonnes
1	36,500	0.57	0.44	673	518	26,405	62,905
2	36,500	0.61	0.46	710	543	27,625	64,125
3	36,500	0.67	0.49	782	574	27,183	63,683
4	36,500	0.71	0.45	837	523	60,484	96,984
5	36,500	0.64	0.44	748	522	63,605	100,105
6	36,500	0.53	0.36	625	424	56,409	92,909
7	36,500	0.62	0.41	723	486	23,330	59,830
8	3,758	0.66	0.48	79	58	775	4,533
Total	**259,259**	**0.62**	**0.44**	**5,177**	**3,648**	**285,816**	**545,074**

TABLE 18.4
MILL PROCESS AND HEAP LEACH PAD PROCESS COST AND AU RECOVERY ASSUMPTIONS USED FOR COMBINED HEAP LEACH AND MILL OPERATION.

Rock Type	Oxidation	Mill Process $/ton	Pad Process $/ton	Mill Recovery %	Pad Recovery %
Cambrian	Oxide	7.55	2.71	85.3%	86.0%
	Trace	7.68	3.03	88.8%	65.0%
	Unoxidized	7.69	3.03	75.0%	61.0%
Upper Seds Sunshine	Oxide	7.73	2.89	81.7%	78.0%
	Trace	7.89	3.01	81.6%	75.0%
	Unoxidized	7.89	3.01	82.0%	50.0%
Upper Seds Main	Oxide	7.73	2.89	74.5%	76.0%
	Trace	7.89	3.01	82.8%	55.0%
	Unoxidized	7.89	3.01	82.0%	50.0%
Lower Seds Sunshine	Oxide	7.73	2.89	72.0%	0.0%
	Trace	7.89	3.01	83.0%	0.0%
	Unoxidized	7.89	3.01	77.8%	0.0%
Lower Seds Main	Oxide	7.73	2.89	72.0%	0.0%
	Trace	7.89	3.01	83.0%	0.0%
	Unoxidized	7.89	3.01	77.8%	0.0%
Volcanics Sunshine	Oxide	8.24	3.28	78.1%	53.0%
	Trace	7.69	2.82	87.2%	35.0%
	Unoxidized	7.19	2.35	82.5%	33.0%
Volcanics Main	Oxide	8.24	3.28	78.1%	53.0%
	Trace	7.69	2.82	87.2%	35.0%
	Unoxidized	7.19	2.35	82.5%	33.0%

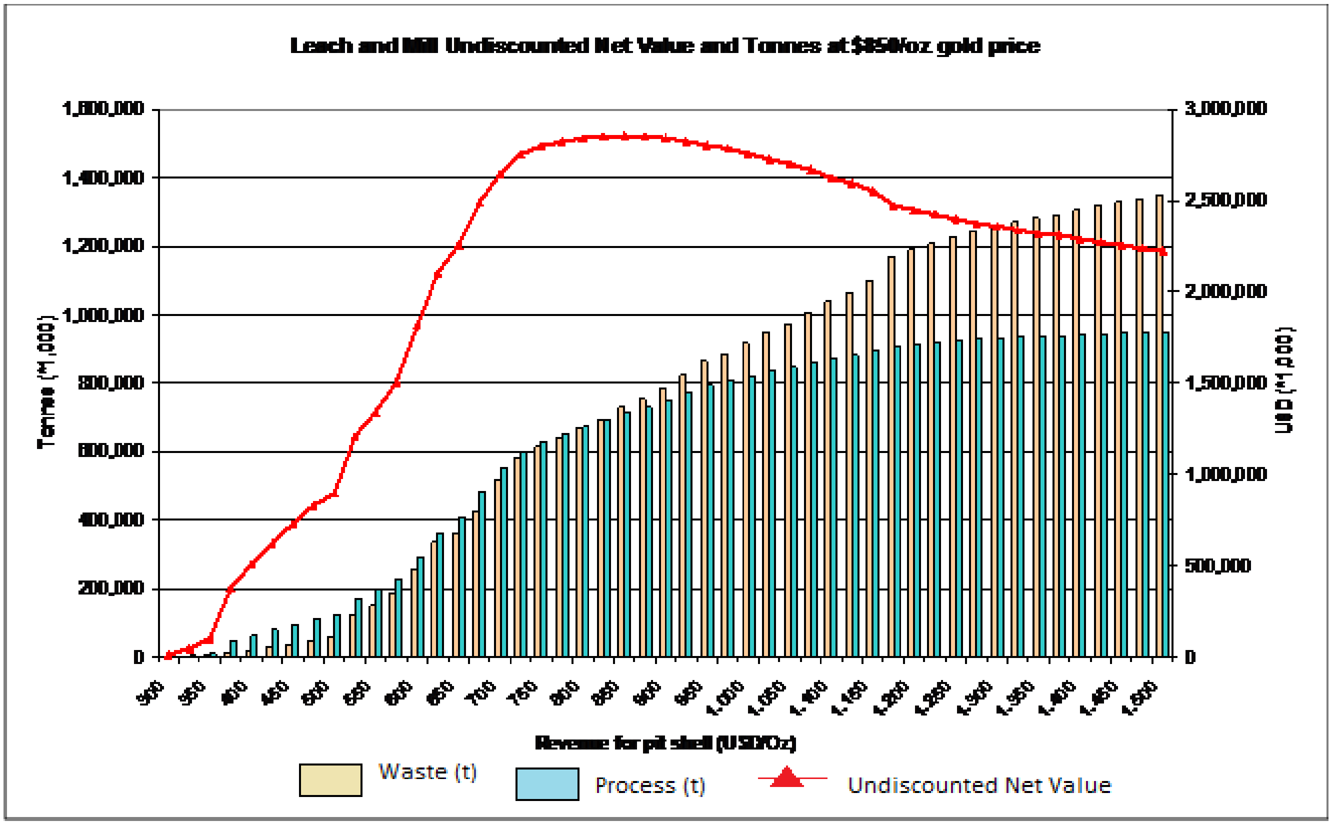

Figure 18.4. Process feed tonnes, waste tonnes and undiscounted net value for Heap Leach and Mill pit shells versus gold price.

The proportion of the feed production for the heap leach and mill configurations coming from the inferred resource was 7.2 % of the total mineralized material.

18.3 Operating and Capital Cost Estimation

The operating and capital cost basis was developed by review of historical data from Alaskan operating mines, similar Arctic mining projects currently under evaluation and work commissioned by ITH as part of the site investigations. The processing operating and capital cost basis was developed by Pennstrom Consulting Inc., as part of the ongoing metallurgical testing and process definition work in progress, which is described in Section 16.0. Mining fleet estimates were prepared by ITH, based on haulage profiles to reach pre-conceptual dump and mill locations, and spreadsheet calculations using the production schedules and equipment performance data. Pre-conceptual Capex estimates for the tailings, heap leach and waste rock facilities were prepared by Knight Piesold Consulting (2010b).

ITH engaged MTB Project Management Professionals, Inc. to review the cost basis and make appropriate adjustments based on experience in previous mining projects, to develop a work breakdown structure (WBS) for the capital cost, and to develop an execution schedule for the capital expenditures. These data are presented in a report to ITH (Bell, 2010) and the results are summarized in the following sections.

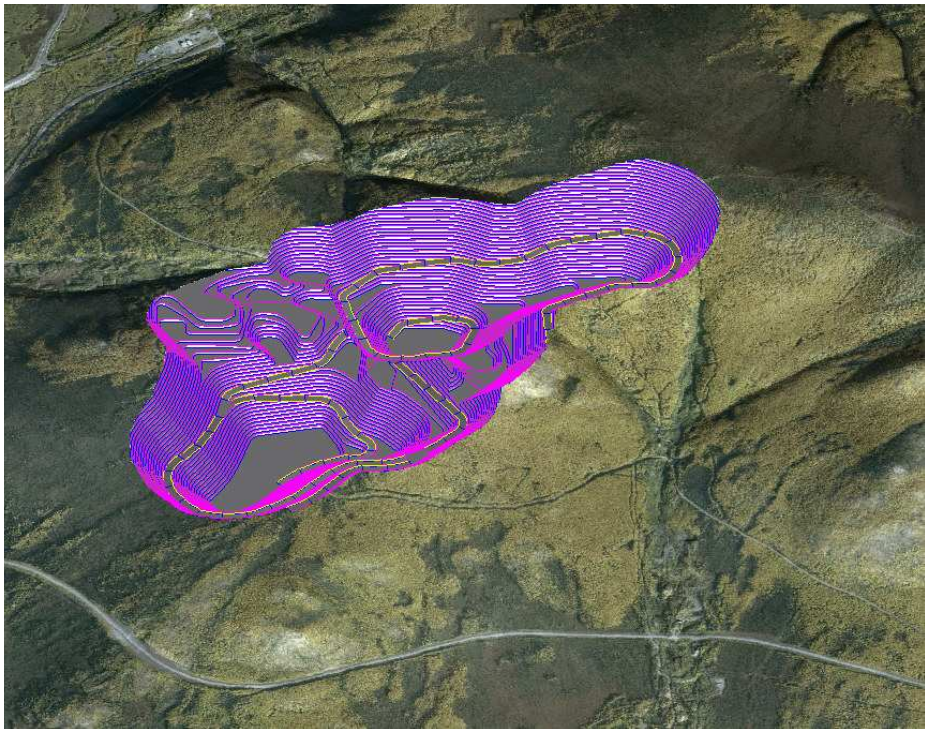

Figure 18.5. Mill and Heap Leach Pit Design shown in a geo-referenced photograph of the Money Knob area.

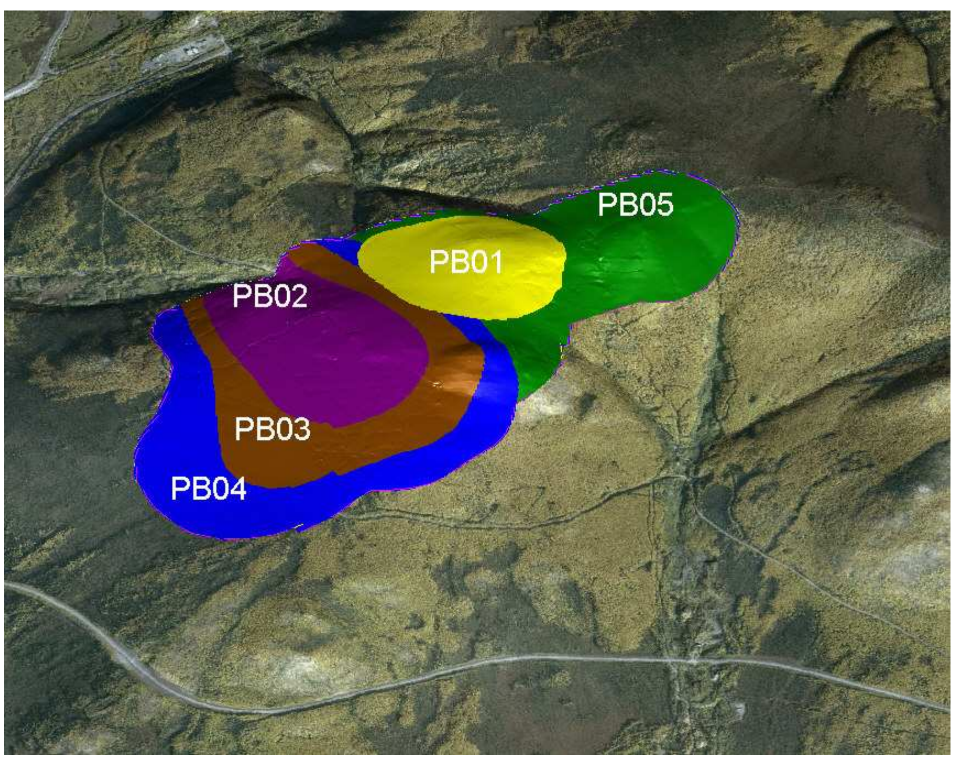

Figure 18.6. Pushback forming the basis of the combined heap leach and mill production schedules.

TABLE 18.5
PRODUCTION SCHEDULE (100 KTPD HEAP LEACH WITH 50 KTPD MILL)

	Leach					Mill					Total Ore			Waste	Total
Pushback	Ktonnes	Contained Au (g/t)	Recovered Au (g/t)	Contained Au (Koz)	Recovered Au (Koz)	Ktonnes	Contained Au (g/t)	Recovered Au (g/t)	Contained Au (Koz)	Recovered Au (Koz)	Ktonnes	Contained Au (Koz)	Recovered Au (Koz)	Ktonnes	Ktonnes
1	34,673	0.57	0.44	637	493	1,826	0.67	0.54	39	31	36,500	676	524	33,582	70,082
2	34,885	0.60	0.46	671	519	1,615	0.66	0.55	34	29	36,500	705	548	32,049	68,549
3	33,962	0.66	0.50	717	546	2,538	0.84	0.69	68	56	36,500	785	602	43,522	80,022
4	19,461	0.63	0.44	394	274	17,039	0.89	0.73	489	397	36,500	883	671	59,940	96,440
5	17,443	0.59	0.44	333	246	19,057	0.79	0.65	483	401	36,500	816	646	59,518	96,018
6	17,914	0.54	0.37	311	212	18,586	0.83	0.69	494	415	36,500	805	627	65,546	102,046
7	16,500	0.52	0.38	274	200	20,000	0.82	0.69	530	445	36,500	804	645	70,136	106,636
8	18,252	0.49	0.36	289	212	18,248	0.75	0.61	437	360	36,500	726	572	76,367	112,867
9	16,500	0.57	0.40	305	215	20,000	0.69	0.56	446	363	36,500	750	577	55,777	92,277
10	18,561	0.57	0.42	342	248	17,939	0.72	0.60	415	345	36,500	758	593	42,441	78,941
11	16,500	0.53	0.36	281	192	20,000	0.70	0.58	450	371	36,500	732	563	30,689	67,189
12	16,353	0.52	0.35	274	186	20,000	0.71	0.58	455	372	36,353	729	558	30,533	66,886
13	17,153	0.57	0.42	316	230	19,347	0.68	0.55	425	343	36,500	741	573	21,927	58,427
14	18,547	0.60	0.43	358	259	17,953	0.67	0.54	384	309	36,500	742	568	19,335	55,835
15	4,858	0.56	0.36	88	56	20,000	0.68	0.54	436	346	24,858	524	402	13,430	38,288
16	3,652	0.51	0.24	60	29	18,348	0.69	0.55	405	325	22,000	465	353	12,591	34,591
17	2,069	0.47	0.18	31	12	19,231	0.64	0.52	397	319	21,300	428	331	10,631	31,931
18	703	0.46	0.16	10	4	19,897	0.65	0.52	414	332	20,600	425	335	7,103	27,703
19	145	0.47	0.17	2	1	19,955	0.69	0.55	443	352	20,100	445	353	4,028	24,128
20						20,000	0.75	0.59	480	378	20,000	480	378	3,254	23,254
21						8,587	0.75	0.58	206	160	8,587	206	160	1,518	10,105
Total	**308,131**	**0.57**	**0.42**	**5,693**	**4,131**	**340,167**	**0.73**	**0.59**	**7,932**	**6,449**	**648,298**	**13,625**	**10,580**	**693,917**	**1,342,215**

