GEOCHEMICAL & TECHNICAL
ASSESSMENT REPORT

On the

SKY 10, SKY 11 and REG 2
MINERAL CLAIMS

Belonging to

SKYLINE GOLD CORPORATION

Located in the

LIARD MINING DISTRICT
NTS 104B 11E
At

56° 38' 00" N. Latitude
131° 04' 00" W. Longitude

RECLAMATION PERMIT
M-178

By

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March 29, 2003

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1.0 INTRODUCTION

Skyline Gold Corporation owns a 100% interest in the Iskut River property that comprises 13 Crown Granted claims and 186 Modified Grid claim units. A one percent Net Smelter Royalty in favour of Skyline exists on a further 20 Modified Grid claim units. The property is located in northwestern British Columbia, six kilometres south of Iskut River, between Craig River and Bronson Creek. The property is 102 kilometres northwest of Stewart, B.C. and 80 kilometres east-northeast of Wrangell, Alaska at approximately 56° 38' 00" N. Latitude, 131° 04' 00" W. Longitude.

This report describes work done on the SKY 10, SKY 11 and REG 2 claims during the period August 20, to August 26, 2003. Work comprised:

- a technical inspection of the tailings impoundment dykes
- a program of tailings sampling (26 samples)
- a geochemical survey of rocks used to surface the airstrip (14 samples)
- a geochemical survey of rocks used to construct the magazine road (11 samples)
- a geochemical survey of the 12 Level and 11 Level waste rock dumps (2 samples)
- a geochemical survey of underground waters (6 samples)
- a geochemical survey of surface waters. (8 samples)

The work was performed in accordance with Reclamation Permit M-178.

2.0 MINERAL TENURE

The claims are located in the Liard Mining District of British Columbia. The following table lists the mineral tenures comprising the Iskut property of Skyline Gold Corporation.

Tenure Name	Tenure Type	Tenure Number	Number of Units	Ownership %
RED BLUFF	Crown Grant	2857	1	100
HOMESTAKE	Crown Grant	2858	1	100
RED BIRD	Crown Grant	2859	1	100
MERMAID	Crown Grant	2860	1	100
EL ORO	Crown Grant	2862	1	100
DISCOVERY	Crown Grant	2863	1	100
GOLDEN PHEASANT	Crown Grant	2864	1	100
BROWN BEAR	Crown Grant	2865	1	100
ISKOOT	Crown Grant	2866	1	100
SILVER DOLLAR	Crown Grant	2867	1	100
MARGURITTE	Crown Grant	2868	1	100
BLU GROUSE	Crown Grant	2869	1	100
COPPER QUEEN	Crown Grant	2870	1	100

Tenure Name	Tenure Type	Tenure Number	Number of Units	Ownership %
REG 1	Mineral Claim	221935	20	100
REG 2	Mineral Claim	221936	20	100
REG 8	Mineral Claim	222130	20	100
WARATAH 7	Mineral Claim	222212	20	1% NSR to Skyline Gold Corporation
SKY 2	Mineral Claim	222218	5	100
KATY 1	Mineral Claim	357913	1	100
KATY 2	Mineral Claim	357914	1	100
HIGH 1	Mineral Claim	357915	1	3.5% NSR to Homestake Canada Inc.; may be reduced to 3% upon payment of $500,000
HIGH 2	Mineral Claim	357916	1	3.5% NSR to Homestake Canada Inc.; may be reduced to 3% upon payment of $500,000
HIGH 3	Mineral Claim	357917	1	3.5% NSR to Homestake Canada Inc.; may be reduced to 3% upon payment of $500,000
SKY 10	Mineral Claim	385146	20	100
SKY 11	Mineral Claim	385147	20	100
SKY 12	Mineral Claim	385148	16	100
REG 9	Mineral Claim	390061	20	100
REG 5	Mineral Claim	390062	20	100
BURNIE	Mineral Claim	390063	20	100

The work was performed on the SKY 10, SKY 11 and REG 2 claims. The claims, comprising 60 units, are listed as Mineral Tenure Numbers 385146, 385147 and 221936 respectively, on British Columbia mineral tenure map M104B065. The claims were in good standing until December 31, 2003 (SKY 10 and SKY 11) and December 31, 2005 (REG 2). If the assessment work described in this report is accepted for assessment credit, the claims will be in good standing until December 31, 2005. The claims are subject to Reclamation Permit Number M-178.

3.0 LOCATION AND ACCESS

Access to the property is presently by air to an airstrip on the property, or alternatively to an airstrip on the neighbouring Snip property (Barrick Gold Corporation). The two airstrips are joined by a 10 kilometre long road. Both airstrips are capable of landing Hercules aircraft.

Air support is available from Smithers, B.C., Terrace, B.C. or Wrangell, Alaska. Smithers and Terrace are approximately 330 kilometres south of the property. Larger airlifts can be arranged from the Bob Quin airstrip, located 65 km northeast of the Iskut property on the Cassiar Stewart Highway (#37). No services are available at Bob Quin.

Additionally, there is a 30 kilometre long bulldozer trail easterly to the Eskay Creek road that can be utilized at certain times of the year with proper permitting.

4.0 PROPERTY HISTORY

4.1 Exploration History

The Iskut property history is summarized below.

In 1907, a prospecting party from Wrangell, Alaska recorded claims on Bronson Creek. These claims were later Crown Granted and remain in existence today. In the period 1911 to 1920 the Iskut Mining Company reported drifting, trenching and stripping a number of gold bearing veins on the Red Bluff and Iskut claims on the northeastern portion of the property. From 1954 to 1960 Hudson Bay Mining and Smelting Co. Ltd. completed exploration drilling resulting in the discovery of copper prospects at the location of the Johnny Mountain Gold Mine. In 1964, Cominco Ltd. optioned claims from Tuksi Mining Company and Jodi Explorations Ltd. and in 1965 completed drilling on the Red Bluff claim for its copper content. In 1973 and 1974 the property was examined by Texas Gulf Sulphur Inc. for its copper and base metal content.

In 1980, Skyline restaked the claims and initiated exploration on the Pickaxe Vein and adjacent area to define its gold potential. In 1981, the Discovery Vein was discovered and subsequent drilling was completed. In 1982 Skyline continued drilling the Discovery Vein and other targets resulting in the discovery of a high grade gold vein that became known as the 16 Vein.

In late 1982, Skyline entered into an agreement with Placer Development Ltd. to explore the property. Placer in turn entered into a joint venture with Anaconda Canada Exploration Ltd. and the joint venture completed exploration during 1983 and 1984.

In late 1984, Skyline completed deep drilling on the 16 Vein and established depth continuity to this gold bearing quartz sulphide vein. From 1985 to 1988 Skyline continued with the surface and underground exploration and development on the several veins that comprise the Stonehouse Gold Deposit.

In August 1988, the Johnny Mountain Gold Mine commenced production. During the period August 1988 to September 1990 a total of 207,058 short tons were milled at an average rate of 323 tons per day grading 0.474 ounces gold per ton. A total of 84,806 ounces of gold, 133,039 ounces of silver and 2,163,000 pounds of copper was produced.

The gold recovery averaged 86.4%. Operations were suspended due to declining gold grades at the end of September 1990.

The mine was restarted in 1993 for three months resulting in the production of an additional 23,762 short tons. This brought the total metals produced to 92,500 ounces of gold, 145,000 ounces of silver and 2,300,000 pounds of copper for total revenue of $45 million.

Androne Resources Ltd. (later Pezgold Resources Ltd.) optioned claims to the south of the mine from Skyline (Burnie and Dan claims) and performed exploration programs in 1987 and 1988. Work comprised geochemistry, prospecting, trenching and geologic mapping. A number of anomalous areas in gold were discovered.

Additionally, Skyline completed large geochemical, geophysical and prospecting programs during 1988, 1989 and 1990 between the mine and the northern and northeastern portion of the claims. These programs resulted in reconnaissance diamond drilling of numerous promising gold targets as well as directed drilling of the Road Show gold vein in 1988, the Red Bluff copper, gold porphyry target in 1988 and the C-3 shear hosted gold prospect in 1990. Several million dollars of flow through exploration funds were spent on these programs.

Skyline also completed exploration programs on behalf of Placer Dome Inc. in 1990 and 1991 on an optioned block of claims on the northeastern portion of the property known as the Bronson Creek Project. Placer was exploring for the southeastern extension of the formerly producing Snip Gold Mine that adjoins the northern boundary of the Iskut Property. In excess of one million dollars was spent on geophysical, geochemical, trenching, prospecting, geologic mapping and diamond drilling programs.

During 1991, Adrian Resources Ltd. performed exploration work on the northwest portion of the claims under an earn-in option agreement. The work comprised geophysics, geochemistry, prospecting, geologic mapping, trenching and diamond drilling. Numerous targets were identified and the SMC Zone, thought to be a gold and base metal, shear hosted deposit, received the bulk of the drilling. Expenditures were reported to be 1.3 million dollars.

At the same time, during 1990 and 1991, Skyline was performing prospecting, geologic mapping, trenching and drilling on shear hosted gold targets on the Burnie claims to the south of the Adrian work. This work was based on the earlier work by Androne/Pezgold and discovered numerous interesting targets.

In 1993, Skyline signed an exploration agreement with Cominco Ltd. in which Cominco performed exploration on a portion of the northeast area of the property. Cominco's interest was in finding a deposit similar to the Snip Gold Mine. During the period 1993 to 1995, Cominco spent approximately $1.4 million on geologic mapping and diamond drilling.

Skyline performed a limited program of Induced Polarization and diamond drilling on the Red Bluff gold, copper, $\pm$ molybdenum porphyry system in 1993. This led to an extensive program of advanced exploration and feasibility study during the period 1994 to 1997; during which time, the deposit was re-named the Bronson Slope porphyry deposit. Field work was stopped in 1998 due to declining metal prices and loss of investor confidence resulting from the Bre-X scandal.

In 1999, Skyline reached an agreement with Homestake Canada Inc. whereby Skyline was given controlled access to the Snip Mine workings to perform underground exploration on an area of Skyline's ground immediately adjacent to the Snip workings. Homestake would act as mining and drilling contractor to Skyline to perform the work, and a revenue sharing agreement was agreed upon should Homestake elect to participate in the mining and milling of any ore developed on the claim. Homestake retained a production royalty on the ground from an earlier agreement. Financing for the work was provided by Royal Gold, Inc. of Denver Colorado in exchange for a royalty on any gold produced from the property. The cost of the program was $CDN300,000.

Since 1999, Skyline's activities on the property have comprised a number of small reclamation programs as well as an examination of the tailings at the Johnny Mountain Gold Mine for their gold content and the recoverability of the gold.

4.2 Mining and Milling

The former Johnny Mountain gold mine extracted ore from narrow quartz plus sulphide veins. Gold occurred free in quartz and on sulphide grain boundaries. Sulphides comprised predominantly pyrite with secondary chalcopyrite and minor sphalerite, pyrrhotite and galena. Underground exploration commenced in 1986 and the property began commercial production in 1988. Commercial production lasted until September 1990. After a brief closure, the mine was re-opened for three months in the fall of 1993. There has been no mining since 1993.

Mining was by shrinkage stoping. The ore was transported by trackless equipment. Four levels were developed in the mine, three above the lowest portal and one below. The lowest level was accessed by an internal decline that is presently filled with water.

The concentrator contained gravity and flotation circuits. A gravity concentrate was removed from the freshly ground ore by Denver lead shot jigs. This concentrate was cleaned on a shaking table. The remainder of the ore was concentrated by rougher flotation then three stages of copper cleaner flotation. Tails were re-ground and passed through a Falcon centrifugal concentrator. The Falcon concentrate was also cleaned on the shaking table. The final tails were pumped to a water retaining, compacted earth fill dyke tailings impoundment.

4.3 Production Statistics

The following table is a compilation of the monthly production reports from the operating period 1988 to 1990. In addition to the 84,806 ounces of gold, 133,039 ounces of silver and 2,162,594 pounds of copper were produced.