TABLE 18.6
PRODUCTION SCHEDULE (100 KTPD HEAP LEACH WITH 100 KTPD MILL)

	Leach					Mill					Total Ore			Waste	Total
Year	Ktonnes	Contained Au (g/t)	Recovered Au (g/t)	Contained Au (Koz)	Recovered Au (Koz)	Ktonnes	Contained Au (g/t)	Recovered Au (g/t)	Contained Au (Koz)	Recovered Au (Koz)	Ktonnes	Contained Au (Koz)	Recovered Au (Koz)	Ktonnes	Ktonnes
1	34,673	0.57	0.44	637	493	1,826	0.67	0.54	39	31	36,500	676	524	33,582	70,082
2	34,885	0.60	0.46	671	519	1,615	0.66	0.55	34	29	36,500	705	548	32,049	68,549
3	33,962	0.66	0.50	717	546	2,538	0.84	0.69	68	56	36,500	785	602	43,522	80,022
4	33,831	0.62	0.44	671	481	32,855	0.84	0.69	890	730	66,686	1,561	1,210	106,200	172,886
5	29,830	0.52	0.37	503	353	36,500	0.84	0.70	987	827	66,330	1,490	1,179	96,690	163,020
6	36,500	0.49	0.35	575	405	36,500	0.73	0.59	854	698	73,000	1,429	1,103	109,081	182,081
7	36,500	0.54	0.37	629	440	36,035	0.69	0.56	803	653	72,535	1,432	1,093	100,271	172,806
8	36,500	0.62	0.45	722	528	36,500	0.70	0.58	820	680	73,000	1,542	1,208	87,552	160,552
9	22,936	0.59	0.41	434	303	36,500	0.68	0.55	798	647	59,436	1,232	950	37,473	96,909
10	6,411	0.51	0.26	105	53	36,500	0.68	0.54	799	638	42,911	904	691	24,311	67,221
11	1,999	0.46	0.17	29	11	36,500	0.65	0.52	759	608	38,499	789	619	14,971	53,470
12	104	0.47	0.17	2	1	36,500	0.72	0.57	845	669	36,604	846	669	6,498	43,102
13						9,798	0.75	0.58	236	184	9,798	236	184	1,717	11,514
Total	**308,131**	**0.57**	**0.42**	**5,693**	**4,131**	**340,167**	**0.73**	**0.59**	**7,932**	**6,449**	**648,298**	**13,625**	**10,580**	**693,917**	**1,342,215**

The capital cost scope was developed to a WBS. This WBS was developed from several historical projects of similar scope. The capital components of the estimate were allocated into three major groupings:

- initial capital;
- deferred capital in the case of delayed mill construction; and
- sustaining capital for both continued incremental investment and replacement.

The operating cost was developed for the mining, the processing and the general and administrative (G&A) groupings.

Costs were defined by the CPM schedule, with an approved feasibility study marking the point of the initiation of capital expenditures; cost prior to the approved feasibility study were considered to be "sunk" costs. Initial capital cost was defined as all cost incurred before startup, which would be when the first mineralized material would be crushed and placed on the heap leach pad. Production year +1 would begin at startup, and would define the beginning of operating cost. Deferred capital cost and sustaining capital cost would occur after year +1 and in production years thereafter during the life of mine (LOM).

The heap leach production was assumed to start in year +1 at a rate of 57 ktpd (57,000 tonnes per day) and ramped up to a nominal 100 ktpd until startup of the milling operation. Thereafter, the heap leach production rate was variable, depending on the pit production schedule necessary to sustain the required mill throughput.

The cost estimate was developed to a baseline established for the Heap Leach with 50ktpd Mill, which is nominally analogous to the Fort Knox mining operation near Fairbanks. This cost basis was subsequently scaled to the alternatives considered (Heap Leach Only and Heap Leach with 100ktpd Mill by factoring.

18.3.1 Estimate Basis and Schedule

The estimate was prepared in Q3 2010 dollars. No forward escalation was included, and the accuracy level is considered to be +/- 35%. Contingency was set at 20% of direct cost for the heap leaching operation and increased to 25% for the more complicated mill and tailings facility construction.

In general, the capital cost is factored from historical project costs for operations in the Alaskan environment. For some WBS accounts, conceptual engineering was quantified and unit cost rates were prepared. In a few WBS accounts, first principle estimating was performed.

Expenditure scheduling was standardized between the cases and the capital schedule is described using the following schedule nomenclature:

- Construction start up at the beginning of year -2;
- Pioneer mining, initial stripping and plant completion in year -1;
- Begin heap leach material placement and gold production in year +1.
- Mill construction (for Cases A and C) begins in year +1;
- Mill start up and gold production in year +4.

18.3.2 Initial Capital Cost

The initial capital cost consisted of cost to be incurred after an approved feasibility study had been prepared. It included all cost up to the start of production, which is defined as when the first mineralized material feed is introduced. The initial capital costs consisted of the scope to construct a heap leach operation of a nominal 100 ktpd leach feed rate, including carbon-in-column processing to produce a doré.

The scope of the initial capital includes contract mine stripping, carbon-in-column process plant construction, material handling, support facilities, freight, design engineering, vendor representatives and commissioning support. It also includes Owner direct and indirect cost and working capital.

18.3.3 Deferred Capital Cost
In all cases which include a milling option, the mill capital cost has been identified separately from sustaining capital. It generally occurs in production years +1 through +3, in order to allow mill production to start at the beginning of year +4.

Deferred capital is the cost to install all of the process plant, support facilities, and Owner cost for milling of the mineralized material and for producing a doré product through gravity and flotation processes. The scope of the work included processing nominally 50-100 kptd of mill feed, with the balance of any oxide material production going to the heap leach pad.

18.3.4 Sustaining Capital Cost
Sustaining capital is cost that is incurred after production starts and includes the incremental capital for expansion of production capability and special production needs, plus replacement capital to sustain capacity.

Sustaining capital consisted of dewatering construction, heap leach pad expansion, tailings facility and waste rock facility expansions, plant mobile equipment replacement, and both mine equipment replacement and major overhauls.

18.3.5 Operating Cost
Operating cost assumptions consisted of mining operating costs, process operating costs and G&A costs. Mining and G&A costs were based on published information on the Fort Knox mine (Quandt, et al, 2008) with some factoring to account for scale of production. Processing costs were based on build-up from flowsheets and factoring for Alaskan historical operating data.

Nominal operating cost assumptions, scaled for impacts of throughput and operating synergies, were:

- Mining cost - $1.45 - $1.37 per tonne process feed or waste;
- Heap Leach Only Processing Cost (nominal) - $3.11 per tonne feed;
- Heap Leach in combination with Milling at 50 ktpd (nominal) - $2.95 per tonne feed;
- Heap Leach in combination with Milling at 100 ktpd -$2.70 per tonne after year +3;
- Mill Processing Cost at 50 ktpd (nominal) - $7.69 per tonne feed;
- Mill Processing Cost at 100 ktpd (nominal) - $7.46 per tonne feed;
- G&A Cost - $24.9 M per year (increased to $34.9 M for years +4 to +9 for the heap leach with 100ktpd mill

Mining cost assumptions were fixed at $1.45 per tonne except where it was scaled for the increased mill throughput to 100ktpd by assuming that 10% of the cost was fixed and 90% was variable. Similarly, G&A cost was scaled for the 100ktpd mill assuming a ratio of 40%/60% fixed and variable.

Mining tonnage shown in year +1 of the production schedules presented in Section 18.2 was spread between year -1 and +1 to acknowledge the requirement for pioneering, initial stripping and production mining ramp-up. It was assumed that the mining start-up would be performed by a contractor at a rate of $2.25 per tonne. Those costs are transferred to the capital scheduled for year -1, but costs to process any mineralized material mined in year -1 occur as operating costs in year +1.

18.3.6 Capital Cost

Capital costs for the three production cases are compared in **Table 18.7**, organized by initial, deferred and sustaining capital. A recovery of consumables stocking associated with first fill assumptions is also listed in the table.

18.4 Economic Modeling

18.4.1 Economic Model Development

A simple financial model was prepared for the Livengood project to get a better understanding of the impacts on the project of the various processing schemes being reviewed. The financial model is based on developing pre-tax and pre-royalty cash flows that can be used for determining comparative NPV and IRR calculations. Constant dollars from Q3 2010 are assumed for all cost information, with no escalation applied. The long term gold price was assumed to be $950 US per Au ounce.

The financial model was derived from the yearly mine schedules. Those schedules incorporate both process recovery assumptions and process operating costs by mineralization type for the processing options under review through the use of the various mineralization types defined in the resource model blocks. The pit optimization assigns recoveries and operating costs to each mineralization type in the mine model and provides a yearly production schedule that considers the economic value of each block in the model. Material that has grade below a selected economic value is determined to be waste and the remainder is process feed. In the processing options that contain both heap leaching and milling, the mine model also economically determines which process scheme is best suited for assignment of the mineralized material to either the heap leaching process or the milling process. Once the mine modeling exercise has been performed for the various options, the yearly mine waste, heap leach tonnes, and mill tonnes and contained grades and recovered grades are tabulated and used as inputs by year in to the economic model. A resulting schedule of recovered gold ounces by year from these values is obtained and is used with the assigned gold price to calculate a revenue by year .

The economic model takes the mine model inputs and calculates a yearly cash flow that is based on the revenue-by-year calculation for gold produced, yearly tonnages for waste and process feed material, contained ounces in the process feed by process (heap leach or mill), recovered ounces in the process feed by process (heap leach or mill), and operating costs for mining, heap leaching, milling, and general and administrative costs. These calculations provide yearly net revenue cash flow numbers.

Capital costs were also estimated and used as inputs to the model. Capital costs were estimated for each processing scenario and for mining fleet requirements for each of the options under review, as discussed previously in Section 18.3. Capital costs were provided as initial, deferred, working, and sustaining capital and were provided as yearly expenditures. These numbers were also introduced into the economic model

TABLE 18.7
COMPARISON OF CAPITAL COSTS AND SCHEDULE FOR THE 3 PRODUCTION CASES.

Case	Initial Capital Year -2	Initial Capital Year -1	Initial Capital Year +1	Total Initial Capital
Heap Leach and 50 ktpd Mill	$ 117,959,726	$ 485,897,274	$ 31,774,000	$ 635,631,000
Heap Leach Only	$ 126,845,346	$ 521,231,654	$ 31,774,000	$ 679,851,000
Heap Leach and 100 ktpd Mill	$ 127,401,326	$ 523,663,674	$ 31,774,000	$ 682,839,000

Case	Deferred Capital Option A Year 1	Deferred Capital Option A Year 2	Deferred Capital Option A Year 3	Total Deferred Capital
Heap Leach and 50 ktpd Mill	$ 112,532,100	$ 225,064,200	$ 412,617,700	$ 750,214,000
Heap Leach Only	$ -	$ -	$ -	$ -
Heap Leach and 100 ktpd Mill	$ 153,998,700	$ 307,997,400	$ 564,661,900	$ 1,026,658,000

Case	Total Sustaining Capital	Spare Parts and Initial Fill Recoveries
Heap Leach and 50 ktpd Mill	$ 503,595,978	$ (21,818,197)
Heap Leach Only	$ 153,481,878	$ (10,102,000)
Heap Leach and 100 ktpd Mill	$ 578,476,440	$ (25,631,000)

Once all of the data had been input, NPVs and IRRs was calculated for each processing scenario for comparison and preliminary valuation purposes.

18.4.2 Processing Options Reviewed

The following discusses the various process options reviewed for this preliminary economic assessment.

18.4.2.1 Heap Leach Only at 100,000 tpd

This process alternative assumes a heap leach only facility with material placed on the pad at a rate of 100,000 tpd. Initial capital is expended in years -2 and -1, with no deferred capital required. Sustaining capital was spread over the LOM. Gold production would begin in year +1.

18.4.2.2 50,000 tpd Mill (gravity and flotation concentration with CIL) and Up to 100,000 tpd Heap Leach

This process alternative assumed an average throughput of 53,500 tpd in a mill utilizing a scalping gravity circuit, flotation, flotation concentrate regrinding, and CIL processing of the reground concentrate, for the higher grade and "preg-robbing" mill feed, combined with a heap leaching operation for the lower grade oxide, non-preg robbing leach pad feed (as described in Section 16). The flotation tails from the milling process would not be leached and would report directly to a benign tailings storage facility. The leached concentrate residue would undergo a cyanide destruct process and would report to a separate, smaller, concentrate leach tails storage facility.

Results from the mine schedule indicate that the first three years of production, years +1, +2, and +3, would provide material for heap leaching only, with minimal tonnage being stockpiled for the mill process. To reduce capital expenditures in the early years, capital expenditure for the mill would be deferred to year +1 and only the heap leach and mine capital would be spent in pre-production years -1 and -2. Mill production revenue and operating costs begin with production from the mill facility in year +4.

18.4.2.3 **100,000 tpd Mill (Flotation with Concentrate CIL) and up to 100,000 tpd Heap Leach**

An alternative with increased mill throughput was developed to examine the effect of reduced mine life on financial performance. The mill is a 100,000 tpd facility utilizing a scalping gravity circuit, flotation, flotation concentrate regrinding, and CIL processing of the reground concentrate, for the higher grade and "preg-robbing" mill feed, combined with a heap leaching operation up to 100,000 tpd for the lower grade oxide, nonpreg-robbing leach pad feed. The flotation tails from the milling process would not be leached and would report directly to a benign tailings storage facility. The leached concentrate residue would undergo a cyanide destruct process and would report to a separate, smaller, concentrate leach tails storage facility.

Results from the mine model indicate that the first three years of production, years +1, +2 and +3, would provide material for heap leaching only, with minimal tonnage being stockpiled for the mill process. To reduce capital expenditures in the early years, capital for the mill would be deferred until year +1 and only the heap leach and mine capital would be spent in pre-production years -1 and -2. Mill production revenue and operating costs would begin with production from the mill facility in year +4.