Units used are Troy ounces and short Tons.

Table 4.3-1: Johnny Mountain Gold Mine Quarterly Production Data Fiscal 1988, 1989, 1990

Yr	Qtr	Tons Milled	Tons Per Day	Calc. Head Oz/T	Gold Recovery %	Gold Ounces Recovered	Calc. Tails Oz/T	Calc. Tailings Ounces
88	4 th	11,706	an	0.36	67.7	2,853	0.116	1,361
89	1 st	20,080	244	0.635	81.3	10,406	0.119	2,345
89	2 nd	25,162	307	0.532	83.5	11,174	0.088	2,212
89	3 rd	25,837	308	0.554	85.0	12,167	0.083	2,147
89	4 th	29,152	317	0.511	86.3	12,862	0.070	2,035
89	TOT	100,231	295	0.552	84.2	46,609	0.087	8,739
90	1 st	30,793	335	0.427	87.1	11,449	0.055	1,697
90	2 nd	30,693	345	0.403	90.6	11,206	0.038	1,163
90	3 rd	33,871	368	0.367	88.8	11,044	0.041	1,387
90	4 th	11,470	396	0.416	87.2	4,498	0.053	274
90	TOT	106,827	354	0.400	89.4	38,197	0.042	4,521
All	All	207,058	323	0.474	86.4	84,806	0.064	13,260

5.0 REGIONAL GEOLOGY

The Iskut River region is within the Intermontane Belt on the western margin of the Stikine terrane. Three distinct stratigraphic elements are recognised in the western portion of the area (Anderson, 1989): (i) Upper Paleozoic schists, argillites, coralline limestone and volcanic rocks of the Stikine Assemblage, (ii) Triassic Stuhini Group volcanic and sedimentary arc related strata, and (iii) Lower to Middle Jurassic Hazelton Group volcanic and sedimentary arc related strata.

Intrusive rocks in the Iskut River region comprise five plutonic suites. The Stikine plutonic suite comprises Late Triassic calc-alkaline intrusions which are coeval with Stuhini Group strata. The Copper Mountain, Texas Creek and Three Sisters plutonic suites are variable in composition but are roughly coeval and cospatial with Hazelton Group volcanic strata. Tertiary elements of the Coast Plutonic Complex are represented by predominantly granodioritic to monzonitic Eocene intrusions of the Hyder plutonic suite, exposed 12 kilometres south of the Bronson Slope deposit (Britton et al., 1990).

The age, mineralogy and texture of the Red Bluff porphyry stock (associated with the Bronson Slope deposit), suggest that it belongs to the metallogenetically important Early Jurassic Texas Creek plutonic suite (Alldrick, 1985; Alldrick et al, 1987; Brown, 1987). Plutons of this suite are widespread in the Stewart, Iskut River region and range in age from 196 to 185 million years (Anderson, 1993; MacDonald et al., 1992).

6.0 PROPERTY GEOLOGY

6.1 Lithology

Strata on the property are divided into a lower and an upper sequence; probably correlating with Triassic Stuhihi Group and Jurassic Hazelton Group respectively. The sequences are separated by a flat lying to gently dipping regional unconformity exposed approximately one kilometre to the north of the Johnny Mountain Gold mine.

The lower sequence comprises folded turbiditic greywackes with interbedded siltstones, mudstones, volcanic conglomerate and rare lenses of carbonate rocks. The sequence is weakly to moderately metamorphosed (lower greenschist facies). Alteration ranges from weak to strong in the vicinity of mineral prospects.

The lower sequence is intruded by the Red Bluff porphyry stock (Bronson Slope deposit), a hydrothermally altered, potassium feldspar megacrystic, plagioclase porphyritic intrusion of probable granodioritic composition. The stock is approximately 2.0 kilometres long, up to 0.3 kilometres wide and trends southeast along the southwest side of the Bronson Creek valley. Contacts of the stock with country rocks are not well defined, but where observed in drill core or underground workings are either faulted or intrusive. The southwest and northeast contacts appear to be southwesterly dipping. The age of the Red Bluff intrusive is lower Jurassic.

Lamprophyre dykes of probable Jurassic age have been mapped at numerous locations on the property and in addition lower Jurassic feldspar porphyry dykes and Tertiary intrusive stocks have been noted. Basalt dykes, possibly correlative with Recent volcanism, have also been observed.

6.2 Structure

To date, with the exception of the Red Bluff porphyry system, all mineral prospects on the property appear to be in veins or silicified shear zones. Most of the mineralized *prospects conform to the following three shear directions:*

- northwest dipping shears ($060^{O}/70^{O}$ NW) - Stonehouse Deposit,
- southwest dipping shears ($120^{O}/45^{O}$ SW) - Snip Deposit and related showings, and
- northeast dipping shears ($130^{O}/45^{O}$ NE) - Burnie Prospect and related showings.

In the case of the Snip shear direction, the shearing may be related to regional folds that vary in intensity from small open fold belts to anticline-syncline pairs that can result locally in overturned bedding. The axial plane cleavage developed in these folds has created weakness in the rock and these zones of weakness have created conditions favourable for shearing in a northwest-southeast direction. The Snip veins appear to be emplaced in a shear zone that has developed in the axial plane cleavage of an anticline inferred from Skyline mapping of the sedimentary rocks further south along the Bronson Creek valley. The Red Bluff porphyry may be emplaced parallel to the axial plane cleavage of the corresponding syncline lying just to the northeast of the Snip anticline.

In the case of the Stonehouse deposit, the vein direction is roughly parallel to the orientation of Jurassic feldspar-porphyry dykes and is likely related to the same structural event. To date, no explanation is apparent for the Stonehouse shear direction as bedding directions are not well constrained in the volcanic conglomerate host rock.

In the case of the Burnie prospect (and others), although geologic mapping has not been carried out to the same extent as at Bronson Creek, the shear direction appears to be related to a possible fold axis as well.

7.0 TAILINGS IMPOUNDMENT DYKES

The tailings impoundment is situated on Johnny Flats, northwest of the mine site and west of the airstrip. It was constructed utilizing pre-existing topography as much as possible; comprising two southeast-northwest trending ridges lying respectively northeast and southwest of a naturally occurring alpine pond. The natural slope above the pond was utilized to form the northwestern flank of the impoundment and a northeast-southwest trending earth fill dyke was constructed to form the flank on the south-eastern side of the impoundment. The southeast-northwest trending ridges were raised by placing earth fill dykes on their crests to form the northeast and southwest flanks.

The dykes were constructed of machine compacted dense basal till excavated from local borrow sources, some of which were located within the impoundment itself. The thickness of dyke material placed on the naturally occurring ridges was generally 1 to 2 metres, whereas the deepest sections of the dyke ranged up to 4.6 metres. The topography uphill of the impoundment was separated from the impoundment by cut-off trenches excavated for the purpose of excluding surface run-off.

The construction was designed and supervised by Mr. R.C. Dick, P. Eng., a geotechnical engineer with considerable experience in the design and construction of such structures. The tailings impoundment discharges via a spillway from a point midway along the northeast flank into a polishing pond wetland then via an excavated ditch into Johnny Creek approximately 750m northeast of the impoundment.

The impoundment covers an area approximately 250m x 350m in size. The capacity of the impoundment was summarized by Mr. Dick in the table of volumes and elevations shown on pg. 6 of the Section 6 application to construct the impoundment (15 September 1987). As the impoundment was constructed in compliance with the permit application, the table is reproduced below.

Table 7.0-1: Tailings Impoundment Storage Volume at Various Crest Elevations

Crest Elevation (ft)	Crest Elevation (m)	Storage Volume to Crest (m3)
3540	1079.0	475,900
3535	1077.5	334,400
3530	1076.0	203,600
3525	1074.4	93,800
3520	1072.9	21,300
3515	1071.4	1,100

The present spillway elevation of 1077.8 m is taken from the survey drawing of the Cross Sections of Completed Dykes accompanying Mr. Dick's Final Dyke Construction Report (12 December 1988). The spillway elevation defines the present water level in the impoundment, which is maintained at 1.0 m above the highest tailings elevation (1076.8 m) to prevent the onset of acid generation in the tailings. The tailings are not distributed at an even elevation. The total design capacity at 1076.8 m elevation can be interpolated from the previous table to be approximately 280,000 m3.

During the period August 1988 to September 1990, 194,269 short tons of tailings were produced at the Johnny Mountain mine. During the three months of operation in 1993, an additional 23,762 short tons of tailings were produced. Therefore, in total 218,031 short tons, or 197,794 metric tonnes, of tailings were produced and are now contained within the impoundment. Assuming a density of 1.60 S.G., the total volume utilized is 123,622 m3 leaving an unused capacity of 156,378 m3 (assuming 1.0 m water cover). This estimate does not take into account some 9,410 m3 that was removed from borrow pits within the impoundment during construction and therefore added to the original design capacity.

Table 7.0-2: Remaining Tailings Impoundment Volumes

Description	Tonnes x 1000	S.G.	Volume m3 x 1000	Elev. m
Present spillway elevation (water level)				1077.8
Design capacity at present water level minus 1m			280.0	1076.8
Total tailings produced	197.8	1.60	123.6	

7.1 *Visual Inspection*

The 2003 inspection of the dykes took place on August 20, 2003. The inspection began at the abutment of the southwest dyke and proceeded along the crest to the abutment of the northwest dyke. The inspection of the toe was made in the opposite order, taking weir flow measurements as the inspection proceeded. Weir locations are shown on Figure 4.

The overall appearance of the dykes appears unchanged from previous years. Individual pebbles and cobbles could be identified in the 2003 inspection that can be identified in photographs from previous years and can be seen to be in the same locations as in previous years.

The southwest and southeast dykes appear unchanged from previous years.

At the northeast corner, a small length (5 metres) of interior dyke slope that had not been covered with rip rap as had the rest of the northeast dyke interior slope during 1994. Some minor wave erosion in this area was noted in the 1995 inspection report, although in subsequent reports it was noted that no further wave damage had occurred. The 2003 inspection of this section indicates that there has been no change since the 2001 inspection.

Midway between the northeast corner and the spillway, minor longitudinal cracks were noted in the rip rap that had been placed on the interior slope of the dyke to protect the dyke from wave erosion. The cracks do not appear to extend into the dyke itself.

The general appearance of the interior slope of the east half of the northeast dyke has not changed since the placement of the rip rap. Individual rocks could be identified in the 2003 inspection of the dyke in locations unchanged from previous years.

The spillway chute and discharge channel appear unchanged from previous years.

The interior slope of the west half of the northeast dyke and northwest dyke appear unchanged from previous years.

At the northwest abutment, the inner slope of the northwest dyke and the saddle dam appear unchanged.

The sloped sides of the excavation and the drainage associated with the refuse site located between the northwest abutment and the southwest abutment appear unchanged from previous years.

The exterior slope and ditch of the northwest dyke and at the northwest corner appear unchanged.

The condition of the waste material placed on the exterior slope of the northeast dyke (west of the spillway) appears unchanged.

The condition of the buttress placed against the exterior slope of the east half of the northeast dyke (east of the spillway) and also used as an access road to the crest and to the top of the spillway appears unchanged.

From the northeast corner to the southwest saddle dam no changes were observed in the condition of the exterior slopes and ditches. The two marmot holes below the southwest dyke noted in previous inspection reports do not appear to have changed and are apparently uninhabited at present.

7.2 Seepage Collection Trenches and Seepage Measurement Weirs

A series of trenches have been excavated at the base of the outer slopes of the dykes in the underlying tills for the purpose of collecting water that seeps through the dykes. These trenches were constructed upon completion of the dykes.

Earth fill seepage measurement weirs have been constructed by hand at a number of places in the trenches for the purpose of measuring the flow of the seepage. The weirs were constructed with small plastic pipes through the crests of the weirs through which the seepage flows. The flow is measured by placing a vessel of known volume beneath the downstream end of the pipe and measuring the time it takes for the flow of seepage to fill the known volume. By keeping a record of the seepage flows, unusual conditions can be detected and corrections made to the dykes in a timely manner. Turbidity (or the lack thereof) is also observed. Turbid conditions would indicate erosion taking place within the dykes and corrective measures would similarly need to be commenced. See Figure 4 in Appendix 4 for the trench and measurement weir locations.