18.4.3 Preliminary Pre-Tax and Pre-Royalty Financial Results

The results of the financial calculations for a long term gold price assumption of $950 per Au ounce for the three process alternatives are compared in **Table 18.8**. The assumption of $950 per Au ounce is consistent with current long term price outlooks which reflect the increasing gold price trend in the markets. A lower price was used in the pit optimizations to assure a margin above cost in the production plan at the long term gold price.

The financial results listed in **Table 18.8**, indicate a positive performance for all three alternatives. The heap leach only operation is projected to deliver a strong IRR at 26.9%, due to a lower operating cost and capital cost per ounce than the combinations heap leach and mill alternatives. The heap leach and mill alternatives are projected to have lower IRRs due to higher operating cost per ounce and capital costs associated with the mill. The 100 ktpd mill improves the IRR and NPV for the same production, due to some economy of scale, and a shorter mine life.

The sensitivity of Internal Rate of Return (IRR) to gold price is listed in **Table 18.9A and the sensitivity of NPV@5% to gold price is listed in Table 18.9B**. Sensitivity of NPV at discount rates ranging from 0% to 10%, to changes in operating cost, capital cost and process gold recovery are listed in **Tables 18.10, 18.11** and **18.12**, respectively. For the operating cost, capital cost and process gold recovery sensitivity calculation, the driving parameter was changed as a percentage of the base analysis value over a range of +/- 15%.

The IRR changes approximately +/- 7% for a variation of +/- 15% in the input cost assumption. The sensitivity to gold price and process recovery is larger, as illustrated in **Table 18.9a** and **18.12**, respectively. The IRR changes +/- 16-20% from the IRR at the $950 price over the range in gold price of $750 - $1200 ounce. The sensitivity to process recovery emphasizes the importance of the on-going metallurgical test work.

The PA is preliminary in nature, and is based on technical and economic assumptions which will be evaluated in the Pre-feasibility Study. The PA is based on the Livengood in-situ resource model (June, 2010) which consists of material in both the indicated and inferred classification. Inferred mineral resources are considered too speculative geologically to have technical and economic considerations applied to them. The current basis of project information is not sufficient to convert the in-situ mineral resources to Mineral Reserves, and mineral resources that are not mineral reserves do not have demonstrated economic viability. Accordingly, there can be no certainty that the results estimated in this PA will be realized. The PA results are only intended as an initial, first-pass review of the potential project economics based on preliminary information.

TABLE 18.8
COMPARION OF FINANCIA PERFORMANCE FOR THE 3 PROCESS OPTIONS AT LIVENGOOD FOR A GOLD PRICE OF $950 PER OUNCE.

Parameter		Heap Leach Only	Heap Leach and 50 ktpd Mill	Heap Leach and 100 ktpd Mill
IRR	%	**26.9%**	**15.4%**	**18.5%**
NPV @0.0%	$1000	**$ 915,338**	**$ 1,982,082**	**$ 2,236,376**
NPV @5.0%	$1000	$ 579,103	$ 813,143	$ 1,112,868
NPV @7.5%	$1000	$ 455,882	$ 495,034	$ 759,768
NPV @10.0%	$1000	$ 354,531	$ 275,370	$ 496,163
Initial Capex	$1000	$ 679,851	$ 635,631	$ 682,839
Deferred Capex	$1000	-	$ 750,214	$ 1,026,658
Sustaining Capex	$1000	$ 153,482	$ 503,596	$ 578,476
Gold recovered-oz		3,648	10,580	10,580
Cash operating cost	$/oz	$486	$ 559	$534
Total cost	$/oz	$704	$ 739	$ 734
Stripping ratio		1.10	1.07	1.07
LOM mill Au recovery	%	-	81.3%	81.3%
LOM leach Au recovery	%	70.5%	72.6%	72.6%

TABLE 18.9A
SENSITIVITY OF INTERNAL RATE OF RETURN (IRR) TO GOLD PRICE

Gold Price	Heap Leach Only	Heap Leach with 50 ktpd Mill	Heap Leach with 100 ktpd Mill
750	6.50%	-0.8%	1.30%
800	12.20%	3.8%	6.10%
850	17.40%	7.9%	10.50%
900	22.30%	11.7%	14.60%
950	26.90%	15.4%	18.50%
1000	31.30%	18.8%	22.20%
1050	35.50%	22.4%	25.80%
1100	39.60%	25.8%	29.30%
1150	43.50%	29.1%	32.60%
1200	47.40%	32.5%	35.90%
1500	68.40%	52.3%	54.40%

TABLE 18.9B
SENSITIVITY OF NET PRESENT VALUE AT 5% DISCOUNT RATE (NPV@5%) TO GOLD PRICE - K $US

Gold Price	Heap Leach Only	Heap Leach with 50 ktpd Mill	Heap Leach with 100 ktpd Mill
750	$ 34,428	$ (380,544)	$ (266,376)
800	$ 170,597	$ (82,122)	$ 78,435
850	$ 306,766	$ 216,299	$ 423,246
900	$ 442,934	$ 514,721	$ 768,057
950	$ 579,103	$ 813,143	$ 1,112,868
1000	$ 715,272	$ 1,111,565	$ 1,457,679
1050	$ 851,441	$ 1,409,987	$ 1,802,490
1100	$ 987,610	$ 1,708,409	$ 2,147,301
1150	$ 1,123,779	$ 2,006,831	$ 2,492,112
1200	$ 1,259,948	$ 2,305,252	$ 2,836,923
1500	$ 2,076,961	$ 4,095,783	$ 4,905,789

TABLE 18.10
SENSITIVITY OF NPV ($1000) FOR DISCOUNT RATES OF 0, 5, 7.5 AND 10% TO OPERATING COST

Variance from Nominal	Heap Leach Only					Heap Leach and 50 ktpd Mill					Heap Leach and 100 ktpd Mill				
	IRR	NPV 0%	NPV 5%	NPV 7.5%	NPV 10%	IRR	NPV 0%	NPV 5%	NPV 7.5%	NPV 10%	% IRR	NPV 0%	NPV 5%	NPV 7.5%	NPV 10%
+15%	20.4%	649,390	382,222	284,932	205,397	9.3%	1,093,978	310,962	104,528	(34,231)	12.3%	1,389,432	564,045	310,962	125,602
+10%	22.6%	738,039	447,849	341,895	255,109	11.4%	1,390,013	478,355	234,697	68,969	14.4%	1,671,747	746,986	460,564	249,122
+5%	24.8%	826,688	513,476	398,858	304,820	13.4%	1,686,047	645,749	364,866	172,170	16.5%	1,954,062	929,927	610,166	372,642
0%	26.9%	915,338	579,103	455,822	354,531	15.4%	1,982,082	813,143	495,034	275,370	18.5%	2,236,376	1,112,868	759,768	496,163
-5%	29.0%	1,003,987	644,730	512,785	404,242	17.2%	2,278,116	980,537	625,203	378,570	20.4%	2,518,691	1,295,809	909,371	619,683
-10%	30.9%	1,092,636	710,358	569,748	453,953	19.1%	2,574,151	1,147,931	755,372	481,771	22.3%	2,801,006	1,478,750	1,058,973	743,203
-15%	32.9%	1,181,285	775,985	626,712	503,664	20.8%	2,870,186	1,315,325	885,541	584,971	24.1%	3,083,320	1,661,691	1,208,575	866,723

TABLE 18.11
SENSITIVITY OF NPV ($1000) FOR DISCOUNT RATES OF 0, 5, 7.5 AND 10% TO CAPITAL COST

Variance from Nominal	Heap Leach Only					Heap Leach and 50 ktpd Mill					Heap Leach and 100 ktpd Mill				
	IRR	NPV 0%	NPV 5%	NPV 7.5%	NPV 10%	IRR	NPV 0%	NPV 5%	NPV 7.5%	NPV 10%	IRR	NPV 0%	NPV 5%	NPV 7.5%	NPV 10%
+15%	21.2%	796,619	473,673	356,076	259,945	11.6%	1,706,704	592,646	295,311	93,235	14.1%	1,901,791	839,069	510,268	267,804
+10%	22.9%	836,192	508,816	389,325	291,474	12.7%	1,798,497	666,145	361,885	153,947	15.4%	2,013,320	930,335	593,435	343,924
+5%	24.9%	875,765	543,960	422,573	323,002	14.0%	1,890,289	739,644	428,460	214,658	16.9%	2,124,848	1,021,602	676,602	420,043
0%	26.9%	915,338	579,103	455,822	354,531	15.4%	1,982,082	813,143	495,034	275,370	18.5%	2,236,376	1,112,868	759,768	496,163
-5%	29.1%	954,910	614,247	489,070	386,059	16.9%	2,073,874	886,642	561,609	336,082	20.2%	2,347,905	1,204,134	842,935	572,282
-10%	31.6%	994,483	649,390	522,318	417,588	18.6%	2,165,667	960,141	628,183	396,793	22.1%	2,459,433	1,295,400	926,102	648,401
-15%	34.2%	1,034,056	684,534	555,567	449,117	20.6%	2,257,459	1,033,641	694,758	457,505	24.2%	2,570,962	1,386,666	1,009,269	724,521

TABLE 18.12
SENSITIVITY OF NPV ($1000) FOR DISCOUNT RATES OF 0, 5, 7.5 AND 10% TO PROCESS GOLD RECOVERY

Variance from Nominal	Heap Leach Only					Heap Leach and 50 ktpd Mill					Heap Leach and 100 ktpd Mill				
	IRR	NPV 0%	NPV 5%	NPV 7.5%	NPV 10%	IRR	NPV 0%	NPV 5%	NPV 7.5%	NPV 10%	IRR	NPV 0%	NPV 5%	NPV 7.5%	NPV 10%
+15%	39.0%	1,432,617	967,185	794,825	652,208	25.3%	3,452,919	1,663,645	1,164,389	812,622	28.7%	3,736,465	2,095,579	1,567,959	1,167,306
+10%	35.1%	1,260,191	837,824	681,824	552,983	22.0%	2,962,640	1,380,145	941,271	633,538	25.4%	3,236,435	1,768,009	1,298,562	943,592
+5%	31.1%	1,087,764	708,464	568,823	453,757	18.7%	2,472,361	1,096,644	718,153	454,454	22.0%	2,736,406	1,440,438	1,029,165	719,877
0%	26.9%	915,338	579,103	455,822	354,531	15.4%	1,982,082	813,143	495,034	275,370	18.5%	2,236,376	1,112,868	759,768	496,163
-5%	22.5%	742,911	449,743	342,820	255,305	11.9%	1,491,803	529,642	271,916	96,286	14.8%	1,736,347	785,297	490,372	272,448
-10%	17.9%	570,484	320,382	229,819	156,079	8.3%	1,001,523	246,142	48,798	(82,798)	10.9%	1,236,317	457,727	220,975	48,733
-15%	13.0%	398,058	191,022	116,818	56,853	4.5%	511,244	(37,359)	(174,320)	(261,882)	6.7%	736,288	130,157	(48,422)	(174,981)

analysis value over a range of +/- 15%. per ounce. **Table 18.8** compares the IRR for the various cases and the sensitivity to gold price.

18.4.4 Conceptual Livengood Project Schedule

The heap leach option has emerged as a very attractive, initial stage for mine development, because it is less complex from the permitting, processing design, and construction standpoints. In order to accelerate evaluation of this option, the Pre-feasibility Study, begun in June 2010, has been divided into two phases for delivery of results in 2011:

- Phase 1 - completion of a PFS for a heap leaching operation by mid-year 2011.
- Phase 2 - completion of a PFS by year-end 2011 that examines the potential for eventual expansion of the operation to include a mill.

For the purposes of this PA, it has been assumed that upon completion of the PFS for the heap leach operation, a process leading up to submission of permit applications would begin. This process would include internal optimization and discussions of the conceptual project configuration and plan with regulatory agencies and the public, preparatory to submission of the permit applications. Feasibility studies would then be in parallel with the permitting process, allowing a construction decision upon receipt of permits and subsequent initial production in 2016.

18.4.5 Preliminary Assessment Conclusions

The Preliminary Assessment of a mining project at Livengood indicates good potential for an economically viable project at the current projection of long term gold price and for the cost and recovery assumptions used in the analysis. The evaluated project configurations are based on large scale mining production from an open pit, with productions rates in the upper range for current gold mining operations. This approach is considered consistent with current understanding of the in-situ resource and the relatively large spacing of the drilling data. Importantly, further infill drilling performed during the summer of 2010 may indicate the existence of continuous zones with higher grade. If true, the current mining assumptions may be altered.

The sensitivity analysis indicates that the potential financial performance is most sensitive to changes in the gold price and gold recovery assumptions. The projected mining operation has relatively high cost per ounce due to the relatively low grade, so changes in price/recovery have large impacts on the projected performance. The importance of the on-going metallurgical work at Livengood is emphasized by this conclusion. The project will perform more detailed column leach testing to verify heap leach recovery assumptions, and continue metallurgical testing of the leach recovery of flotation concentrates, particularly from the lower sediments and volcanic unit, to verify the mill performance assumptions. A metallurgical engineering contract is planned to begin in October 2010, as new sample materials become available for the required testing.

Although the internal rate of return (IRR) and NPV results from the Preliminary Assessment are very sensitive to gold price assumptions, the project performance remains relatively robust down to $850 per ounce. At gold prices greater than the selected $950 long term base price, the projected financial performance is very strong. The leverage to gold price is large for all of the processing alternatives, but especially for the combined heap leach and milling, which have substantially greater gold production than the heap leach alone. Although the heap leaching alternative is projected to have attractive financial performance, the combined operation must continue to be studied because of the potential to produce substantial cash profits if gold prices remain at current levels. Heap leaching alone does not capture the full potential of the deposit to yield substantial financial profits.

The projected financial performance was less sensitive to capital and operating cost assumptions.

Based on the current evaluations, it is recommended that ITH pursue the Heap Leach Only alternative on the basis that it would provides the earliest gold production, even though it does not address the full potential of the deposit. The potential for the addition of a mill should continue to be evaluated, because it will allow exploitation of the full resource. This is a first pass assessment. Future changes in the price of gold and improved resource modeling may have significant impact on factors outlined above resulting in changed recommendations.

The PA is preliminary in nature, and is based on technical and economic assumptions which will be evaluated in the Pre-feasibility Study. The PA is based on the Livengood in-situ resource model (June, 2010) which consists of material in both the indicated and inferred classification. Inferred mineral resources are considered too speculative geologically to have technical and economic considerations applied to them. The current basis of project information is not sufficient to convert the in-situ mineral resources to Mineral Reserves, and mineral resources that are not mineral reserves do not have demonstrated economic viability. Accordingly, there can be no certainty that the results estimated in this PA will be realized. The PA results are only intended as an initial, first-pass review of the potential project economics based on preliminary information.

19.0 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.

20.0 Interpretation and Conclusions

Exploration work over the past 5 years and initial efforts in the Pre-feasibility Study by ITH have resulted in establishing the presence of 15.7M oz of Indicated and 4.9 Moz of inferred in-situ resources (cut-off grade 0.3 g//t) as outlined in section 17.

Pre-feasibility Study for this resource is currently in progress. This report provides an overview of the geological, exploration, metallurgical, engineering, and pit planning work that has been completed to date. The **PA** is an initial effort to outline the basic framework of how gold will be mined, mineralized material processed, and recovery achieved. As more information becomes available, it is envisioned that these plans will be fine tuned and will mature into a complete mining plan.

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. Anomalous gold in soil samples occurs in a northeast trend over an area of approximately 6 x 2 km with a principal concentration of surface anomalies in a smaller area measuring approximately 2.3 x 1.1 km. Previous drilling by 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 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 initial exploration in this location 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.

An updated resource estimate has been calculated and is based on all drill data through May 31, 2010. This new estimate includes the addition of data from 56 drill holes received after completion of the March 2010 resource estimate. The current resource estimate increases the total tonnes and ounces in the Indicated category and reduces the number of ounces and tonnes in the Inferred category for cutoff grades of 0.3, 0.5, and 0.7 g/t Au. This change is due to addition of newly defined resources in the SW Zone and between the Core and Sunshine Zones.

Comparison of block model sections (**Figures 20.1 to 20.6**) with geologic sections interpreted by ITH geologists (**Figures 7.8-7.12**) reveals good correspondence. 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 at Money Knob and the surrounding area. Dedicated drilling has continuously enlarged the resource over the past several years. Current metallurgical studies are underway and results indicate that gold is recoverable through heap leach, and combined mill, CIP, CIL, gravity, and flotation techniques.

Preliminary designs for an open pit mine have been developed on the basis of incremental revenue optimization using the in-situ resource block model and projected operating costs. The optimization produced pit shells for both a constrained pit limited to production of the oxidized portion of the resource and for an unconstrained pit which included both oxidized and sulphide rich portions of the resource. Those shells assumed a 45 degree pit slope, because geotechnical data is not available to establish a more rigorous basis. Pit designs were produced to include the effect of additional waste production through inclusion of ramps and other access facilities in the pit.

A series of push-backs were generated to help in the generation of a mining production schedule. Production rates were proposed by ITH and tested against required vertical advance rates in the push-backs to assure reality. ITH set nominal mine production rates at 100,000 mineral tonnes per day for the stand alone heap leach mine. For the combined heap leach and mill alternatives, ITH specified nominal production rates of 50,000 and 100,000 tonnes per day for the mill, with heap leach production governed by the requirement to achieve the mill requirements.

Metallurgical data have been assembled as the basis for pre-conceptual process flow sheet. Processing facilities have been conceptualized on the basis of the flow sheets, and were used to estimate construction and equipment costs. Process metallurgical recovery assumptions have been derived based on the metallurgical data.

Project configurations have been developed for the mining production schedules. These configurations included capital costs for process facilities, and tailings, waste dump, and heap leach facilities. Mining fleet estimates have been developed for the different production schedules. Sustaining capital costs have been estimated for the facilities and equipment.

A **PA** has been conducted. The result of this assessment indicates that the Livengood project has the potential to be a commercial facility in several different configurations. The **PA** is based on estimates of operating costs and capital costs that must be further validated. The gold production projected in the **PA** is based on the in-situ resource model and estimates of mining recoverable resources at a 0.3 g/t cut-off. Infill drilling is on-going at Livengood in the Summer 2010 program to increase confidence in mineral continuity and to test the resource model predictions.

> **The PA is preliminary in nature, and is based on technical and economic assumptions which will be evaluated in the Pre-feasibility Study. The PA is based on the Livengood in-situ resource model (June, 2010) which consists of material in both the indicated and inferred classification. Inferred mineral resources are considered too speculative geologically to have technical and economic considerations applied to them. The current basis of project information is not sufficient to convert the in-situ mineral resources to Mineral Reserves, and mineral resources that are not mineral reserves do not have demonstrated economic viability. Accordingly, there can be no certainty that the results estimated in this PA will be realized. The PA results are only intended as an initial, first-pass review of the potential project economics based on preliminary information.**

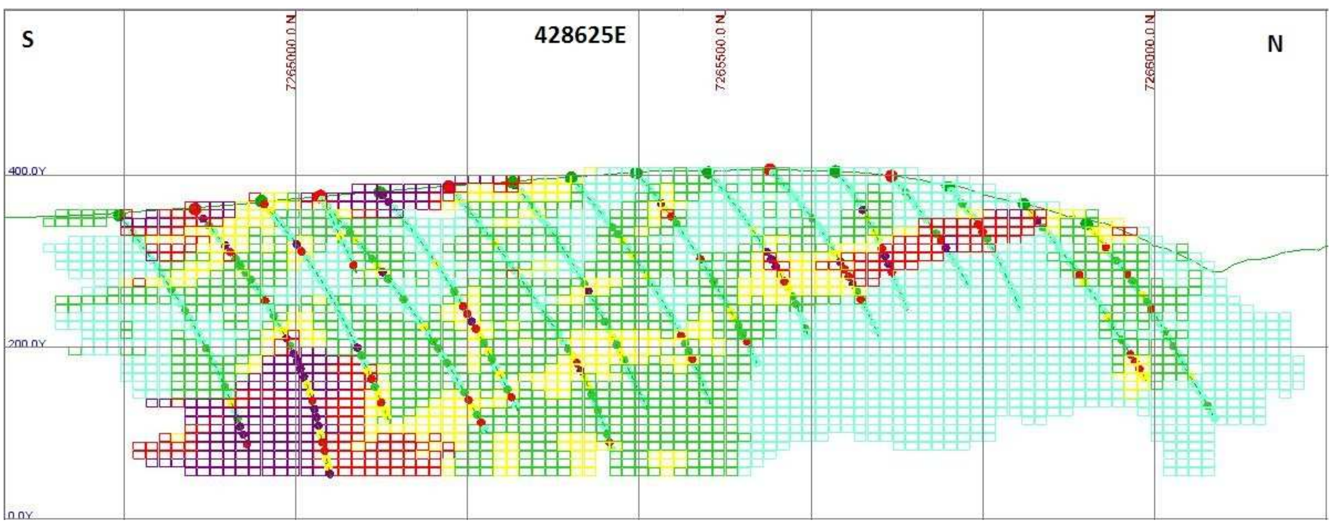

Figure 20.1. Block model for section 428625 E. Grid squares are 200m.

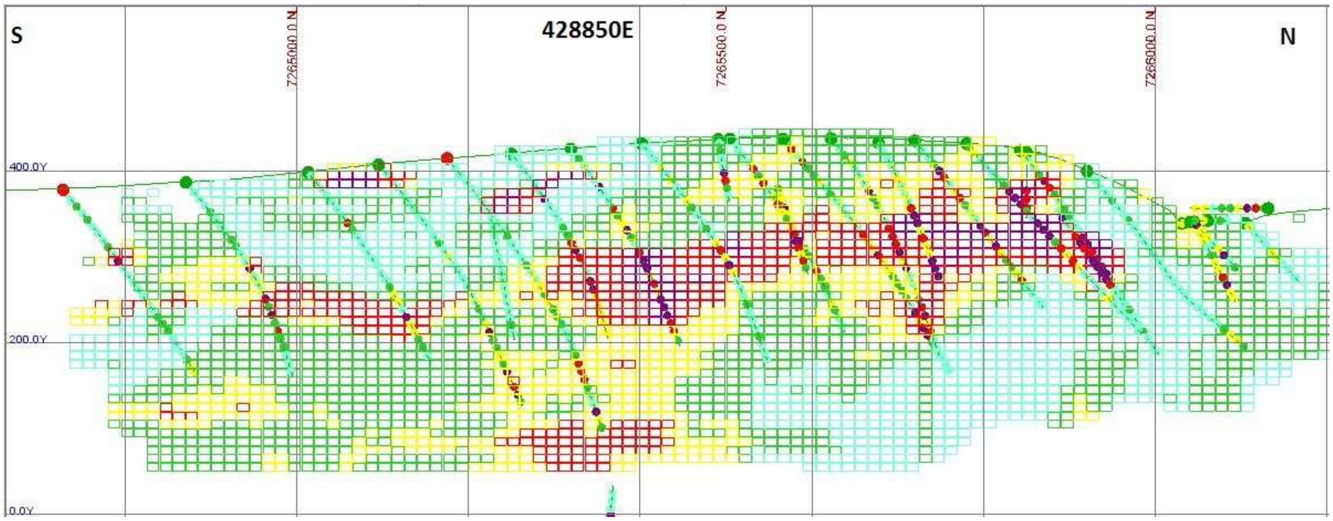

Figure 20.2. Block model for section 428850 E. Grid squares are 200m.

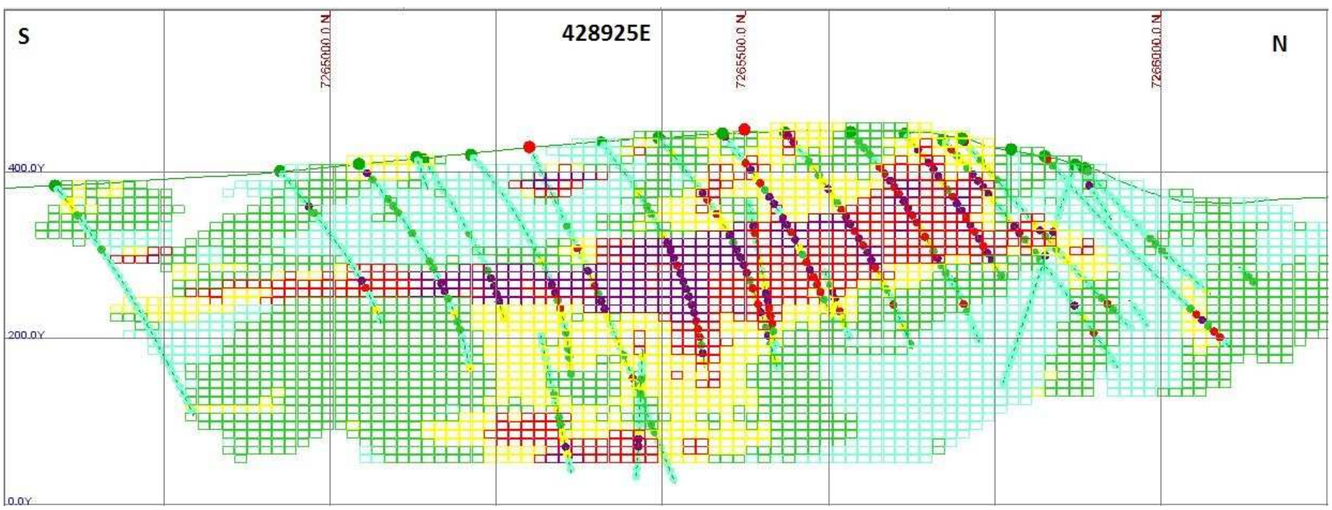

Figure 20.3. Block model for section 428925 E. Grid squares are 200m.

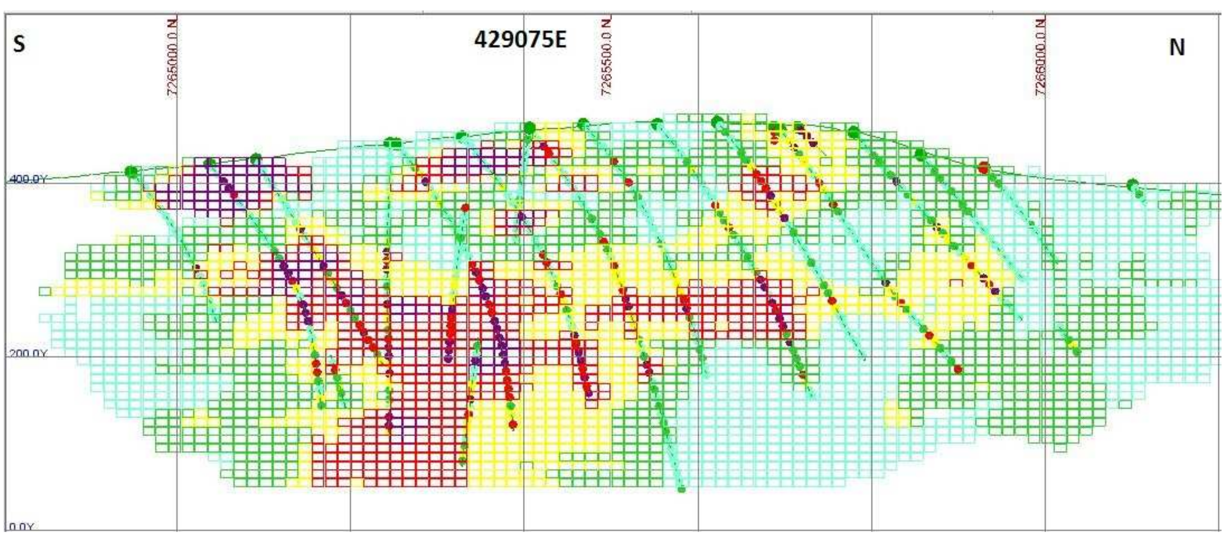

Figure 20.4. Block model for section 429075 E. Grid squares are 200m.

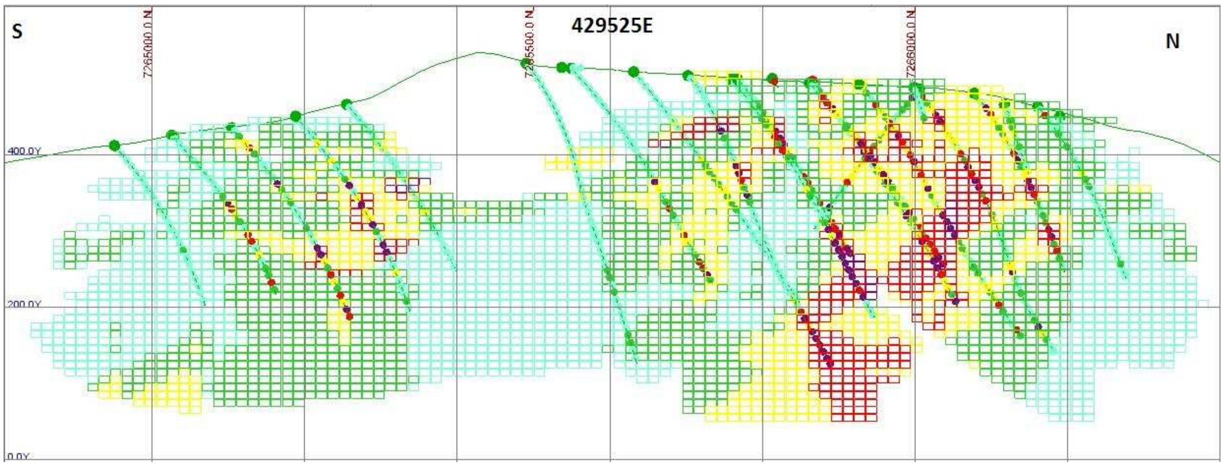

Figure 20.5. Block model for section 429525 E. Grid squares are 200m.