The 2003 measurements are presented with a record of historical measurements in Table 3 of Appendix 5 of this report.

7.3 Piezometer Measurements

Piezometers have been installed at a number of locations in the impoundment dykes and in the soils surrounding the dykes. Piezometers are tubes installed vertically in the ground to beneath the water table. The 1" diameter plastic tubes are kept open by a perforated section near the bottom of the tube. Fitted plastic caps are installed to keep out debris, unless the piezometer demonstrates artesian flow. See Figure 4 in Appendix 4 for the piezometer locations.

The piezometers are measured using a length of coaxial cable. The cable is inserted into the tube until it meets refusal. The cable is marked with a length of plastic flagging tape and extracted from the piezometer. The water level is visible on the walls of the cable when it is first extracted, and the water level is also marked with flagging. The depth from the top of the piezometer to water is recorded, as is the depth from the top of the piezometer to refusal.

Records have been kept since the installation of the piezometers. The 2003 measurements are presented with a record of historical measurements in Tables 1 and 2 of Appendix 5 of this report.

8.0 TAILINGS SAMPLING

8.1 Sampling Method

The tailings were exposed by setting up siphons using lengths of pipe available at the site. The water level was lowered approximately 2.4 metres, exposing a considerable surface area of the tailings.

Two samples were taken at each station for a total of 26 samples. The samples were taken using a soil sampling auger. The first sample was taken from the first 0.1 metre of depth. The second was taken from approximately 1.0 metre of depth. The samples were labeled using the grid location with the suffix "A" and "B" respectively.

Each sample was placed individually in a Kraft soil sample envelope. The sample weights were approximately 0.36 kg each.

The samples were air dried and placed inside 12 inch by 20 inch 6 mil poly bags which were then packed inside woven poly sacks and shipped via Canadian Airlines to ALS Chemex Labs in North Vancouver, B.C. The samples were fire assayed for gold using a 30 gram assay ton and an atomic absorption finish. Samples containing greater than 10 grams gold per tonne were re-assayed using a gravimetric finish.

See Appendix 6 for the assay certificate.

8.2 Sample Locations

A sample grid was established on the surface of the tailings using wooden lath pickets. Surveyors tape was attached to the lath and sample stations were marked on the tape.

The sample grid was surveyed using Hip Chain and compass from a baseline established in 2001 that had been used for sampling another area of the tailings. Cross lines were run at 20 metre intervals and stations were placed 20 metres apart on the cross lines. See Figure 5 in Appendix 4 for sample locations and results.

Two samples were taken from the bottom of a pit that had been excavated in the tailings for a bulk sample to test for gold recovery. These two samples are labeled 10N 200E A and B.

8.3 Results

With the exception of the two samples taken from 10N 200E, assays ranged from 0.761 g/t to 5.95 g/t gold. Higher results were obtained from samples in Line 40N and from samples closer to the baseline (200E) than from samples farther to the east. This is likely due to the gold being more dense than the rest of the tailings and being deposited closer to the spigot point (0N 200E) whereas the less dense fraction of the tailings would be carried farther from the spigot point. Metallurgical testing of the tailings sampled in 2001 (closer to the spigot point) indicated that gold could be recovered from tailings containing higher amounts of gold. Metallurgical testing has not been performed on the tailings sampled in 2003.

9.0 CHEMISTRY OF AIRSTRIP SURFACE

9.1 Sampling Method

The airstrip was constructed of native tills occurring at the site. These tills become easily saturated when exposed to precipitation and will not withstand the tire pressure of vehicular traffic or of aircraft. A more porous layer of rock fill was placed in order to pave the runway surface. Some of the rock used to pave the runway surface comprised sulphide bearing waste rock from the mining. The runway surface required characterization with respect to Acid Base Accounting as a preliminary measure in the determination of its potential to form Acid Rock Drainage.

The airstrip was sampled by taking a composite sample across the entire width of the airstrip at each sample location. The composite was formed by taking approximately 20 to 30 cm3 of rock and till at 0.5m intervals along the length of the composite. Composite sample weights were approximately 5 to 9 kilograms each.

The samples were collected in 12 inch by 20inch 6 mil poly bags. The samples were air dried then packed in woven poly sacks and shipped to BC Research Inc. of Vancouver, B.C. to be analysed using acid base accounting procedures.

9.2 Sample Locations

The airstrip was sampled at 100m intervals. The southern end of the airstrip was designated as 0m and the northern end as 1440m. A total of fourteen samples were collected from the airstrip. The sample locations are shown on Figure 3 in Appendix 4.

The following table presents the information collected during the sampling.

No.	Location	Sample Length	Airstrip Profile		Remarks
			West	East	
5784	0+00m	34.0m	0.5m fill	0.5m fill	mine rock, minor S, to Stonehouse Ck.
5785	1+00m	35.5m	0.5m fill	0.5m fill	mine rock, minor S, to Stonehouse Ck.
5786	2+00m	33.5m	0.5m fill	0.5m fill	mine rock, minor S, to Stonehouse Ck.
5787	3+00m	32.0m	cut	cut	minor mine rock, little S, stream divide
5788	4+00m	30.0m	cut	cut	minor mine rock, little S, to Johnny Ck.
5789	5+00m	30.5m	cut	cut	no mine rock, little S, to Johnny Ck.
5790	6+00m	31.0m	cut	cut	no mine rock, little S, to Johnny Ck.
5791	7+00m	29.0m	cut	cut	no mine rock, little S, to Johnny Ck.
5792	8+00m	33.5m	even	1.5m fill	no mine rock, little S, to Johnny Ck.
5793	9+00m	37.0m	3.0m fill	1.0m fill	no mine rock, little S, to Johnny Ck.
5794	10+00m	43.0m	1.0m fill	1.0m fill	minor mine rock, little S, to Johnny Ck.
5795	11+00m	47.0m	even	5.0m fill	no mine rock, little S, to Johnny Ck.
5796	13+00m	42.5m	5.0m fill	15.0m fill	no mine rock, little S, to Johnny Ck.
5797	14+40m	42.0m	0.5m fill	1.5m fill	no mine rock, little S, to Johnny Ck.

9.3 Results

The Certificate of Analysis issued by BC Research Ltd. is shown in Appendix 6. The results are tabulated in Table 4 in Appendix 5.

The paving on the southern part of the airstrip, represented by samples 5784 to 5788, contained mine rock. The Neutralization Potential Ratio of the samples (the ratio of neutralization potential to maximum potential acidity) ranged from 0.5 to 2.6 and had an arithmetic mean value of 1.0. The three lowest values, in samples 5784 to 5786, indicate that these samples would likely produce acid drainage at some time in the future. To prevent acid formation from occurring, the paving layer will likely have to be mixed with the underlying tills using deep scarification. The underlying tills must be characterized for their acid base accounting characteristics to demonstrate that this technique will be successful.

The paving used on the northern part of the airstrip, represented by samples 5789 to 5797, contained no mine rock other than a minor amount at sample location 5794. The Neutralization Potential Ratio ranged from 5.0 to 31.4 and had an arithmetic mean value of 9.6. These samples will not produce acid drainage.

10.0 CHEMISTRY OF MAGAZINE ROAD

10.1 Sampling Method

The magazine road was constructed of native tills occurring at the site as well as mine rock where insufficient till was available for construction. These tills become easily saturated when exposed to precipitation and will not withstand the tire pressure of vehicular traffic. A more porous layer of rock fill was placed in order to pave the road surface. Some of the rock used to pave the road surface comprised sulphide bearing rock from the mine. Other sections of the road were constructed entirely of sulphide bearing rock from the mine. The road construction materials required characterization with respect to Acid Base Accounting as a preliminary measure in the determination of its potential to form Acid Rock Drainage.

The magazine road was sampled by taking a composite sample across the entire width of the road at each sample location. In addition, the outer side slopes of the road were sampled separately in some locations. The composites was formed by taking approximately 20 to 30 cm3 of rock and till at 0.5m intervals along the length of the composite. Composite sample weights were approximately 2 to 3 kilograms each.

The samples were collected in 12 inch by 20 inch 6 mil poly bags. The samples were air dried then packed in woven poly sacks and shipped to BC Research Inc. of Vancouver, B.C. to be analysed using acid base accounting procedures.

10.2 Sample Locations

The magazine road was surveyed by Hip Chain and found to be 710 metres in length from its start at the edge of the 10 Level waste rock storage area (0+00m S) to the powder magazine site (7+10m S).

The road was spot sampled at irregular intervals to gather information on sections obviously containing greater amounts of mine rock. A total of 10 samples were taken. The following table presents the information gathered at each sample site.

No.	Location	Road Width	Road Profile		Remarks
			West	East	
314451	0+12m S	5.5m	2.1m	4.6m	Width section only
314452	0+12m S			4.6m	East profile only
314453	0+65m S	6.0m			Width section only
314454	0+65m S		2.5m		West profile only
314455	0+65m S			3.1m	East profile only
314456	1+00m S	7.0m	1.5m	2.0m	West, width and east, all in same sample
314457	1+70m S	6.0m	3.5m	3.5m	West, width and east, all in same sample
314458	3+00m S	7.0m	4.0m	0.3m	Width section only
314459	5+00m S	6.0m	1.5m	0.0m	West and width sections only
314460	6+85m S	8.0m	3.5m	2.5m	West, width and east, all in same sample

In addition, the following information was gathered from observing the condition of the road fill:

- 0+00m to 0+45m: mine rock fill; low S
- 0+45m to 0+85m: mine rock fill; high S
- 0+85m to 1+57m: mine rock fill; low S
- 1+57m to 1+90m: mine rock fill; high S
- 1+90m to 2+15m: mine rock fill; low S
- 2+15m to 6+60m: road built on native till with thin layer of low S mine rock paving
- 6+60m to 6+75m: mine rock fill; moderate S
- 6+75m to 7+10m: mine rock fill; high S

10.3 Results

The Certificate of Analysis issued by BC Research Ltd. is shown in Appendix 6. The results are tabulated in Table 4 in Appendix 5.

The neutralization potential ratio results for the magazine road ranged from 0.1 to 2.2. The arithmetic mean of the neutralization potential ratio for the samples taken was 0.3. The more sulphidized rocks making up the base of the magazine road would likely produce acid rock drainage if left in their present configuration. These rocks may have to be placed subaqueously in the tailings impoundment.

10.4 Control Sample

A control sample was taken from fresh tills exposed in the road cut at 0+80m S. The sample site was located on the east side of the ditch on the east side of the road. The sample was taken from a 1.0m high channel cut into the bank of the road cut.

The neutralization potential ratio result for the control sample was 7.7. Almost all of the potential acidity and most of the neutralization potential had been leached out of the soil.

11.0 CHEMISTRY OF WASTE ROCK DUMPS

11.1 Sampling Method

The waste rock dumps were used for storage of mine rock that did not contain sufficient gold to be treated in the concentrator. The waste rock dumps required characterization with respect to Acid Base Accounting as a preliminary measure in the determination of their potential to form Acid Rock Drainage.

Two waste rock dumps were chosen for sampling, the 12 Level waste rock dump and the 11 Level waste rock dump.

The samples were collected in 12 inch by 20 inch 6 mil poly bags. The samples were air dried then packed in woven poly sacks and shipped to BC Research Inc. of Vancouver, B.C. to be analysed using acid base accounting procedures.

11.2 Sample Locations

The 12 Level dump was sampled from within a trench that had been excavated in the dump. The trench direction was due north-south and the sample was taken from the east wall, at 30.0 metres north of the south end of the trench. The sample (sample number 5799) was taken from the east wall of the trench. The sample comprised a 2.5m vertical channel of dump material. The material sampled generally comprised rock fragments less than half inch in size down to very fine grained material. The sample weighed approximately 4 kilograms.

The 11 Level dump was sampled on the north side of the dump, at 40m in a 030 degree direction from the northwest corner of the concrete slab on the surface of the dump. Two channels were cut for sampling purposes and the material was combined into one sample (sample number 5800). The channels were 2.0 m apart in an east-west direction and were 2.0m in height. The material sampled generally comprised rock fragments less than half inch in size down to very fine grained material. The sample weighed approximately 4 kilograms.

11.3 Results

The Certificate of Analysis issued by BC Research Ltd. is shown in Appendix 6. The results are tabulated in Table 4 in Appendix 5.

The neutralization potential ratio results for the waste rock dumps were 1.1 for the 12 Level dump and 0.6 for the 11 Level dump. These rocks contained high sulphide. These rocks may have to be placed subaqueously in the tailings impoundment.

12.0 CHEMISTRY AND FLOWS OF UNDERGROUND WATERS

12.1 Sampling Method

A water sampling program was initiated in the underground workings of the Johnny Mountain gold mine to provide information regarding chemistry and flows of the underground water for the purpose of determining the likelihood of acid rock drainage from the underground workings.

The sampling equipment was provided by ALS Environmental (Aurora Laboratory Services Ltd.) of Vancouver, B.C. The equipment comprised sample bottles, bottle labels, filter apparatus, filters, forceps, gloves, dilute nitric acid vials, gel pack coolants and shipping coolers. The equipment was returned to the lab with the samples.

The sampling was performed on August 24, 2003. The bottle labels were filled out and affixed to the sample bottles prior to embarking on the sample traverse. One of the shipping coolers was used to ensure safe and orderly transportation of the sample bottles during sampling.

The portion of the water samples taken for analysis of dissolved metals require filtration of suspended solids in order that only the dissolved metal fraction is available for filtration. Samples for dissolved metals analysis are submitted to the lab in 250ml nalgene bottles. The water samples were collected in the field then filtered in the field and placed in the individual 250ml bottles for submission to the lab. The samples were preserved with dilute nitric acid.

The sample bottles were placed in the shipping coolers with the frozen gel packs and crumpled sheets of used newspaper were used for packing and insulation to prevent damage to the sample bottles, bottle labels and filtration equipment. The shipping coolers were sealed shut using commercially available duct tape.

The shipping coolers were delivered to Air Canada in Smithers, B.C. for shipment to ALS Environmental Labs in Vancouver, B.C.

12.2 Sample Locations

The principal streams of underground water were sampled and measured at locations that characterized the waters from different parts of the mine. A total of six samples were taken.

The sample locations are shown on Figure 6 of Appendix 4 which is a schematic cross section of the underground workings demonstrating the water sample locations and the underground water flow paths. This schematic drawing also contains estimated flows of the various underground water streams.

12.3 Results

All water samples were analysed for the physical parameters of Hardness and pH; the dissolved anions Total Alkalinity and Sulphate; and a suite of 32 Dissolved Metals. The dissolved metals were analysed by inductively coupled plasma and mass spectrometry.

Results of the analyses are presented in Appendix 6.

The sample from the Discovery Zone on the 10 Level (U2) demonstrated acid rock drainage conditions. However, the rest of the mine waters contained sufficient alkalinity, that upon mixing with them the water from the Discovery Zone was neutralized. This is demonstrated by the sample of all underground water taken at the 10 Level portal (JM4), downstream of the point where the Discovery Zone water mixes with the rest of the mine water.

13.0 CHEMISTRY OF SURFACE WATERS

13.1 Sampling Method

The surface water sampling program was performed to in order to monitor the quality of surface waters leaving the site.

See Section 12.1 of this report for a description of the water sampling method.

13.2 Sample Locations

The sample locations are shown on Figure 3 in Appendix 4. In addition, a duplicate sample, JM8, was taken at sample site JM7, and a travel blank of distilled water, JM9, accompanied the program. There was no flow at sample site JM 5 at the time of the sampling.

13.3 Results

All water samples were analysed for the physical parameters of Hardness and pH; the dissolved anions Total Alkalinity and Sulphate; and a suite of 32 Dissolved Metals. In addition, Johnny Creek (JM6) and Stonehouse Creek (JM7) were analysed for total suspended solids. The dissolved metals were analysed by inductively coupled plasma and mass spectrometry.

Results of the analyses are presented in Appendix 6.

To date, the surface water quality draining the site is of an acceptable nature.

Respectfully Submitted,

PROFESSIONAL
PROVINCE OF
D. A. YEAGER
BRITISH COLUMBIA
GEOSCIENTIST

David A. Yeager, P.Geo.

APPENDIX 1

LIST OF REFERENCES

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APPENDIX 2

STATEMENT OF QUALIFICATIONS

I, David A. Yeager, do hereby state:

1. That I am the President of Skyline Gold Corporation, with offices located at RR#1 Site T Box 6, Bowen Island, B.C.
2. That I am a member of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.
3. That I am a graduate of the University of British Columbia (B.Sc., 1972) and have been employed as an exploration and mining geologist since that time.
4. That my experience has given me considerable knowledge in geological, geochemical, geophysical and prospecting techniques as well as in the planning, execution and evaluation of exploration drilling programs.
5. That I have visited and am familiar with the Johnny Mountain Gold Mine property.
6. That the program described in this report was performed by me.

Signed and Sealed on the 29 day of March, 2004.

PROFESSIONAL
PROVINCE OF
D. A. YEAGER
BRITISH COLUMBIA
GEOSCIENTIST

APPENDIX 3

STATEMENT OF COSTS

APPENDIX 3: Statement of Costs
2003 Assessment Report on SKY 10, SKY 11, and REG 2

Details		Cost
Mobilization/demobilization: Vancouver/Smithers/Vancouver Hawkair		
David Yeager		409.35
Raymond Chan		229.67
Mobilization/demobilization: Smithers/Bronson Creek/Smithers		
Northern Thunderbird Air Inc. Cessna 206		2,829.00
Sample Analysis		
BC Research Inc.: 27 samples for Acid Base Accounting @ $100/sample		2,700.00
ALS Chemex: 26 fire assays for gold		467.50
ALS Environmental: 13 water samples		1,820.95
Labour: Ray Chan		
Aug. 20 - Aug 28: 8 days @ $200/day		1,600.00
Food		
Camp and restaurant		446.00
Expenses		
Sample shipments, fuel, taxi, miscellaneous expenses		676.16
Consulting: David Yeager, P.Geo		
Organization and Planning: during period Aug 1 - Aug 18: 4 days @ $360/day		1,440.00
Field Work: Aug 19 - Aug 28: 9 days @ $360/day		3,240.00
Reporting: during period October 15 - November 15: 7 days @ $360/day		2,520.00
	Total	18,378.63

APPENDIX 4

FIGURES

SKY 10, SKY 11, REG 2 Claims - 1:7,500,000 - 104-B-11E

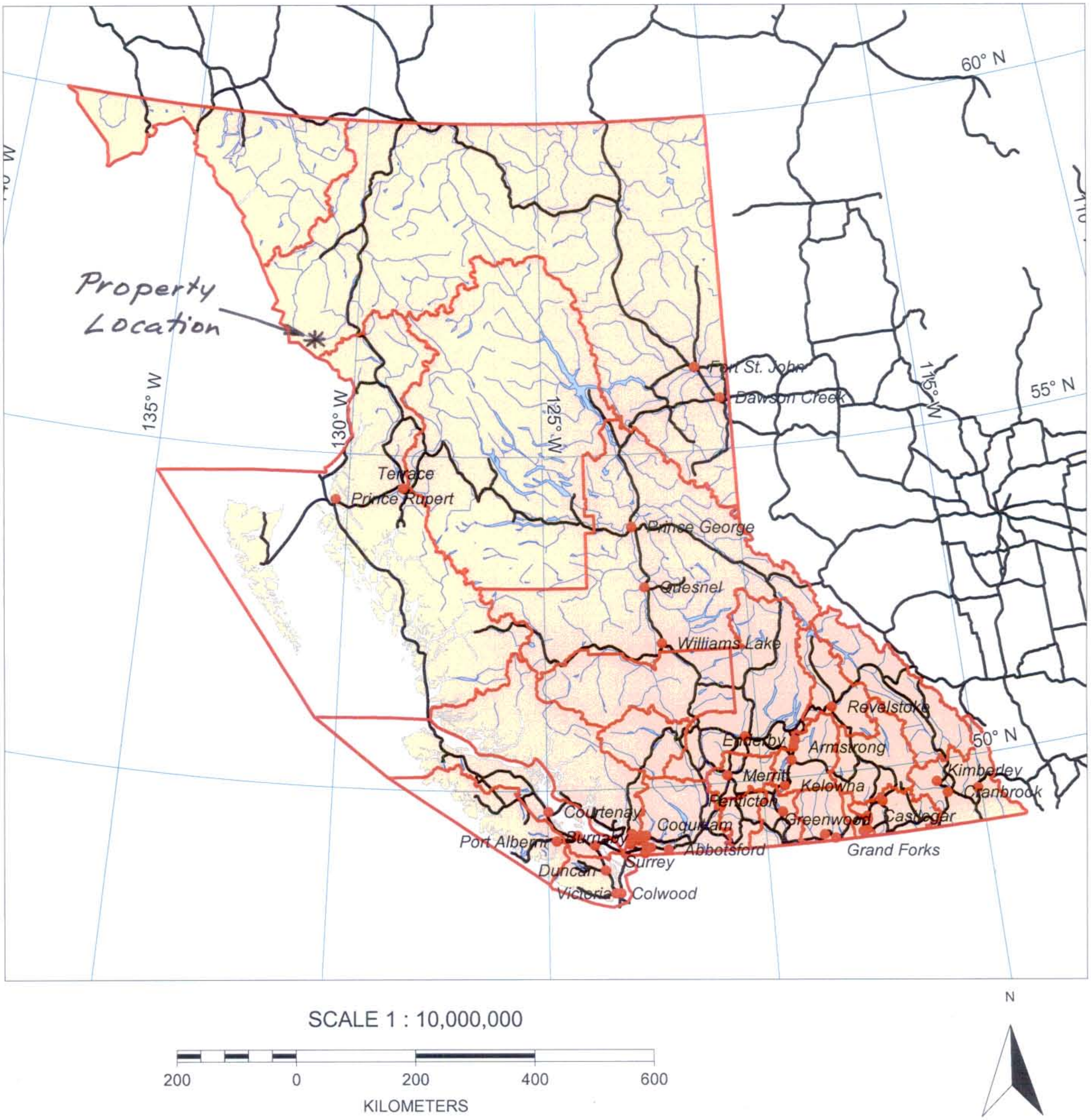
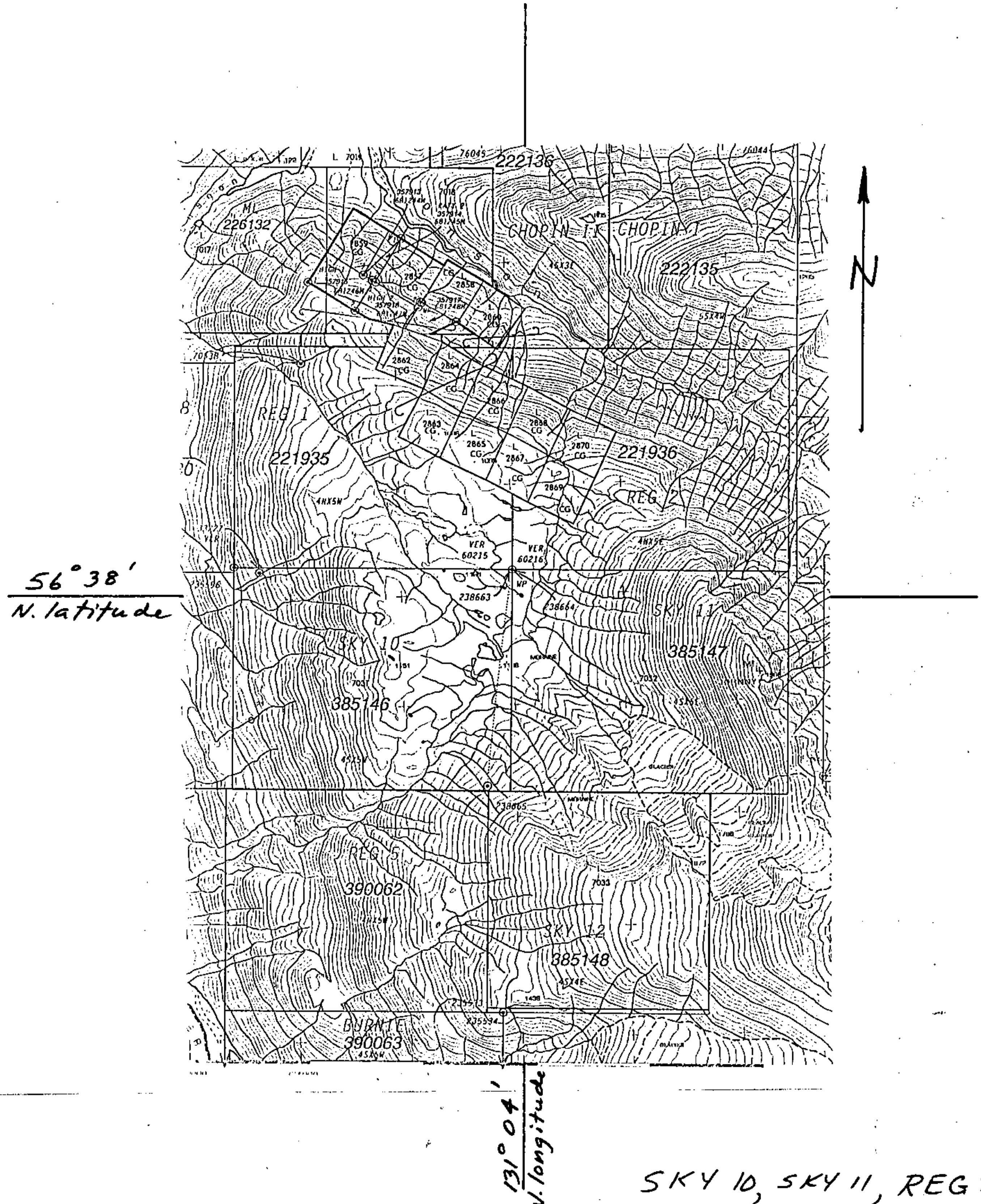