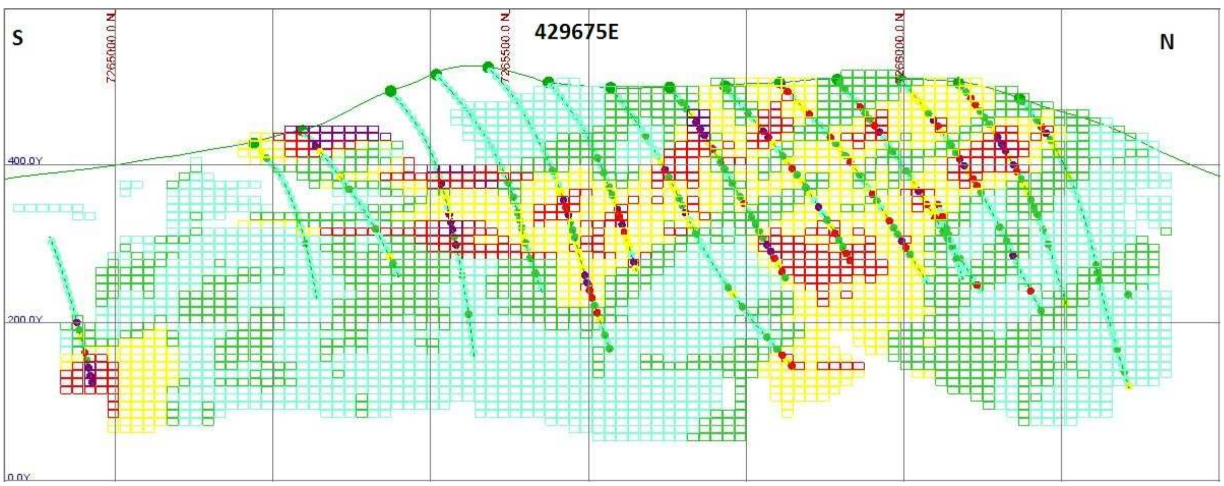

Figure 20.6. Block model for section 429675 E. Grid squares are 200m.

21.0 Recommendations

21.1 Recommended Exploration and Prefeasibility Program

Exploration of the Livengood project should continue with the aim of advancing the project toward a pre-feasibility status. During Q1 and Q2 of 2010, ITH began the transition from exploration to prefeasibility assessment with the addition of a Chief Operating Officer (Carl Brechtel) and an Alaska based Project Manager (Karl Hanneman). Subsequently, ITH has added a Site Operations Manager (Richard Moses) and a Technical Services Manager (Keith Malone). ITH will continue to expand the Fairbanks based technical management team, adding capacity in the financial control, health and safety, infrastructure engineering, environmental management and community engagement areas. A two story office facility has been rented to house the Fairbanks Team and provide workspace for contractor technical personnel when they visit the site. This group will act as owner representatives in the direction and management of the specialty consulting and engineering contractors required to conduct Pre-feasibility and Feasibility Studies.

ITH plans to continue drilling, with another 40,000 m planned for the second half of 2010 to accomplish this goal. In addition, it will continue environmental baseline studies, community engagement activities, and metallurgical studies, and it will initiate mine infrastructure and engineering studies. Engineering contracts are underway for infrastructure siting and evaluation, and are being developed for the Metallurgical Plant design. The drilling program is being expanded by addition of helicopter transportable core rigs to test outlying gold anomalies as part of condemnation efforts. Characterization work will continue into the deep winter in order to meet the requirements of the PFS scheduling. Groundwater hydrogeologic testing has been on-going, with packer testing in core holes, and jet testing of RC holes being drilled in the Summer 2010 program. A pump test is being configured to stress the aquifers in the open pit area and is planned for late October 2010. Activities that will help advance the project include those listed below.

1. Focus infill drilling on areas where Inferred resource blocks can be converted to Indicated resources.
   a. extend drilling into areas where the search neighbourhood has extrapolated Inferred resource blocks laterally beyond existing drilling; and
   b. drill infill holes between existing patterns to greater depth to test the extrapolation of inferred resource below the general base of the existing drilling.

2. Drill close spaced holes to define a variographic cross in order to demonstrate mineralization continuity and to better determine the drill spacing required to convert indicated resources into measured resources.

3. Drill core holes to gather sample material for advance metallurgical and comminution testing.

4. Continue to drill holes for hydrologic testing and monitoring to support open pit mine design, and for groundwater water quality monitoring.

5. Continue and advance metallurgical, mineralized material characterization, and mineral

processing studies. This should include:
   a. evaluation of flotation methods;
   b. production of sufficient volumes of flotation and gravity concentrates to evaluate treatment options for the concentrates;
   c. quantification of the distribution of preg-robbing carbon in the deposit; and
   d. use SEM studies to better characterize gold mineralization, its exact mineral association, and relationship to gangue.

6. 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. down dip of currently identified mineralization in the Sunshine Zone;
   d. to the southwest along the trend of the surface geochemical anomaly; and
   e. to the south of Money Knob and southeast of the Core Zone.

7. Assess geotechnical characteristics of the mineralized zone and potential pit walls using core data, and perform rock mechanics testing of core samples.

8. Appoint a geomechanical design consultant and initiate pit slope stability studies.

9. Continue the sterilization process for land that might be covered by facilities. This should start with surface geochemical surveys to be followed up with drilling on potentially mineralized zones.

10. Perform infrastructure siting and alternatives ranking assessments, and develop prioritized plans for infrastructure geotechnical characterization.

11. Continue and expand environmental base line studies including:
    a. expansion of surface water quality studies to include additional stations to cover expanded land holdings and measurement of stream flow;
    b. expansion of aquatic macro fauna studies to cover expanded land holding;
    c. initiation of subsurface hydrological investigations; and
    d. installation of meteorological stations on the property.

12. Examine the potential for mill processing of a higher grade component of the heap leach material with agglomeration of the tailings and placement in the heap leach pad.

13. Perform a detailed alternatives assessment for optimization of the project configuration.

ITH should conduct a two phase Pre-feasibility Study with the projected completion of Phase I - Heap Leach Operation in July 2011, and the projected completion of Phase II - Milling Operation in December 2011. Drilling operations should expand to include condemnation and geotechnical investigations for the PFS. Metallurgical testing for Phase I should consist of additional column leach tests at 1/2 inch, 1.5 inch and run-of-mine top sizes that are scheduled to begin in October 2010. Engineering studies, required to support the Phase I PFS should be:

- Metallurgical engineer to design the CIC process plant (out for tender);
- Site location, geotechnical assessment and design of the heap leach pad waste dumps and water storage facilities (underway);
- Site infrastructure, reticulation and road corridor placement and design (to be defined);
- Geotechnical design of pit slopes (to be defined);
- Open pit design and mining production scheduling (underway);
- Open pit dewatering, site water balance and storage requirements (underway); and
- Construction cost and production operating cost estimation (to be defined).

21.2 Budget for 2011

ITH has proposed expenditure of approximately CDN $37.5 million dollars in FY 2011 for further evaluation of the Livengood project (**Table 21.1**). This budget will be allocated to drilling, geological and geotechnical analysis of the deposit, metallurgical and comminution studies, facilities site planning, environmental and social base line studies, and completion of a Preliminary Economic Assessment. The budget is significant, but appropriate for the studies and drilling planned and feasible within the time allocated. ITH has sufficient funds to accomplish this goal. The authors recommend implementation of this program in order to accomplish ITH's goal of advancing the Livengood project.

TABLE 21.1
YEAR ENDED MAY 31, 2012 PROJECT BUDGET

Expenditure	$M CAD	Comments
Project Admin	10.4	Admin, project management, Claim and lease fees, materials, purchase agreements, permits, office, salaries, travel, reporting, permitting
Geological and field Operations	15.3	Operations, contract/consulting fees for geologic and geotechnical studies, other field activities
Metallurgical Studies	3.9	Colum leach testing, comminution testing, flow sheet specification, plant design
Infrastructure and Engineering	3.5	Site alternatives assessment, geotechnical investigations, design
Environmental and Community Engagement	2.9	Baseline studies, community engagement and metallurgical testing
Mining Studies	0.9	Geotechnical design, pit optimization scheduling, design and equipment
Project Integration	0.6	Integration of technical reports into PFS documents, risk assessment, front end loading planning

TOTAL	**37.5**	

Preliminary budgets have been developed to carry the project through the Feasibility Studies and Permitting Process

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23.0 Date and Signature Page

The effective date of this technical report, entitled "**November 2010 Summary Report on the Livengood Project, Tolovana District, Alaska**" is June 1, 2010.

Dated: November 1, 2010

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

(signed) *William Pennstrom, Jr.*
William Pennstrom, Jr. M.A., QP-MMSA

(signed) *Roscoe J. Bell*
Roscoe J. Bell, ProfGradIMM

(signed) *Quinton de Klerk*
Quinton de Klerk, AusIMM

24.0 Certificates of Authors

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 (Professional Geoscientist 19706)
   b. Institute of Mining, Metallurgy and Materials (Professional Member 46233)

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, including geologic experience in similar lithotechtonic terranes (Cassiar, northern British Columbia) and with vein and disseminated type gold deposits in the U.S. (Florida Canyon, NV), South America (Nassau, Suriname), and Asia (Boroo, Mongolia), 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 “**November 2010 Summary Report on the Livengood Project, Tolovana District, Alaska”** and dated November 1, 2010 (the “Technical Report”) relating to the Livengood property, except sections 16 and 18. I have visited the Livengood property on four occasions for a total of thirty days, the most recent being from October 24 - 27, 2010.

7. Prior to being retained by ITH in 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 the Technical Report for which I am responsible 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 subject matter of the Technical Report for which I am responsible has been prepared in compliance with that instrument and form.

Dated this 1st day of November, 2010

(signed) *Tim Carew* [Sealed]
Signature of Qualified Person

Timothy J. Carew P.Geo.
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 May of 2001 with a Master of Arts degree in Management from Webster University, St. Louis, Missouri.

3. I am a recognized Qualified Professional (QP) Member with expertise in Metallurgy of the Mining and Metallurgical Society of America (MMSA Member No. 01313QP).

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. I have held several operating positions from Plant Metallurgist to Mill Manager for companies including Kennecott (Ozark Lead Company and Ridgeway Mining Company), Santa Fe Pacific Gold Corporation (Mule Canyon and Corporate Metallurgist), and Goldfields Operating Company (Ortiz and Mesquite Mines). I have also been employed as a consultant for Kilborn Engineering and Knight Piesold LLC. I have been an independent process / metallurgical consultant for the last nine (9) years for the mining industry.

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, and past relevant work experience with gold deposits requiring heap leach and mill recovery methods, including operations in Alaska, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101. I have served as a QP on numerous NI 43-101 technical reports and have served as a contributor to many other technical reports for more than ten different precious metal and non-ferrous mining companies. I have been involved as a consultant for gold mining project studies and operating gold properties that include Osisko Exploration Ltd. (Canadian Malartic project), Donlin Creek LLC (Donlin Creek project), AngloGold Ashanti (Colorado) Corp. (Cripple Creek & Victor Gold Mining Company), Coeur Alaska Inc. (Kensington Mine), Barrick Gold Corporation (Bald Mountain, Turquoise Ridge, and Ruby Hill Mines), NovaGold Resources Inc. (Rock Creek and Arctic projects), Anatolia Minerals Development Ltd. (Çöpler project), and Romarco Minerals Inc. (Haile Gold Mine).

6. I am responsible for the preparation of section 16 and the financial analysis portion of section 18 and the corresponding portions in Section 1 (Summary) of the technical report titled "**November 2010 Summary Report on the Livengood Project, Tolovana District, Alaska"** and dated November 1, 2010 (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 the Technical Report for which I am responsible 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 1.4 of NI 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1 and the subject matter of the Technical Report for which I am responsible has been prepared in compliance with that instrument and form.

Dated this 1st day of November, 2010.

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

William Pennstrom Jr. MA
Print name of Qualified Person

CERTIFICATE OF ROSCOE J. BELL, CPEng

I, Roscoe J. Bell, do hereby certify that:

1. I am Vice President and Chief Financial Officer of:

   MTB Project Management Professionals, Inc.
   8301 E Prentice Avenue
   Greenwood Village, Colorado 80111

2. I have graduated from the following Universities with degrees as follows:
   a. Clarkson University B.S. civil engineering 1964
   b. University of Phoenix M.B.A, business administration 1987

3. I am a Professional Graduate (ProfGradIMM) member in good standing of the Institute of Materials, Minerals, and Mining (IMMM) Membership Number 457295
.
4. I have worked on engineering and construction projects for 46 years, since my graduation from Clarkson 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 a professional association and past relevant work experience, which includes: International Nickel shaft sinking and development, Western Australia – Project Engineer, Yeelirrie Uranium Project Final Feasibility Study, Western Australia – Project Controls Manager, Robe River Iron Ore Project, Western Australia – Project Engineer, Estimating, Construction Manager, Hamersley Iron Port Upgrade Project, Western Australia – Project Controls Manager, BHP Teralba Project Shaft Sinking, New South Wales, Australia - Contracts Manager, Construction Manager, Sipan Gold Project, Peru, engineering, procurement, construction - Project Controls Manager, Site Contracts Manager, Rio Blanco Copper Project, Peru, Prefeasibility Study – Project Controls Manager, Magistral Copper Project, Peru, Final Feasibility Study - Project Manager, Galeno Copper Project, Peru, Scoping Study, Prefeasibility Study, and Final Feasibility Study - Project Controls Assistance, La Colosa Gold Project, Colombia, Prefeasibility Study - Project Execution Plan and Cost Reporting Set-up, La Bodega Gold Project, Colombia, Scoping Study - Project Controls Assistance, San Jose Silver Project, Argentina - Estimate and Schedule Reviews, Los Azules Copper Project, Argentina, Scoping Study - Project Controls Assistance, Questa Molybdenum Project, New Mexico - Project Estimating, Scheduling, Project Controls, Idaho Cobalt Project, Idaho, Prefeasibility and Feasibility Study - Project Management and Controls Assistance, Lisbourne Oil Production Module, North Slope of Alaska - Estimating, Pebble Copper/Gold Project, Alaska, Feasibility Study - Estimate Review and Schedule Preparation, Pebble Copper/Gold Project, Alaska, Prefeasibility Study - Execution Planning and Scheduling, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am responsible for the review, structuring, and adjustments of cost for preparation of Section 18 and corresponding portions in Section 1 (Summary) of the technical report titled "**November**

2010 Summary Report on the Livengood Project, Tolovana District, Alaska" and dated November 1, 2010 (the "Technical Report") relating to the Livengood Project. . I have visited the Livengood Project site for two days, from October 25 – 26, 2010.