Figure 1

http://webmap.em.gov.bc.ca/mapplace/maps/minpot/dep_find.MWF



SKY 10, SKY 11, REG 2

Claim Location

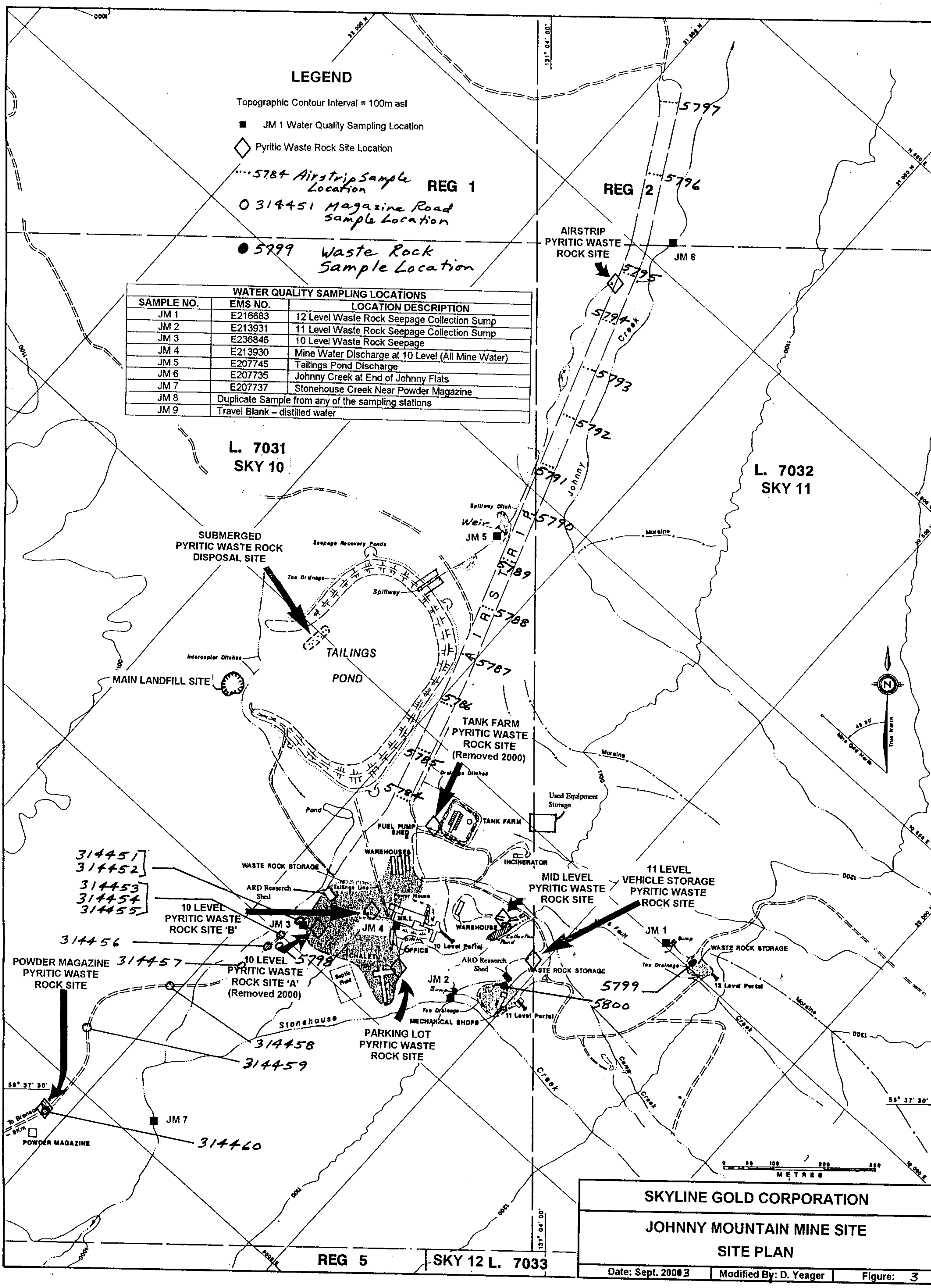
Map

NTS:- 104 B 11 E

March 29, 2004

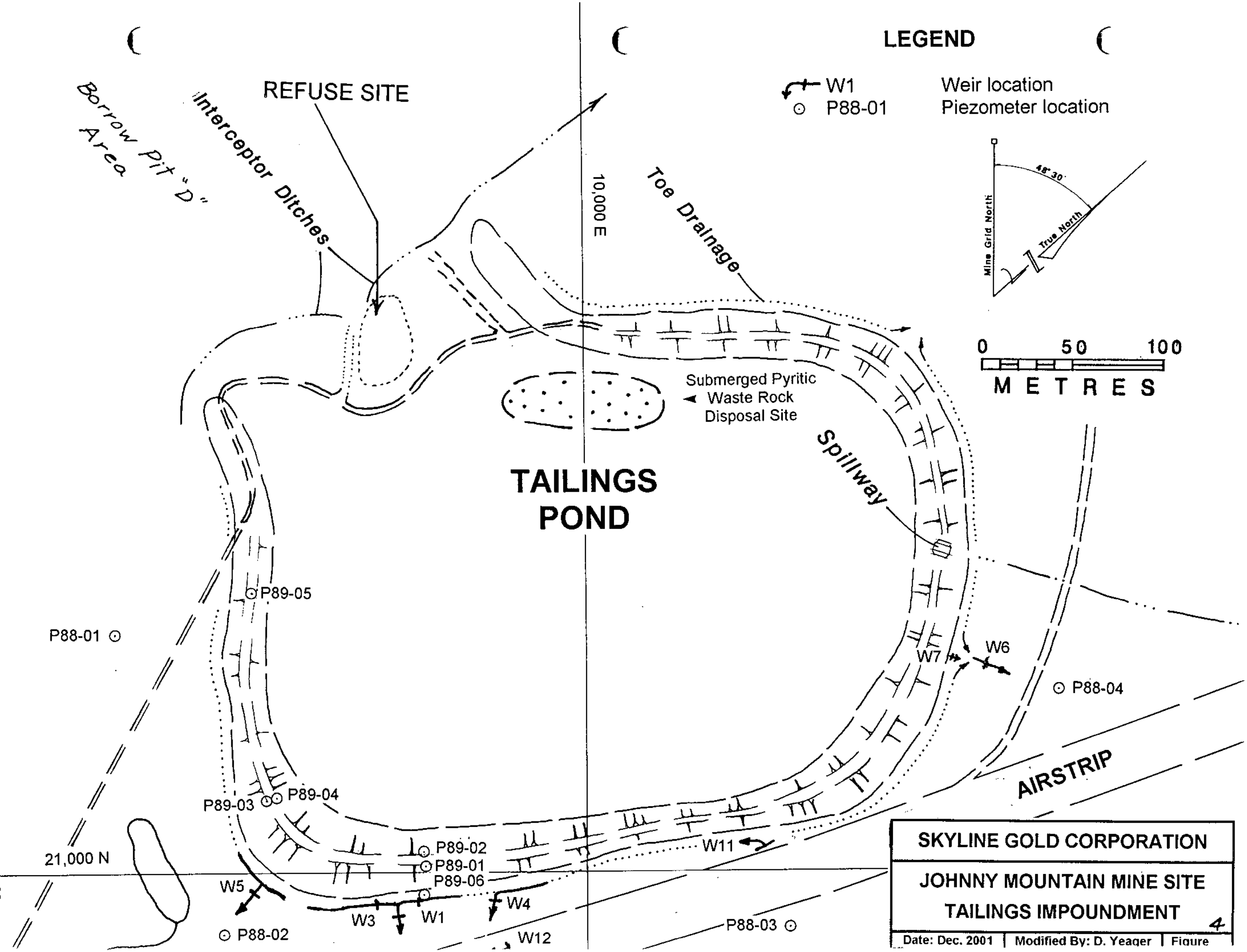
Scale :- 1:50,000

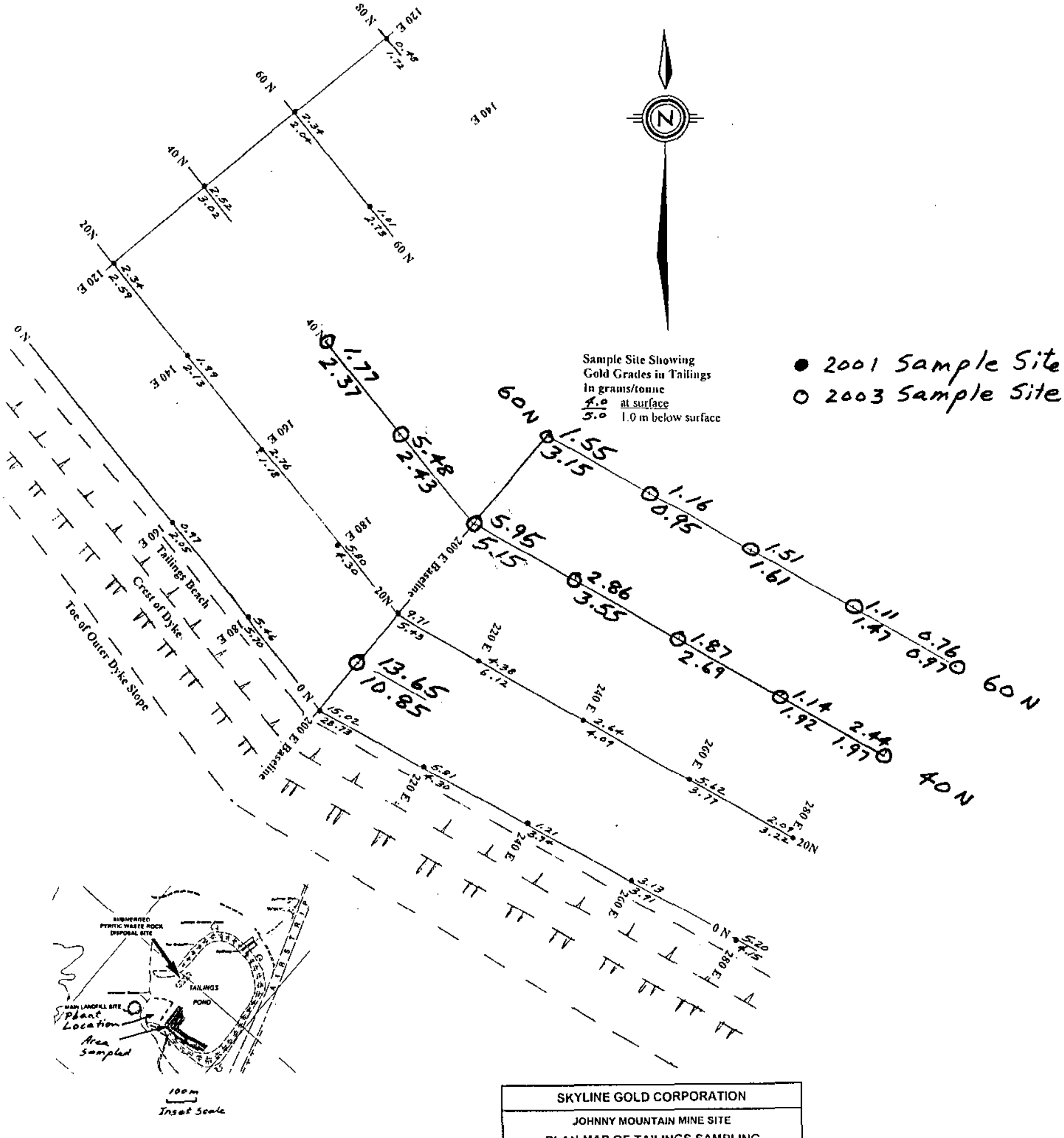
Figure 2



WATER QUALITY SAMPLING LOCATIONS

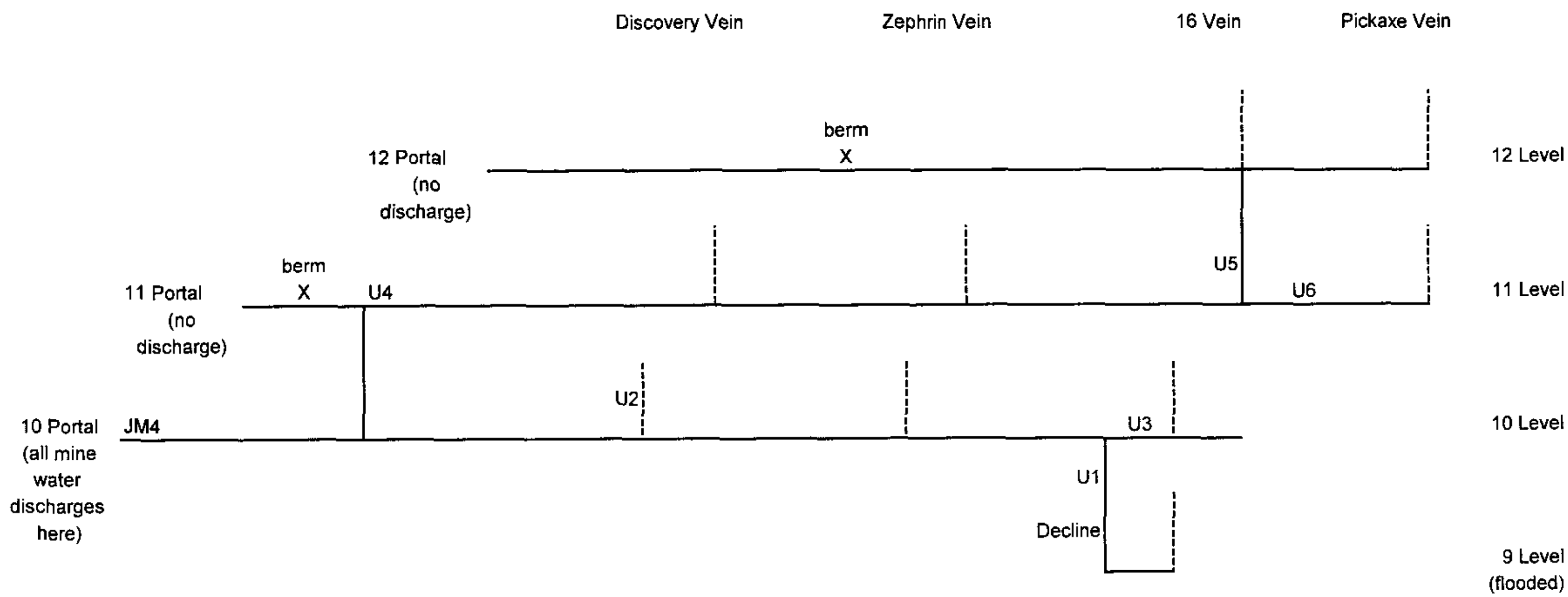
SAMPLE NO.	EMS NO.	LOCATION DESCRIPTION
JM 1	E216683	12 Level Waste Rock Seepage Collection Sump
JM 2	E213931	11 Level Waste Rock Seepage Collection Sump
JM 3	E236846	10 Level Waste Rock Seepage
JM 4	E213930	Mine Water Discharge at 10 Level (All Mine Water)
JM 5	E207745	Tailings Pond Discharge
JM 6	E207735	Johnny Creek at End of Johnny Flats
JM 7	E207737	Stonehouse Creek Near Powder Magazine
JM 8	Duplicate Sample from any of the sampling stations	
JM 9	Travel Blank – distilled water	


LEGEND
W1 Weir location
P88-01 Piezometer location
Mine Grid North
48° 30'
True North
0 50 100
METRES
Borrow Pit "D" Area
Interceptor Ditches
REFUSE SITE
10,000 E
Toe Drainage
Submerged Pyritic Waste Rock Disposal Site
TAILINGS POND
Spillway
P89-05
P88-01
W7
W6
P88-04
AIRSTRIP
P89-03
P89-04
21,000 N
P89-02
P89-01
P89-06
W11
W5
W3
W1
W4
P88-02
W12
P88-03
SKYLINE GOLD CORPORATION
JOHNNY MOUNTAIN MINE SITE
TAILINGS IMPOUNDMENT
4
Date: Dec. 2001
Modified By: D. Yeager
Figure



SKYLINE GOLD CORPORATION		
JOHNNY MOUNTAIN MINE SITE		
PLAN MAP OF TAILINGS SAMPLING		
Date: Nov. 2003	Drawn By: D. Yeager	Figure: 5

Figure 6: SCHEMATIC CROSS SECTION OF JOHNNY MOUNTAIN MINE WORKINGS (looking northeasterly)
Showing Water Sample Locations and Underground Water Flow Paths
Note: Berms Direct All Waters Internally to the 10 Level
David Yeager, P.Geo.



Sample No.	Source	Mine Area Drained	Estimated Flow
U1	Decline	9 Level	2.00 l/sec
U2	Discovery	Above 10, 11 Levels	0.15 l/sec
U3	16	Above 10, 11 Levels	1.85 l/sec
U4	All 11, 12	All 11, 12 Waters	2.25 l/sec
U5	16	Above 11, 12 Levels	1.68 l/sec
U6	Pickaxe	Above 11, 12 Levels	0.57 l/sec
JM4		All Mine Waters	6.70 l/sec

Note: The JM4 flow estimate is a measured figure. All other flows are reasonably accurate, semi-quantitative estimates. U1 + U2 + U3 + U4 should equal JM4. However, JM4 is 0.45 l/sec greater than the sum of U1 to U4. This is attributable to estimation errors plus minor unrecorded flows from other sources on the 10 Level.
Flow measurements and estimates made on August 24, 2003.

APPENDIX 5

TABLES

TABLE 1

SKYLINE GOLD CORPORATION - JOHNNY MOUNTAIN MINE SITE

PIEZOMETER DATA - 1988 PIEZOMETERS

Piezometer	88-01		88-02		88-03	88-04	
Ground Elev.	76.60		68.15		76.35	69.83	
Tube Number	P1	P2	P1	P2	P1	P1	P2
Sealed Section:							
Top	65.0	69.5	59.2	65.0	69.2	62.2	66.2
Last Perf.							
Bottom	61.9	67.7	56.1	63.2	67.6	60.4	65.3
2 Sep. 88	72.93*	72.93*	66.95	67.89	74.41	69.91*	69.85*
Oct. 88	-	-	-	-	-	74.27*	-
24 Aug. 89	72.82*	72.78*	66.95	67.63	76.85	*	69.83
10 Oct. 89	72.94*	72.85*	67.22	67.74	75.13	*	-
19 Oct. 89	-	-	-	-	-	70.29*	-
30 Jul. 90	>73.60*	>73.56*	66.94	-	74.30	-	68.36
2 Sep. 90	-	-	66.94	67.35	-	69.95*	69.36
4 Sep. 90	>73.60*	>73.56*	67.17	65.57	74.45	*	69.57
16 Sep. 90	>73.60*	>73.56*	Dry	Dry	<74.43	*	69.52
29 Sep. 90	>73.60*	>73.56*	67.32	68.12	74.73	*	*
13 Jun. 91	S	S	S	S	S	S	S
2 Jul. 91	S	S	S	S	75.05	S	S
15 Jul. 91	S	S	67.37	67.94	74.12	69.71	*
30 Jul. 91	S	S	67.15	67.75	74.75	69.73	*
16 Aug. 91	>73.60*	>73.56*	67.15	67.35	74.49	69.57	69.65
29 Aug. 91	>73.60*	>73.56*	67.39	67.91	74.91	69.72	*
19 Jul. 92	S	S	S	S	74.99	S	S
5 Aug. 92	>73.60*	>73.56	67.75	67.65	74.80	>69.83*	69.58
18 Sep. 92	>73.60*	>73.56*	67.34	67.88	74.75	-	69.68
27 Sep. 92	>73.60*	>73.56*	67.49	67.98	74.93	>69.83*	69.75
27 Aug. 93	>73.60*	>73.56*	-	-	-	-	-
21 Jul. 94	73.58*	-	66.85	-	-	69.90*	-
28 Jul. 94	73.58*	-	67.78	-	-	69.85*	-
4 Aug. 94	73.59*	-	67.66	-	-	69.85*	-
13 Aug. 94	73.57*	73.54*	67.35	67.43	74.65	70.10*	69.71
26 Aug. 00	>73.60*	>73.56*	68.30	68.30	75.40	>70.10*	>70.10*
27-Sep-01	73.58	>73.56*	67.93	67.93	75.62	>70.10*	69.98
20-Aug-03	73.60	>73.56*	-	68.35	74.93	>70.10*	70.00