7. Prior to being retained by ITH in 2010, 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 for which I am responsible 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 subject matter of the Technical Report for which I am responsible has been prepared in compliance with that instrument and form.

Dated this 1st day of November, 2010

(signed) *Roscoe J. Bell*
Signature of Qualified Person

Roscoe J. Bell
Print name of Qualified Person

CERTIFICATE OF QUINTON DE KLERK

I, Quinton de Klerk, do hereby certify that:

1. I am self employed as a Mining Engineer and Director, Mining Engineering of:

   Cube Consulting Pty. Ltd.
   Level 4, 1111 Hay Street
   West Perth, Western Australia 6005
   Australia

2. I graduated with a National Higher Diploma in Metalliferous Mining at the Technikon of the Witwatersrand, South Africa in 1993.

3. I am a corporate member of the Australasian Institute of Mining and Metallurgy (MAusIMM No 210114).

4. I have worked in the Mineral Processing Industry for over 20 years since, before, during, and after my attending the Technikon Witwatersrand.

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, and past relevant work experience, which includes modeling, planning, scheduling, development and pit construction of numerous deposits including gold deposits in many locations including Africa, Australia, Indonesia, and China, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.

6. I am responsible for the preparation of the pit optimization and production scheduling results summarized in Section 18 and corresponding portions in Section 1 (Summary) of the technical report titled "**November 2010 Summary Report on the Livengood Project, Tolovana District, Alaska"** and dated November 1, 2010 (the "Technical Report") relating to the Livengood property. I have not visited the Livengood Project site.

7. Prior to being retained by ITH in May, 2010, 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 for which I am responsible 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 1.4 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1, and the subject matter of the Technical Report for which I am responsible has been prepared in compliance with that instrument and form.

Dated this 1st day of November, 2010

(signed) *Quinton de Klerk*
Signature of Qualified Person

Quinton de Klerk
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
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
Karl Hanneman and Bergelin Family Trust	330940	LUCKY 66	14-May-1981	F009N004W33

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
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

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
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
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

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
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
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

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
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

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
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
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

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
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
Talon Gold Alaska Inc	669377	LVG 1	02/20/10	F8N4W9SESE
Talon Gold Alaska Inc	669378	LVG 2	02/20/10	F8N4W16NWNE
Talon Gold Alaska Inc	669379	LVG 3	02/20/10	F8N4W16NWSW
Talon Gold Alaska Inc	669380	LVG 4	02/20/10	F8N4W16NWSE
Talon Gold Alaska Inc	669381	LVG 5	02/20/10	F9N4W20NW
Talon Gold Alaska Inc	669382	LVG 6	02/20/10	F9N4W20NE
Talon Gold Alaska Inc	669383	LVG 7	02/20/10	F9N4W21NW
Talon Gold Alaska Inc	669384	LVG 8	02/20/10	F9N4W21NE
Talon Gold Alaska Inc	669385	LVG 9	02/20/10	F9N4W22NW
Talon Gold Alaska Inc	669386	LVG 10	02/20/10	F9N4W22NE
Talon Gold Alaska Inc	669387	LVG 11	02/20/10	F9N4W20SW
Talon Gold Alaska Inc	669388	LVG 12	02/20/10	F9N4W20SE
Talon Gold Alaska Inc	669389	LVG 13	02/20/10	F9N4W21SW
Talon Gold Alaska Inc	669390	LVG 14	02/20/10	F9N4W21SE
Talon Gold Alaska Inc	669391	LVG 15	02/20/10	F9N4W22SW
Talon Gold Alaska Inc	669392	LVG 16	02/20/10	F9N4W22SE
Talon Gold Alaska Inc	669393	LVG 17	02/20/10	F9N5W25NW
Talon Gold Alaska Inc	669394	LVG 18	02/20/10	F9N5W25NE
Talon Gold Alaska Inc	669395	LVG 19	02/20/10	F9N4W30NW
Talon Gold Alaska Inc	669396	LVG 20	02/20/10	F9N4W30NE
Talon Gold Alaska Inc	669397	LVG 21	02/20/10	F9N4W29NW
Talon Gold Alaska Inc	669398	LVG 22	02/20/10	F9N4W29NE
Talon Gold Alaska Inc	669399	LVG 23	02/20/10	F9N5W25SW
Talon Gold Alaska Inc	669400	LVG 24	02/20/10	F9N5W25SE
Talon Gold Alaska Inc	669401	LVG 25	02/20/10	F9N4W30SW
Talon Gold Alaska Inc	669402	LVG 26	02/20/10	F9N4W30SE
Talon Gold Alaska Inc	669403	LVG 27	02/20/10	F9N4W29SW
Talon Gold Alaska Inc	669404	LVG 28	02/20/10	F9N4W29SE
Talon Gold Alaska Inc	669405	LVG 29	02/20/10	F9N5W35NW
Talon Gold Alaska Inc	669406	LVG 30	02/20/10	F9N5W35NE
Talon Gold Alaska Inc	669407	LVG 31	02/20/10	F9N5W36NW
Talon Gold Alaska Inc	669408	LVG 32	02/20/10	F9N5W36NE
Talon Gold Alaska Inc	669409	LVG 33	02/20/10	F9N5W35SW
Talon Gold Alaska Inc	669410	LVG 34	02/20/10	F9N5W35SE

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
Talon Gold Alaska Inc	669411	LVG 35	02/20/10	F9N5W36SW
Talon Gold Alaska Inc	669412	LVG 36	02/20/10	F9N5W36SE
Talon Gold Alaska Inc	669413	LVG 37	02/20/10	F8N5W3NW
Talon Gold Alaska Inc	669414	LVG 38	02/20/10	F8N5W3NE
Talon Gold Alaska Inc	669415	LVG 39	02/20/10	F8N5W3SW
Talon Gold Alaska Inc	669416	LVG 40	02/20/10	F8N5W3SE
Talon Gold Alaska Inc	669417	LVG 41	02/20/10	F9N4W27NW
Talon Gold Alaska Inc	669418	LVG 42	02/20/10	F9N4W27NE
Talon Gold Alaska Inc	669419	LVG 43	02/20/10	F9N4W27SW
Talon Gold Alaska Inc	669420	LVG 44	02/20/10	F9N4W27SE
Talon Gold Alaska Inc	669421	LVG 45	02/20/10	F9N4W34NW
Talon Gold Alaska Inc	669422	LVG 46	02/20/10	F9N4W34NE
Talon Gold Alaska Inc	669423	LVG 47	02/20/10	F9N4W34SW
Talon Gold Alaska Inc	669424	LVG 48	02/20/10	F9N4W34SE
Talon Gold Alaska Inc	669425	LVG 49	02/20/10	F8N4W4NE
Talon Gold Alaska Inc	669426	LVG 50	02/20/10	F8N4W3NW
Talon Gold Alaska Inc	669427	LVG 51	02/20/10	F8N4W3NE
Talon Gold Alaska Inc	669428	LVG 52	02/20/10	F8N4W2NW
Talon Gold Alaska Inc	669429	LVG 53	02/20/10	F8N4W2NE
Talon Gold Alaska Inc	669430	LVG 54	02/20/10	F8N4W4SE
Talon Gold Alaska Inc	669431	LVG 55	02/20/10	F8N4W3SW
Talon Gold Alaska Inc	669432	LVG 56	02/20/10	F8N4W3SE
Talon Gold Alaska Inc	669433	LVG 57	02/20/10	F8N4W2SW
Talon Gold Alaska Inc	669434	LVG 58	02/20/10	F8N4W2SE
Talon Gold Alaska Inc	669435	LVG 59	02/20/10	F8N4W10NW
Talon Gold Alaska Inc	669436	LVG 60	02/20/10	F8N4W10NE
Talon Gold Alaska Inc	669437	LVG 61	02/20/10	F8N4W11NW
Talon Gold Alaska Inc	669438	LVG 62	02/20/10	F8N4W11NE
Talon Gold Alaska Inc	669439	LVG 63	02/20/10	F8N4W10SW
Talon Gold Alaska Inc	669440	LVG 64	02/20/10	F8N4W10SE
Talon Gold Alaska Inc	669441	LVG 65	02/20/10	F8N4W11SW
Talon Gold Alaska Inc	669442	LVG 66	02/20/10	F8N4W11SE
Talon Gold Alaska Inc	669443	LVG 67	02/20/10	F8N4W16NE
Talon Gold Alaska Inc	669444	LVG 68	02/20/10	F8N4W15NW
Talon Gold Alaska Inc	669445	LVG 69	02/20/10	F8N4W15NE
Talon Gold Alaska Inc	669446	LVG 70	02/20/10	F8N4W14NW
Talon Gold Alaska Inc	669447	LVG 71	02/20/10	F8N4W14NE
Talon Gold Alaska Inc	669448	LVG 72	02/20/10	F8N4W16SW
Talon Gold Alaska Inc	669449	LVG 73	02/20/10	F8N4W16SE

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
Talon Gold Alaska Inc	669450	LVG 74	02/20/10	F8N4W15SW
Talon Gold Alaska Inc	669451	LVG 75	02/20/10	F8N4W15SE
Talon Gold Alaska Inc	669452	LVG 76	02/20/10	F8N4W14SW
Talon Gold Alaska Inc	669453	LVG 77	02/20/10	F8N4W14SE
Talon Gold Alaska Inc	669454	LVG 78	02/20/10	F8N4W21NW
Talon Gold Alaska Inc	669455	LVG 79	02/20/10	F8N4W21NE
Talon Gold Alaska Inc	669456	LVG 80	02/20/10	F8N4W22NW
Talon Gold Alaska Inc	669457	LVG 81	02/20/10	F8N4W22NE
Talon Gold Alaska Inc	669458	LVG 82	02/20/10	F8N4W23NW
Talon Gold Alaska Inc	669459	LVG 83	02/20/10	F8N4W23NE
Talon Gold Alaska Inc	669460	LVG 84	02/20/10	F8N4W21SW
Talon Gold Alaska Inc	669461	LVG 85	02/20/10	F8N4W21SE
Talon Gold Alaska Inc	669462	LVG 86	02/20/10	F8N4W22SW
Talon Gold Alaska Inc	669463	LVG 87	02/20/10	F8N4W22SE
Talon Gold Alaska Inc	669464	LVG 88	02/20/10	F8N4W23SW
Talon Gold Alaska Inc	669465	LVG 89	02/20/10	F8N4W23SE
Talon Gold Alaska Inc.	700008	LVG 90	03/21/10	F9 N4 W17NW
Talon Gold Alaska Inc.	700009	LVG 91	03/21/10	F9 N4 W17NE
Talon Gold Alaska Inc.	700010	LVG 92	03/21/10	F9 N4 W16NW
Talon Gold Alaska Inc.	700011	LVG 93	03/21/10	F9 N4 W16NE
Talon Gold Alaska Inc.	700012	LVG 94	03/21/10	F9 N4 W17SW
Talon Gold Alaska Inc.	700013	LVG 95	03/21/10	F9 N4 W17SE
Talon Gold Alaska Inc.	700014	LVG 96	03/21/10	F9 N4 W16SW
Talon Gold Alaska Inc.	700015	LVG 97	03/21/10	F9 N4 W16SE
Talon Gold Alaska Inc.	700016	LVG 98	03/21/10	F8 N5 W9NW
Talon Gold Alaska Inc.	700017	LVG 99	03/21/10	F8 N5 W9NE
Talon Gold Alaska Inc.	700018	LVG 100	03/21/10	F8 N5 W9SW
Talon Gold Alaska Inc.	700019	LVG 101	03/21/10	F8 N5 W9SE
Talon Gold Alaska Inc.	700020	LVG 102	03/21/10	F8 N4 W31NW
Talon Gold Alaska Inc.	700021	LVG 103	03/21/10	F8 N4 W31NE
Talon Gold Alaska Inc.	700022	LVG 104	03/21/10	F8 N4 W32NW
Talon Gold Alaska Inc.	700023	LVG 105	03/21/10	F8 N4 W32NE
Talon Gold Alaska Inc.	700024	LVG 106	03/21/10	F8 N4 W31SW
Talon Gold Alaska Inc.	700025	LVG 107	03/21/10	F8 N4 W31SE
Talon Gold Alaska Inc.	700026	LVG 108	03/21/10	F8 N4 W32SW
Talon Gold Alaska Inc.	700027	LVG 109	03/21/10	F8 N4 W32SE
Talon Gold Alaska Inc.	700028	LVG 110	03/21/10	F7 N4 W6NW
Talon Gold Alaska Inc.	700029	LVG 111	03/21/10	F7 N4 W6NE
Talon Gold Alaska Inc.	700030	LVG 112	03/21/10	F7 N4 W5NW

Owner	File Number	Tenure Name	Date Acquired	MTRS Location
Talon Gold Alaska Inc.	700031	LVG 113	03/21/10	F7 N4 W5NE
Talon Gold Alaska Inc.	700032	LVG 114	03/21/10	F7 N4 W4NW
Talon Gold Alaska Inc.	700033	LVG 115	03/21/10	F7 N4 W4NE