* = Artesian S= Covered by Snow

Notes: Elevations are in metres minus 1,000.00

Borrow pit adjacent to 88-03 was filled in prior to 1992

TABLE 2

SKYLINE GOLD CORPORATION - JOHNNY MOUNTAIN MINE SITE

PIEZOMETER DATA - 1989 PIEZOMETERS

Piezometer	89-01	89-02	89-03	89-04	89-05	89-06	Pond Water
Ground Elev.	79.00	79.20	79.40	79.70	79.10	72.50	Spillway=
Sealed Section:							77.8
Top	72.6	73.3	75.4	76.8	78.6	71	
Last Perf.	71.3	72.0	74.1	75.5	77.8	70.2	
Bottom	69.2	69.4	73.3	73.8	75.7	69.4	
20 Oct. 89	73.88	-	-	-	-	-	-
21 Oct. 89	73.10	73.99	<74.10	77.85	-	-	-
22 Oct. 89	-	-	<74.10	76.61	78.78	<70.20	-
23 Oct. 89	72.56	73.36	<74.10	76.39	78.78	<70.20	74.70
24 Oct. 89	-	-	-	-	78.76	-	-
28 Oct. 89	<69.20	-	<73.30	<75.50	-	<70.20	-
2 Nov. 89	<69.20	-	<73.30	<75.50	-	<70.20	-
15 May 90	-	-	-	-	-	-	76.42
31 May 90	-	-	-	-	-	-	76.40
1 Jun. 90	-	-	-	-	-	-	76.40
18 Jun. 90	-	-	-	-	-	-	75.90
1 Jul. 90	-	-	-	-	-	-	75.00
30 Jul. 90	<69.20	76.30	73.75	-	78.26	70.83	-
2 Aug. 90	-	-	-	-	-	-	74.91
4 Sep. 90	<69.20	<69.40	<73.70	-	-	70.68	74.74
16 Sep. 90	72.20	73.45	73.87	-	-	70.72	-
29 Sep. 90	73.60	73.69	76.10	-	-	70.80	75.48
13 Jun. 91	72.72	S	76.31	C	78.73 D	S	78.00
2 Jul. 91	<74.60	S	74.99	C	77.89	S	78.00
15 Jul. 91	72.73	74.49	73.97	C	78.59	70.74	78.00
30 Jul. 91	72.33	74.20	73.87	C	78.53	70.77	78.00
16 Aug. 91	73.56	74.67	73.96	C	78.19 R	70.78	78.00
29 Aug. 91	73.15	74.59	75.36	C	79.10 F	70.68	78.00
9 Jun. 92	78.50	S	78.40	S	S	S	78.00
15 Jun. 92	76.60	S	77.98	S	S	S	78.00
12 Jul. 92	73.85	74.55	77.62	S	78.10	S	78.00
19 Jul. 92	72.81	74.33	74.07	S	78.40	S	78.00
5 Aug. 92	73.36	73.50	74.04	C	78.39	70.70	78.00
18 Sep. 92	73.29	74.30	74.10	C	78.72	70.70	78.00
27 Sep. 92	73.28	74.30	74.05	C	78.74	70.70	78.00
27 Aug. 93	<75.84	<76.32	<76.37	C	?	<70.72	78.00
15 Nov. 93	73.30	74.00	74.10	S	S	S	78.00
21 Jul. 94	73.10	73.89	74.03	C	78.52	70.71	78.00
28 Jul. 94	73.01	73.69	73.95	C	78.26	70.72	78.00
4 Aug. 94	72.95	73.60	73.99	C	78.16	70.75	77.64
13 Aug. 94	72.86	73.41	73.98	C	78.02	70.74	77.02
7 Oct. 95	72.21	73.79	73.46	C	B	70.15	77.40
24 Sep. 99	B	B	74.12	C	B	B	77.40
26 Aug. 00	71.35	73.59	74.02	C	78.60	70.95	77.40
27-Sep-01	72.27	73.51	74.06	C	78.71	70.89	76.60
20-Aug-03	71.80	73.00	74.07	C	C	70.95	75.40

S = Covered by snow C = Covered by soil

R = Rain entering tube F = Tube full of rain water

Note: Elevations are in metres minus 1,000.00

TABLE 3
SKYLINE GOLD CORPORATION
JOHNNY MOUNTAIN MINE SITE
WEIR FLOW MEASUREMENTS (litres/minute)

Weir / Date	1	2	3	4	5	6	7
1989	2.19						
	4.72						
1990	0.77						
	1.30						
1991	3.00						
1992	0.70		0.00	0.00	6.00	0.00	0.20
	0.97		0.26	0.06	20.00	5.00	3.75
1993	0.00		0.00	0.00	8.10	0.00	0.00
	0.12		0.00	0.00	9.20	0.00	0.00
1994	0.00		0.00	0.00	4.00	0.00	0.00
	0.26		0.84	0.90	12.00	6.36	0.20
1995	1.48		0.64		10.00	5.00	0.56
	4.23		1.29		11.61		
1999	4.00		0.70	3.60	15.00	6.00	0.70
				6.00			
2000	2.40		1.50	4.00	16.00	2.00	1.10
2001	2.00		0.50	1.50	10.80	2.28	
2003	1.36		0.44	1.58	5.00	1.76	0.36

Notes:

Previous annual inspection reports have presented complete data from all weir readings from 1989 onward, and attempted to correlate flows with precipitation. The occasional nature of the weir measurements, and the lack of continuous precipitation data, make such analysis approximate at best. In the above table, ranges are given where two or more measurements have been made.

Weir 1 data from 1989 omits the high flows on the first day of exposure, believed due to water stored in nearby soil piles being released by excavation.

Table 4: ABA Results by Standard Sobek Method for Johnny Mountain Samples - August 29, 2003

Airstrip (south)

Sample	Paste pH	CO2 (Wt.%)	CaCO3 Equiv. (Kg CaCO3/Tonne)	Total Sulphur (Wt.%)	Sulphate Sulphur (Wt.%)	Sulphide Sulphur* (Wt.%)	Maximum Potential Acidity** (Kg CaCO3/Tonne)	Neutralization Potential (Kg CaCO3/Tonne)	Net Neutralization Potential (Kg CaCO3/Tonne)	Fizz Rating	Neutralization Potential Ratio (NP/MPA)
5784	7.8	1.41	32.0	1.52	<0.01	1.52	47.5	53.2	5.7	moderate	1.1
5785	7.2	1.05	23.9	1.56	<0.01	1.56	48.8	41.8	-7.0	moderate	0.9
5786	7.2	0.93	21.1	3.41	0.01	3.40	106.3	49.4	-56.9	moderate	0.5
5787	8.1	1.59	36.1	0.62	<0.01	0.62	19.4	46.8	27.4	moderate	2.4
5788	8.0	1.61	36.6	0.71	<0.01	0.71	22.2	57.0	34.8	moderate	2.6
90th Percentile	8.1	1.6	36.4	2.7		2.7	83.3	55.5	31.9		2.5
Mean	7.7	1.3	30.0	1.6		1.6	48.8	49.6	0.8		1.0
10th Percentile	7.2	1.0	22.2	0.7		0.7	20.5	43.8	-36.9		0.6
Std Deviation	0.4	0.3	7.1	1.1		1.1	34.9	5.8	36.3		0.9

Airstrip (north)

Sample	Paste pH	CO2 (Wt.%)	CaCO3 Equiv. (Kg CaCO3/Tonne)	Total Sulphur (Wt.%)	Sulphate Sulphur (Wt.%)	Sulphide Sulphur* (Wt.%)	Maximum Potential Acidity** (Kg CaCO3/Tonne)	Neutralization Potential (Kg CaCO3/Tonne)	Net Neutralization Potential (Kg CaCO3/Tonne)	Fizz Rating	Neutralization Potential Ratio (NP/MPA)
5789	8.0	1.37	31.1	0.31	<0.01	0.31	9.7	48.1	38.4	moderate	5.0
5790	8.1	1.31	29.8	0.23	<0.01	0.23	7.2	46.8	39.6	moderate	6.5
5791	8.3	1.45	33.0	0.17	<0.01	0.17	5.3	49.4	44.1	moderate	9.3
5792	8.3	1.39	31.6	0.09	<0.01	0.09	2.8	50.6	47.8	moderate	18.0
5793	8.2	1.2	27.3	0.09	<0.01	0.09	2.8	46.8	44.0	moderate	16.6
5794	8.4	1.48	33.6	0.20	<0.01	0.20	6.3	49.4	43.2	moderate	7.9
5795	8.1	0.99	22.5	0.04	<0.01	0.04	1.3	39.2	38.0	moderate	31.4
5796	7.4	1.13	25.7	0.07	<0.01	0.07	2.2	26.6	24.4	moderate	12.2
5797	7.0	0.23	5.2	0.03	<0.01	0.03	0.9	13.5	12.6	slight	14.4
90th Percentile	8.3	1.5	33.1	0.2		0.2	7.7	49.6	44.8		20.7
Mean	8.0	1.2	26.6	0.1		0.1	4.3	41.2	36.9		9.6
10th Percentile	7.3	0.8	19.0	0.0		0.0	1.2	24.0	22.0		6.2
Std Deviation	0.5	0.4	8.8	0.1		0.1	3.0	12.8	11.3		8.1

Magazine Road

Sample	Paste pH	CO2 (Wt.%)	CaCO3 Equiv. (Kg CaCO3/Tonne)	Total Sulphur (Wt.%)	Sulphate Sulphur (Wt.%)	Sulphide Sulphur* (Wt.%)	Maximum Potential Acidity** (Kg CaCO3/Tonne)	Neutralization Potential (Kg CaCO3/Tonne)	Net Neutralization Potential (Kg CaCO3/Tonne)	Fizz Rating	Neutralization Potential Ratio (NP/MPA)
314451	7.7	0.77	17.5	0.61	<0.01	0.61	19.1	24.8	5.7	slight	1.3
314452	6.3	0.23	5.2	0.28	0.01	0.27	8.4	13.3	4.9	slight	1.6
314453	6.1	0.46	10.5	1.67	0.02	1.65	51.6	15.5	-36.1	slight	0.3
314454	6.3	0.68	15.5	1.92	0.02	1.90	59.4	18.0	-41.4	slight	0.3
314455	5.7	0.4	9.1	4.18	0.05	4.13	129.1	13.3	-115.8	slight	0.1
314456	5.4	0.16	3.6	0.92	0.02	0.90	28.1	7.0	-21.1	none	0.2
314457	5.2	0.32	7.3	4.95	0.08	4.87	152.2	12.5	-139.7	slight	0.1
314458	7.1	0.76	17.3	0.52	<0.01	0.52	16.3	22.3	6.1	slight	1.4
314459	7.0	0.67	15.2	0.27	<0.01	0.27	8.4	18.8	10.4	slight	2.2
314460	4.9	0.39	8.9	3.96	0.19	3.77	117.8	8.5	-109.3	slight	0.1
90th Percentile	7.2	0.8	17.3	4.3	0.1	4.2	131.4	22.6	6.5		1.6
Mean	6.2	0.5	11.0	1.9	0.1	1.9	59.0	15.4	-43.6		0.3
10th Percentile	5.2	0.2	5.1	0.3	0.0	0.3	8.4	8.4	-118.2		0.1
Std Deviation	0.9	0.2	5.0	1.8	0.1	1.7	54.4	5.7	57.2		0.8

Mine Rock

Sample	Paste pH	CO2 (Wt.%)	CaCO3 Equiv. (Kg CaCO3/Tonne)	Total Sulphur (Wt.%)	Sulphate Sulphur (Wt.%)	Sulphide Sulphur* (Wt.%)	Maximum Potential Acidity** (Kg CaCO3/Tonne)	Neutralization Potential (Kg CaCO3/Tonne)	Net Neutralization Potential (Kg CaCO3/Tonne)	Fizz Rating	Neutralization Potential Ratio (NP/MPA)
5799	8.3	2.3	52.3	2.12	<0.01	2.12	66.3	75.9	9.7	moderate	1.1
5800	8.3	2.23	50.7	3.88	<0.01	3.88	121.3	67.1	-54.2	moderate	0.6

Control

Sample	Paste pH	CO2 (Wt.%)	CaCO3 Equiv. (Kg CaCO3/Tonne)	Total Sulphur (Wt.%)	Sulphate Sulphur (Wt.%)	Sulphide Sulphur* (Wt.%)	Maximum Potential Acidity** (Kg CaCO3/Tonne)	Neutralization Potential (Kg CaCO3/Tonne)	Net Neutralization Potential (Kg CaCO3/Tonne)	Fizz Rating	Neutralization Potential Ratio (NP/MPA)
5798	6.3	<0.01	<0.2	0.02	<0.01	0.02	0.6	4.8	4.2	none	7.7

*Based on difference between total sulphur and sulphate-sulphur

**Based on sulphide-sulphur

APPENDIX 6

CERTIFICATES OF ANALYSIS

ALS Environmental

CHEMICAL ANALYSIS REPORT

Date:	September 23, 2003
ALS File No.	T3356
Report On:	Johnny Mountain Water Analysis
Report To:	**Skyline Gold Corp.** Suite 910 Cathedral Place 925 West Georgia Street Vancouver, BC V6C 3L2
Attention:	**Mr. David Yeager**
Received:	August 29, 2003

ALS ENVIRONMENTAL
per:

Frederick Chen, B.Sc. - Special Projects Manager
Can Dang, B.Sc. - Project Chemist

ALS CANADA LTD.
1988 Triumph Street, Vancouver, BC Canada V5L 1K5
Phone: 604-253-4188 Fax: 604-253-6700 Website: www.alsenviro.com

File No. T3356

RESULTS OF ANALYSIS - Water

Sample ID		U-1	U-2	U-3	U-4	U-5
Sample Date		03 08 24	03 08 24	03 08 24	03 08 24	03 08 24
ALS ID		*1*	*2*	*3*	*4*	*5*
Physical Tests						
Hardness	CaCO3	367	261	167	182	210
pH		8.10	2.95	8.03	8.13	8.22
Dissolved Anions						
Alkalinity-Total	CaCO3	225	<1	88	114	144
Sulphate	SO4	180	430	86	80	83

Results are expressed as milligrams per litre except where noted.
< = Less than the detection limit indicated.