* - 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	508.30	213.40
BAF-2	430073.00	7266149.00	512.70	152.40
BAF-3	429760.00	7266096.00	501.30	150.90
BAF-4	430073.00	7265881.00	470.20	216.40
BAF-5	430078.00	7265765.00	444.40	189.90
BAF-6	429745.00	7265979.00	511.90	134.10
BAF-7	430056.00	7266034.00	508.30	304.80
BAF-8	430342.00	7266042.00	510.10	152.40
L-1	429726.00	7265450.00	508.70	31.00
L-2	429350.00	7265457.00	496.80	73.00
L-3	429050.00	7265715.00	464.50	46.00
L-4	429045.00	7265688.00	464.40	20.00
L-5	428910.00	7265675.00	447.00	70.00
L-6	428805.00	7265640.00	432.10	70.00
LC-TR-01	428883.00	7266132.00	356.70	91.44
LC-TR-02	428859.00	7266041.00	340.70	68.58
MK-04-01	428734.41	7265596.00	421.50	109.73
MK-04-02	428492.09	7265738.00	361.60	305.71
MK-04-03	428674.69	7265520.50	412.20	208.79
MK-04-04	428547.69	7265813.50	354.40	137.77
MK-04-TP1	429594.00	7265670.00	505.20	2.00
MK-04-TP2	429583.00	7265653.00	506.00	2.00
MK-04-TR1	429541.09	7265537.00	514.40	34.00
MK-04-TR2E	429598.00	7265763.00	503.20	85.00
MK-04-TR2S	429598.00	7265763.00	503.20	20.00
MK-04-TR2W	429597.09	7265763.50	503.10	85.00
MK-04-TR3	429603.00	7265704.00	504.40	33.40
MK-04-TR5	429570.00	7265621.00	508.30	15.00
MK-06-05	429099.00	7266101.00	397.90	305.10
MK-06-06	429299.00	7266298.00	395.00	205.44
MK-06-07	428772.31	7265845.00	412.80	276.45
MK-06-08	428915.31	7265897.00	408.70	288.34
MK-06-09	427614.00	7264251.00	213.40	124.66
MK-06-10	427533.00	7264335.00	210.50	10.36
MK-06-11	427691.00	7264430.00	230.00	17.07
MK-07-12	428915.31	7265897.00	408.70	282.85
MK-07-13	428773.31	7265847.50	412.80	351.13

MK-07-14	428774.81	7265846.00	412.80	44.81
MK-07-15	428774.81	7265849.00	412.80	281.64
MK-07-16	430220.00	7265985.00	517.60	332.84
MK-07-17	428773.41	7265621.50	427.70	421.84
MK-07-18	428853.59	7265780.00	431.80	301.14
MK-07-19	429002.59	7265704.00	458.40	436.17
MK-07-20	428851.69	7265720.00	435.30	244.30
MK-07-21	428925.81	7265760.50	440.20	309.98
MK-07-22	428703.31	7265764.00	408.50	382.83
MK-07-23	429075.81	7265779.50	458.80	290.17
MK-07-24	429529.81	7265631.00	508.90	372.16
MK-07-25	428399.59	7265253.00	368.20	330.40
MK-07-26	429900.00	7265470.00	448.30	28.35
MK-08-27	429592.59	7265927.50	499.89	201.78
MK-08-28	429518.31	7266005.50	485.94	229.21
MK-08-29	429896.00	7265778.50	470.12	266.70
MK-08-30	428891.91	7265738.00	438.73	345.19
MK-08-31	429142.41	7265606.50	479.10	376.43
MK-08-32	429186.50	7265431.00	474.07	343.81
MK-08-33	429066.31	7265091.00	427.53	276.76
MK-08-TR01	428869.81	7266061.50	342.36	21.34
MK-08-TR02	428834.59	7266031.00	338.84	28.04
MK-08-TR03	428834.59	7266031.00	338.84	4.11
MK-08-TR04	428869.81	7266061.50	342.36	26.06
MK-09-34	428771.91	7265545.00	427.53	296.27
MK-09-35	428851.09	7265491.00	437.15	276.45
MK-09-36	428782.50	7265215.50	409.49	697.93
MK-09-37	429109.09	7265406.00	463.73	527.30
MK-09-38	429251.31	7265388.00	477.33	215.80
MK-09-39	429524.81	7265999.00	487.82	309.37
MK-09-40	429254.09	7265386.00	477.68	584.61
MK-09-41	430048.59	7265922.00	480.85	407.82
MK-09-42	429604.09	7265703.00	503.41	341.38
MK-09-43	429562.31	7265813.00	500.02	428.24
MK-09-44	428946.31	7265103.50	417.87	313.33
MK-09-45	429451.69	7265094.50	441.06	174.80
MK-1	428945.00	7265820.00	427.40	76.00
MK-10-46	429519.31	7265865.50	496.13	350.67
MK-10-47	428962.50	7265499.00	451.16	297.79
MK-10-48	428930.31	7265240.00	430.04	441.05
MK-10-49	428778.19	7265393.00	422.27	305.71

MK-10-50	428775.81	7264872.50	379.07	263.35
MK-10-51	428702.00	7265024.50	383.39	339.55
MK-2	428825.00	7265850.00	422.30	77.00
MK-3	429500.00	7266190.00	450.40	28.04
MK-4	429493.00	7266117.00	466.00	15.24
MK-4B	429493.00	7266117.00	466.00	106.68
MK-5	428660.00	7265925.00	357.20	0.00
MK-6	428680.00	7265940.00	357.70	0.00
MK-RC-0001	428996.00	7265778.00	449.00	321.56
MK-RC-0002	429001.81	7265854.50	426.10	335.28
MK-RC-0003	428703.19	7265998.50	335.90	222.50
MK-RC-0004	428612.00	7265921.00	343.50	274.32
MK-RC-0005	428561.81	7265841.50	350.00	269.75
MK-RC-0006	429045.69	7265695.50	460.70	353.57
MK-RC-0007	428846.00	7265843.00	423.59	286.51
MK-RC-0008	428925.00	7265691.50	445.87	213.36
MK-RC-0009	428997.91	7265632.00	456.48	246.89
MK-RC-0010	428547.69	7265471.00	393.24	240.79
MK-RC-0011	428925.69	7265626.50	447.99	225.55
MK-RC-0012	428997.00	7265544.50	459.54	307.85
MK-RC-0013	428624.19	7265480.00	403.22	225.55
MK-RC-0014	428176.91	7265590.50	357.31	217.93
MK-RC-0015	428323.09	7265696.50	349.18	195.07
MK-RC-0016	428319.50	7265542.50	367.72	134.11
MK-RC-0017	428779.09	7265774.00	423.18	297.18
MK-RC-0018	428710.91	7265834.00	396.94	252.98
MK-RC-0019	428550.00	7265925.00	329.20	54.86
MK-RC-0020	428549.69	7265910.00	331.52	213.36
MK-RC-0021	428470.00	7265852.00	330.47	213.36
MK-RC-0022	428847.91	7265920.50	399.81	280.42
MK-RC-0023	428849.31	7265622.50	437.70	288.04
MK-RC-0024	428697.81	7265630.00	413.90	207.26
MK-RC-0025	428920.91	7265909.00	404.48	213.36
MK-RC-0026	428622.91	7265760.00	385.76	167.64
MK-RC-0027	428559.09	7265704.00	381.63	131.06
MK-RC-0028	428844.50	7266105.50	339.35	92.96
MK-RC-0029	429057.91	7265856.50	432.49	256.03
MK-RC-0030	428777.19	7265548.00	425.83	243.84
MK-RC-0031	428926.50	7265548.00	447.21	303.28
MK-RC-0032	428554.91	7265783.00	363.55	91.44
MK-RC-0033	428849.41	7265566.50	437.15	335.28

MK-RC-0034	429073.81	7265553.50	467.93	365.76
MK-RC-0035	429071.91	7265468.00	467.93	330.71
MK-RC-0036	429001.59	7265463.50	453.22	259.08
MK-RC-0037	429149.41	7265558.50	483.55	295.66
MK-RC-0038	428784.09	7265918.50	392.47	234.70
MK-RC-0039	428999.09	7265410.00	450.69	277.37
MK-RC-0040	428927.41	7265860.50	418.95	335.28
MK-RC-0041	428850.69	7265504.00	437.48	262.13
MK-RC-0042	428778.59	7265473.00	425.93	274.32
MK-RC-0043	428940.31	7265472.50	446.35	265.18
MK-RC-0044	428698.09	7265487.50	417.56	237.74
MK-RC-0045	428922.00	7265395.50	441.12	316.99
MK-RC-0046	429084.00	7265622.50	470.46	323.09
MK-RC-0047	429152.59	7265477.50	475.36	326.75
MK-RC-0048	429144.00	7265399.00	466.90	350.52
MK-RC-0049	428697.69	7265404.50	416.86	274.32
MK-RC-0050	429225.09	7265481.50	488.52	350.82
MK-RC-0051	428699.81	7265549.50	416.63	239.27
MK-RC-0052	428625.50	7265848.00	366.64	249.94
MK-RC-0053	428544.00	7265550.00	393.22	204.22
MK-RC-0054	429297.19	7265483.50	493.39	341.38
MK-RC-0055	428706.41	7265927.00	368.89	262.13
MK-RC-0056	428477.41	7265560.00	384.55	195.07
MK-RC-0057	429374.31	7265487.00	504.80	304.80
MK-RC-0058	428700.09	7266242.00	334.28	213.36
MK-RC-0059	429450.19	7265478.50	511.60	262.13
MK-RC-0060	429077.09	7265328.50	453.48	336.80
MK-RC-0061	429225.81	7265326.50	468.33	302.06
MK-RC-0062	429150.19	7265323.50	460.55	312.42
MK-RC-0063	429299.59	7265329.00	474.40	359.66
MK-RC-0064	429072.41	7265252.50	445.29	363.32
MK-RC-0065	429302.81	7265425.00	484.82	345.95
MK-RC-0066	429156.31	7265243.00	452.11	304.80
MK-RC-0067	429155.31	7265175.00	448.17	349.00
MK-RC-0068	429227.31	7265403.50	476.18	396.24
MK-RC-0069	429147.50	7265098.50	434.71	256.03
MK-RC-0070	429452.09	7265549.00	509.89	377.95
MK-RC-0071	428928.31	7265326.00	435.54	301.75
MK-RC-0072	428997.91	7265324.00	444.93	262.13
MK-RC-0073	429521.59	7265549.50	513.20	335.28
MK-RC-0074	428474.00	7265632.50	377.26	158.50

MK-RC-0075	428477.19	7265482.00	386.50	243.84
MK-RC-0076	429151.09	7265033.50	425.54	284.99
MK-RC-0077	428475.91	7265930.00	312.15	152.40
MK-RC-0078	429225.91	7265026.50	428.22	298.70
MK-RC-0079	428399.41	7265859.00	319.96	161.54
MK-RC-0080	428626.69	7265396.50	402.58	265.18
MK-RC-0081	428841.59	7265250.00	419.89	243.84
MK-RC-0082	429073.59	7265037.50	421.65	316.99
MK-RC-0083	428911.09	7265169.50	420.59	300.23
MK-RC-0084	429224.50	7265250.50	458.21	374.90
MK-RC-0085	429599.09	7265554.50	510.81	326.14
MK-RC-0086	429377.91	7265391.00	491.43	36.58
MK-RC-0087	429148.50	7264950.00	417.20	254.51
MK-RC-0088	429003.41	7265008.50	413.46	115.82
MK-RC-0089	429003.41	7265008.50	413.46	374.90
MK-RC-0090	429070.09	7264947.00	413.33	201.17
MK-RC-0091	429007.09	7264948.00	407.44	283.46
MK-RC-0092	429377.91	7265391.00	491.43	344.42
MK-RC-0093	429226.09	7265104.00	439.05	323.10
MK-RC-0094	429747.50	7265480.50	497.82	329.18
MK-RC-0095	429595.81	7266007.00	499.95	268.22
MK-RC-0096	428780.91	7265218.00	410.00	262.13
MK-RC-0097	429897.41	7265464.50	447.73	237.74
MK-RC-0098	428925.00	7265112.00	415.29	219.46
MK-RC-0099	429296.69	7264947.00	419.03	268.22
MK-RC-0100	429214.00	7264951.50	418.33	274.32
MK-RC-0101	429294.00	7265028.00	429.73	295.70
MK-RC-0102	429296.31	7265176.00	453.02	274.32
MK-RC-0103	429229.09	7265170.50	449.21	307.85
MK-RC-0103a	429225.00	7265175.00	449.90	6.10
MK-RC-0104	429159.81	7264696.00	386.59	128.02
MK-RC-0105	429138.41	7264694.50	387.76	190.50
MK-RC-0106	429071.19	7265245.00	445.85	335.28
MK-RC-0107	429296.00	7264725.00	378.26	224.03
MK-RC-0108	429296.69	7265103.00	442.38	271.27
MK-RC-0109	428934.31	7265034.50	409.73	284.99
MK-RC-0110	428996.00	7265174.50	430.45	355.09
MK-RC-0111	429446.91	7265638.00	504.18	303.58
MK-RC-0112	429376.09	7265625.50	500.43	356.62
MK-RC-0113	429296.69	7265617.50	493.55	334.37
MK-RC-0114	429229.31	7265624.50	486.66	307.85

MK-RC-0115	428694.09	7264869.50	369.12	265.18
MK-RC-0116	428636.09	7264960.00	369.94	295.66
MK-RC-0117	428775.00	7265085.50	397.62	182.88
MK-RC-0118	428761.00	7264784.00	370.36	289.56
MK-RC-0119	428774.31	7265081.50	397.66	225.55
MK-RC-0120	428610.50	7264794.50	353.25	313.94
MK-RC-0121	428693.59	7265241.50	401.21	231.65
MK-RC-0122	428773.41	7264966.50	385.03	295.66
MK-RC-0123	428694.81	7265247.50	401.57	332.84
MK-RC-0124	428627.50	7265097.50	380.21	301.75
MK-RC-0125	428764.91	7265308.50	414.57	306.93
MK-RC-0126	428851.31	7265319.50	425.80	263.65
MK-RC-0127	428617.19	7265252.50	391.91	307.90
MK-RC-0128	429302.19	7265768.00	476.88	320.04
MK-RC-0129	428846.59	7265013.00	398.61	262.13
MK-RC-0130	429150.69	7265775.50	462.44	286.51
MK-RC-0131	428848.69	7264870.50	386.75	260.60
MK-RC-0132	428928.81	7264939.50	401.00	220.98
MK-RC-0133	428845.81	7265095.50	407.35	327.70
MK-RC-0134	428627.31	7265628.50	404.32	182.88
MK-RC-0135	429376.69	7265704.50	491.98	301.80
MK-RC-0136	428854.09	7265401.50	432.04	297.20
MK-RC-0137	429466.41	7265926.50	482.68	280.42
MK-RC-0138	428992.31	7265089.00	421.79	269.75
MK-RC-0139	429368.09	7265989.00	456.91	289.56
MK-RC-0140	428700.31	7265164.50	396.11	320.04
MK-RC-0141	429304.19	7265999.50	443.23	198.12
MK-RC-0142	428686.41	7265104.00	388.41	280.42
MK-RC-0143	430273.19	7266146.00	542.38	301.75
MK-RC-0144	429677.19	7265407.00	513.99	310.90
MK-RC-0145	430421.00	7266012.00	477.81	311.51
MK-RC-0146	429818.91	7265396.50	473.50	256.03
MK-RC-0147	429245.41	7264877.00	408.21	350.52
MK-RC-0148	430417.41	7266142.50	504.45	307.85
MK-RC-0149	429826.00	7265555.00	464.11	170.69
MK-RC-0150	429380.00	7264892.00	412.05	193.55
MK-RC-0151	429673.31	7265549.00	504.29	266.70
MK-RC-0152	430124.41	7265924.50	486.84	306.93
MK-RC-0153	429372.81	7265019.00	429.49	262.13
MK-RC-0154	429373.19	7265177.00	454.74	344.42
MK-RC-0155	429984.41	7265930.00	483.41	300.23

MK-RC-0156	429670.19	7265842.50	503.93	316.99
MK-RC-0157	429374.19	7265251.00	466.40	301.75
MK-RC-0158	429672.00	7265916.00	507.81	324.61
MK-RC-0159	429825.09	7265848.00	491.69	272.80
MK-RC-0160	429673.91	7266069.50	503.64	316.99
MK-RC-0161	429458.41	7264796.00	389.32	242.93
MK-RC-0162	429524.41	7266077.50	480.71	263.65
MK-RC-0163	429376.41	7264799.50	389.75	325.22
MK-RC-0164	429302.00	7264795.50	391.90	334.67
MK-RC-0165	429746.19	7265846.50	500.61	249.94
MK-RC-0166	429740.31	7265918.00	509.18	240.79
MK-RC-0167	429676.31	7265703.00	497.76	286.51
MK-RC-0168	429356.50	7264949.00	419.58	312.42
MK-RC-0169	430124.59	7266078.50	531.45	339.85
MK-RC-0170	429526.00	7265861.50	494.19	301.75
MK-RC-0171	429454.41	7264940.00	413.51	276.80
MK-RC-0172	429602.59	7264877.50	391.76	298.70
MK-RC-0173	429520.31	7264950.50	412.08	242.32
MK-RC-0174	429602.41	7265860.50	502.14	321.60
MK-RC-0175	428413.09	7265552.00	377.14	198.12
MK-RC-0176	429447.41	7265018.50	430.48	248.41
MK-RC-0177	429969.41	7266055.00	502.98	278.90
MK-RC-0178	429302.69	7264870.50	407.19	316.99
MK-RC-0179	428545.09	7265409.00	393.24	73.15
MK-RC-0180	429670.91	7265996.50	507.25	347.47
MK-RC-0181	429372.50	7265122.50	446.07	262.13
MK-RC-0182	428817.41	7265677.50	432.97	274.32
MK-RC-0183	429301.91	7265247.00	463.90	332.23
MK-RC-0184	428545.09	7265409.00	393.24	268.22
MK-RC-0185	429599.00	7266016.00	499.65	289.56
MK-RC-0186	429176.41	7265350.50	465.01	365.76
MK-RC-0187	429971.59	7265854.50	470.45	317.60
MK-RC-0188	429602.91	7266075.50	496.19	268.22
MK-RC-0189	429451.69	7265098.50	440.82	233.48
MK-RC-0190	429889.91	7265852.00	483.59	286.51
MK-RC-0191	430205.09	7265555.00	391.17	170.69
MK-RC-0192	429522.69	7265104.50	435.33	300.53
MK-RC-0193	430351.31	7265706.00	413.67	368.81
MK-RC-0194	429524.09	7265025.50	425.60	251.46
MK-RC-0195	430349.69	7266235.00	529.90	319.74
MK-RC-0196	430493.31	7265843.50	427.60	359.66

MK-RC-0197	428480.50	7265397.50	385.59	313.94
MK-RC-0198	430637.31	7265918.50	451.05	335.28
MK-RC-0199	430343.50	7266153.50	523.27	341.38
MK-RC-0200	428400.59	7265467.50	377.58	277.37
MK-RC-0201	429533.41	7265188.50	450.49	298.70
MK-RC-0202	430278.31	7266088.00	539.29	365.76
MK-RC-0203	430501.19	7265922.00	443.48	365.76
MK-RC-0204	429597.91	7265251.50	455.69	213.36
MK-RC-0205	429453.00	7265178.00	452.58	252.98
MK-RC-0206	429829.00	7265995.50	508.01	402.34
MK-RC-0207	429974.09	7265999.00	500.76	399.30
MK-RC-0208	429525.19	7265255.00	465.99	262.13
MK-RC-0209	429900.31	7265925.50	492.04	408.43
MK-RC-0210	429448.81	7265248.00	467.86	278.90
MK-RC-0211	429754.41	7266003.00	510.63	402.34
MK-RC-0212	429598.31	7265192.50	441.60	214.88
MK-RC-0213	429901.19	7266006.00	504.06	411.48
MK-RC-0214	429599.69	7265094.50	424.97	201.17
MK-RC-0215	429756.19	7266074.50	505.81	396.24
MK-RC-0216	429680.09	7265175.50	427.24	216.41
MK-RC-0217	429818.69	7265922.50	501.72	396.24
MK-RC-0218	429600.50	7265023.50	413.88	224.03
MK-RC-0219	429598.09	7264949.00	403.46	356.62
MK-RC-0220	429602.69	7265774.00	501.13	396.24
MK-RC-0221	430029.09	7265465.00	401.76	353.60
MK-RC-0222	429530.81	7265926.50	492.43	399.29
MK-RC-0223	429678.50	7265773.50	499.54	341.38
MK-RC-0224	429467.19	7265932.50	481.72	376.43
MK-RC-0225	429968.09	7265308.50	397.59	347.47
MK-RC-0226	429747.09	7265700.50	485.81	341.38
MK-RC-0227	429898.69	7265163.50	386.41	256.03
MK-RC-0228	429527.69	7266161.00	463.33	254.51
MK-RC-0229	429605.81	7265704.50	503.25	237.74
MK-RC-0230	429743.31	7265023.50	391.32	316.99
MK-RC-0231	429457.41	7266073.50	466.34	335.28
MK-RC-0232	429000.59	7264436.50	368.25	243.84
MK-RC-0233	429454.41	7266001.00	473.30	350.52
MK-RC-0234	429606.50	7265701.00	503.74	423.70
MK-RC-0235	428953.91	7264668.50	383.42	323.10
MK-RC-0236	429600.69	7265621.50	505.76	396.24
MK-RC-0237	429519.41	7265999.50	487.49	396.24

MK-RC-0238	428998.09	7264724.00	390.06	271.30
MK-RC-0239	429673.19	7265628.50	497.41	426.72
MK-RC-0240	429598.00	7266145.00	480.22	320.04
MK-RC-0241	428776.09	7264507.00	363.65	292.61
MK-RC-0242	429750.31	7265775.50	492.54	426.72
MK-RC-0243	430040.59	7266070.50	506.17	310.90
MK-RC-0244	429673.50	7266147.00	483.75	396.24
MK-RC-0245	431101.91	7266322.00	539.84	409.96
MK-RC-0246	429753.69	7265626.00	480.95	91.44
MK-RC-0247	429745.09	7266148.50	484.07	271.27
MK-RC-0248	430949.31	7266219.00	548.09	371.86
MK-RC-0249	429753.69	7265624.50	480.95	402.34
MK-RC-0250	429824.19	7265702.00	471.17	353.60
MK-RC-0251	429823.81	7266149.00	485.27	219.50
MK-RC-0252	430832.69	7266113.00	533.22	361.19
MK-RC-0253	429824.31	7266147.50	485.49	353.57
MK-RC-0254	429822.09	7265783.00	485.01	429.77
MK-RC-0255	430200.31	7266241.50	524.14	286.51
MK-RC-0256	429525.59	7265702.50	503.85	426.72
MK-RC-0257	429527.00	7265767.50	499.12	274.32
MK-RC-0258	429824.09	7266074.00	502.81	374.90
MK-RC-0259	429742.91	7265548.50	493.03	315.47
MK-RC-0260	430126.00	7266147.50	525.95	396.24
MK-RC-0261	430283.41	7266008.00	519.95	390.14
MK-RC-0262	429751.00	7265403.00	489.92	377.95
MK-RC-0263	430200.31	7266141.50	544.83	377.95
MK-RC-0264	429518.50	7265490.00	519.77	423.67
MK-RC-0265	430418.91	7266070.50	491.30	396.24
MK-RC-0266	430339.69	7266001.50	500.35	365.76
MK-RC-0267	429526.69	7265759.50	499.12	365.76
MK-RC-0268	430510.69	7266079.00	478.62	365.80
MK-RC-0269	429450.31	7265873.50	484.07	371.86
MK-RC-0270	429597.31	7265932.00	501.62	152.40
MK-RC-0271	430492.19	7266001.50	461.52	280.42
MK-RC-0272	430345.31	7265919.50	471.50	387.10
MK-RC-0273	430054.91	7266004.50	505.07	338.33
MK-RC-0274	430421.81	7265923.00	455.73	379.48
MK-RC-0275	430353.09	7266078.50	512.38	320.04
MK-RC-0276	428623.09	7265320.50	397.08	310.90
MK-RC-0277	430957.41	7265913.00	496.01	304.80
MK-RC-0278	428775.59	7265175.00	406.60	243.84

MK-RC-0279	428095.31	7265434.00	334.72	332.23
MK-RC-0280	429671.31	7265349.50	493.26	359.66
MK-RC-0281	427898.19	7265539.00	291.19	286.51
MK-RC-0282	431106.50	7265779.50	479.68	347.47
MK-RC-0283	428030.50	7265567.50	317.61	214.88
MK-RC-0284	429746.59	7265331.50	471.40	396.24
MK-RC-0285	430049.69	7266159.50	506.86	353.57
MK-RC-0286	429826.81	7265465.50	469.68	326.14
MK-RC-0287	428403.31	7265395.50	375.88	274.32
MK-RC-0288	430273.31	7266233.50	533.41	396.24
MK-RC-0289	428846.81	7266470.50	358.08	355.09
MK-RC-0290	430052.19	7265866.00	468.48	350.52
MK-RC-0291	429755.81	7264873.00	370.08	234.70
MK-RC-0292	429379.50	7266155.00	431.86	187.45
MK-RC-0293	429603.81	7265476.50	533.63	396.24
MK-RC-0294	429369.69	7265988.50	457.47	318.52
MK-RC-0295	429825.00	7265325.00	451.00	99.06
MK-RC-0296	429821.09	7265328.00	451.80	274.32
MK-RC-0297	429377.19	7265928.00	466.77	314.86
MK-RC-0298	429674.19	7265473.00	523.46	396.24
MK-RC-0299	429737.41	7265261.50	443.11	265.18
MK-RC-0300	429454.69	7266156.00	447.55	152.40
MK-RC-0301	429682.09	7265237.50	443.08	222.50
MK-RC-0302	429377.91	7265323.00	479.02	307.85
MK-RC-0303	429378.59	7266083.50	444.10	256.03
MK-RC-0304	429374.19	7265554.00	503.42	368.81
MK-RC-0305	429296.91	7265555.00	497.09	408.43
MK-RC-0306	429151.59	7265705.50	472.79	416.05
MK-RC-0307	429447.00	7265702.00	501.00	298.70
MK-RC-0308	429228.09	7265549.50	490.55	441.96
MK-RC-0309	429224.59	7265704.50	479.63	365.76
MK-RC-0310	429373.09	7265859.50	473.63	402.34
MK-RC-0311	429295.81	7265697.50	485.47	341.38
MK-RC-0312	429447.41	7265780.50	491.82	457.20
MK-RC-0313	429376.50	7265777.50	484.01	432.82
MK-RC-0314	429226.91	7265854.50	455.52	274.32
MK-RC-0315	429076.69	7265406.00	461.29	457.20
MK-RC-0316	429148.91	7265841.00	450.05	340.46
MK-RC-0317	429009.31	7265932.00	403.23	280.42
MK-RC-0318	429218.00	7265789.00	466.14	109.73
MK-RC-0319	428776.69	7265708.00	426.44	231.65

MK-RC-0320	429305.09	7266001.50	443.16	288.04
MK-RC-0321	429220.00	7265787.00	466.14	457.20
MK-RC-0322	429296.69	7265939.50	451.04	246.89
MK-RC-0323	428702.00	7265710.00	414.01	188.98
MK-RC-0324	428629.59	7265552.00	406.34	231.65
MK-RC-0325	428623.59	7265693.50	399.19	158.50
MK-RC-0326	429080.00	7265930.00	418.21	140.21
MK-RC-0327	428701.81	7265333.00	408.01	356.62
MK-RC-0328	429300.91	7265852.00	465.23	306.32
MK-RC-0329	428849.19	7265175.00	414.68	368.81
MK-RC-0330	429013.41	7265239.00	439.75	365.76
MK-RC-0331	429285.91	7266075.50	428.34	314.25
MK-RC-0332	428552.31	7264733.00	342.72	384.05
MK-RC-0333	428550.81	7265029.00	367.01	256.03
MK-RC-0334	428549.91	7264870.00	351.65	327.66
MK-RC-0335	428550.69	7265175.50	378.12	243.84
MK-RC-0336	428701.00	7264732.50	362.85	91.44
MK-RC-0337	428475.00	7264953.00	350.84	332.23
MK-RC-0338	428478.09	7264804.00	338.43	353.57
MK-RC-0339	428701.00	7264734.50	362.85	324.61
MK-RC-0340	429224.59	7266071.50	419.78	201.17
MK-RC-0341	428399.81	7265028.00	347.40	323.09
MK-RC-0342	428700.59	7264951.50	375.90	262.43
MK-RC-0343	429225.81	7265930.50	442.29	231.65
MK-RC-0344	428479.19	7265101.00	363.13	323.09
MK-RC-0345	428622.50	7264882.50	360.83	356.01
MK-RC-0346	428848.91	7264727.50	378.22	265.18
MK-RC-0347	428626.69	7265029.00	375.89	251.46
MK-RC-0348	428626.59	7265027.00	374.97	338.33
MK-RC-0349	428626.19	7265178.00	386.90	365.76
MK-RC-0350	429153.59	7265924.00	432.57	259.08
MK-RC-0351	429231.81	7266009.50	430.07	274.32
MK-RC-0352	428098.69	7265394.50	335.47	390.14
MK-RC-0353	428175.50	7265405.00	348.03	377.95
MN-1	428864.00	7266045.00	341.00	106.68
MN-2	428864.00	7266045.00	341.00	106.68
MN-3	428745.00	7266065.00	313.60	106.70
TL-10	428183.00	7265586.00	358.50	79.00
TL-11	429528.00	7266520.00	352.60	105.00
TL-12	429223.00	7266654.00	297.80	200.41
TL-13	429054.00	7266654.00	318.40	150.27

TL-14	427780.00	7265504.00	272.80	124.00
TL-6	433265.00	7269380.00	277.00	43.89
TL-7	428443.00	7266477.00	303.20	101.19
TL-8	428443.00	7266477.00	303.20	192.93
TL-9	428443.00	7266477.00	303.20	105.00