File No. T3356

RESULTS OF ANALYSIS - Water

Sample ID		U-1	U-2	U-3	U-4	U-5
Sample Date		03 08 24	03 08 24	03 08 24	03 08 24	03 08 24
ALS ID		*1*	*2*	*3*	*4*	*5*
Total Metals						
Calcium	T-Ca	114	74.2	48.8	52.1	60.1
Magnesium	T-Mg	20.4	13.7	9.1	9.5	10.2

Results are expressed as milligrams per litre except where noted.
< = Less than the detection limit indicated.

File No. T3356

RESULTS OF ANALYSIS - Water

Sample ID		U-1	U-2	U-3	U-4	U-5
Sample Date		03 08 24	03 08 24	03 08 24	03 08 24	03 08 24
ALS ID		*1*	*2*	*3*	*4*	*5*
Dissolved Metals						
Aluminum	D-Al	<0.002	1.57	0.014	0.018	0.011
Antimony	D-Sb	0.0011	<0.0005	0.0003	0.0003	0.0004
Arsenic	D-As	0.0003	0.0053	<0.0001	0.0001	0.0002
Barium	D-Ba	0.0250	0.0142	0.0194	0.0199	0.0192
Beryllium	D-Be	<0.001	<0.003	<0.0005	<0.0005	<0.0005
Bismuth	D-Bi	<0.001	<0.003	<0.0005	<0.0005	<0.0005
Boron	D-B	<0.02	<0.05	<0.01	<0.01	<0.01
Cadmium	D-Cd	0.0002	0.0070	0.00061	0.00038	0.00020
Calcium	D-Ca	113	79.9	51.0	56.1	65.0
Chromium	D-Cr	<0.001	<0.003	<0.0005	<0.0005	<0.0005
Cobalt	D-Co	0.0015	0.0304	<0.0001	0.0012	0.0003
Copper	D-Cu	<0.0002	9.50	0.0148	0.188	0.0063
Iron	D-Fe	<0.03	62.6	<0.03	<0.03	<0.03
Lead	D-Pb	<0.0001	0.0099	<0.00005	<0.00005	0.00006
Lithium	D-Li	<0.01	<0.03	<0.005	<0.005	<0.005
Magnesium	D-Mg	20.6	14.9	9.5	10.2	11.6
Manganese	D-Mn	1.44	3.00	0.0365	0.251	0.194
Molybdenum	D-Mo	0.0008	<0.0003	0.00049	0.00063	0.00044
Nickel	D-Ni	<0.001	0.004	<0.0005	<0.0005	<0.0005
Phosphorus	D-P	<0.3	<0.3	<0.3	<0.3	<0.3
Potassium	D-K	2	3	2	<2	<2
Selenium	D-Se	<0.002	<0.005	<0.001	<0.001	<0.001
Silicon	D-Si	4.47	3.05	1.48	2.22	1.76
Silver	D-Ag	<0.00002	0.00017	0.00002	<0.00001	<0.00001
Sodium	D-Na	6	<2	<2	<2	<2
Strontium	D-Sr	2.72	0.387	0.444	0.542	0.583
Thallium	D-Tl	<0.0002	<0.0005	<0.0001	<0.0001	<0.0001
Tin	D-Sn	<0.0002	<0.0005	<0.0001	<0.0001	<0.0001
Titanium	D-Ti	<0.01	<0.01	<0.01	<0.01	<0.01
Uranium	D-U	0.00450	0.00090	0.00142	0.00096	0.00153
Vanadium	D-V	<0.002	<0.005	<0.001	<0.001	<0.001
Zinc	D-Zn	0.089	0.963	0.088	0.052	0.032

Results are expressed as milligrams per litre except where noted.
< = Less than the detection limit indicated.

File No. T3356

RESULTS OF ANALYSIS - Water

Sample ID		U-6	JM-1	JM-2	JM-3	JM-4
Sample Date		03 08 24	03 08 25	03 08 25	03 08 25	03 08 25
ALS ID		*6*	*7*	*8*	*9*	*10*
Physical Tests						
Hardness	CaCO3	199	122	211	204	246
pH		8.20	8.04	8.00	7.79	8.20
Dissolved Anions						
Alkalinity-Total	CaCO3	127	64	70	40	139
Sulphate	SO4	87	66	160	177	120

Results are expressed as milligrams per litre except where noted.
< = Less than the detection limit indicated.

File No. T3356

RESULTS OF ANALYSIS - Water

Sample ID		U-6	JM-1	JM-2	JM-3	JM-4
Sample Date		03 08 24	03 08 25	03 08 25	03 08 25	03 08 25
ALS ID		*6*	*7*	*8*	*9*	*10*
Total Metals						
Calcium	T-Ca	55.1	42.9	62.9	68.2	71.2
Magnesium	T-Mg	11.8	2.9	11.3	6.0	13.1

Results are expressed as milligrams per litre except where noted.
< = Less than the detection limit indicated.

File No. T3356

RESULTS OF ANALYSIS - Water

Sample ID		U-6	JM-1	JM-2	JM-3	JM-4
Sample Date		03 08 24	03 08 25	03 08 25	03 08 25	03 08 25
ALS ID		*6*	*7*	*8*	*9*	*10*
Dissolved Metals						
Aluminum	D-Al	0.003	0.001	0.002	0.009	0.014
Antimony	D-Sb	0.0004	0.0001	0.0002	0.0001	0.0006
Arsenic	D-As	0.0002	<0.0001	<0.0001	<0.0001	<0.0001
Barium	D-Ba	0.0188	0.0494	0.0239	0.0397	0.0213
Beryllium	D-Be	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
Bismuth	D-Bi	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
Boron	D-B	<0.01	<0.01	0.01	<0.01	<0.01
Cadmium	D-Cd	0.00048	<0.00005	0.00014	0.00015	0.00025
Calcium	D-Ca	58.6	43.9	65.1	71.2	75.3
Chromium	D-Cr	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
Cobalt	D-Co	0.0002	<0.0001	0.0004	<0.0001	0.0009
Copper	D-Cu	0.0202	0.0007	0.0045	0.0124	0.0414
Iron	D-Fe	<0.03	<0.03	<0.03	<0.03	<0.03
Lead	D-Pb	<0.00005	<0.00005	0.00006	0.00006	<0.00005
Lithium	D-Li	<0.005	<0.005	<0.005	<0.005	<0.005
Magnesium	D-Mg	12.7	3.0	11.9	6.5	14.0
Manganese	D-Mn	0.270	0.00053	0.356	0.00181	0.417
Molybdenum	D-Mo	0.00119	0.00021	0.00035	<0.00005	0.00067
Nickel	D-Ni	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
Phosphorus	D-P	<0.3	<0.3	<0.3	<0.3	<0.3
Potassium	D-K	<2	3	3	3	2
Selenium	D-Se	<0.001	<0.001	<0.001	<0.001	<0.001
Silicon	D-Si	3.01	1.39	1.63	1.74	2.72
Silver	D-Ag	<0.00001	<0.00001	<0.00001	<0.00001	<0.00001
Sodium	D-Na	4	<2	<2	<2	3
Strontium	D-Sr	0.826	0.174	0.347	0.302	1.23
Thallium	D-Tl	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001
Tin	D-Sn	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001
Titanium	D-Ti	<0.01	<0.01	<0.01	<0.01	<0.01
Uranium	D-U	0.00095	0.00009	0.00023	0.00007	0.00224
Vanadium	D-V	<0.001	<0.001	<0.001	<0.001	<0.001
Zinc	D-Zn	0.077	0.002	0.025	0.038	0.036

Results are expressed as milligrams per litre except where noted.
< = Less than the detection limit indicated.

File No. T3356

RESULTS OF ANALYSIS - Water

Sample ID		JM-6	JM-7	JM-8	JM-9 ALS Trav Blank
Sample Date		03 08 25	03 08 25	03 08 25	03 08 15
ALS ID		*11*	*12*	*13*	*14*
Physical Tests					
Hardness	CaCO3	22.5	64.5	65.4	<0.6
pH		7.76	8.00	7.99	6.01
Total Suspended Solids		41	157	-	-
Dissolved Anions					
Alkalinity-Total	CaCO3	20	44	44	61
Sulphate	SO4	3	22	22	<1

Results are expressed as milligrams per litre except where noted.
< = Less than the detection limit indicated.

File No. T3356

RESULTS OF ANALYSIS - Water

Sample ID		JM-6	JM-7	JM-8	JM-9 ALS Trav Blank
Sample Date		03 08 25	03 08 25	03 08 25	03 08 15
ALS ID		*11*	*12*	*13*	*14*
Total Metals					
Aluminum	T-Al	-	-	-	<0.001
Antimony	T-Sb	-	-	-	<0.0001
Arsenic	T-As	-	-	-	<0.0001
Barium	T-Ba	-	-	-	<0.00005
Beryllium	T-Be	-	-	-	<0.0005
Bismuth	T-Bi	-	-	-	<0.0005
Boron	T-B	-	-	-	<0.01
Cadmium	T-Cd	-	-	-	<0.00005
Calcium	T-Ca	8.18	22.0	22.6	<0.05
Chromium	T-Cr	-	-	-	<0.0005
Cobalt	T-Co	-	-	-	<0.0001
Copper	T-Cu	-	-	-	0.0015
Iron	T-Fe	-	-	-	<0.03
Lead	T-Pb	-	-	-	<0.00005
Lithium	T-Li	-	-	-	<0.005
Magnesium	T-Mg	1.0	3.9	3.9	<0.1
Manganese	T-Mn	-	-	-	<0.00005
Molybdenum	T-Mo	-	-	-	<0.00005
Nickel	T-Ni	-	-	-	<0.0005
Phosphorus	T-P	-	-	-	<0.3
Potassium	T-K	-	-	-	<2
Selenium	T-Se	-	-	-	<0.001
Silicon	T-Si	-	-	-	<0.05
Silver	T-Ag	-	-	-	<0.00001
Sodium	T-Na	-	-	-	<2
Strontium	T-Sr	-	-	-	<0.0001
Thallium	T-Tl	-	-	-	<0.0001
Tin	T-Sn	-	-	-	0.0002
Titanium	T-Ti	-	-	-	<0.01
Uranium	T-U	-	-	-	<0.00001
Vanadium	T-V	-	-	-	<0.001
Zinc	T-Zn	-	-	-	<0.001

Results are expressed as milligrams per litre except where noted.
< = Less than the detection limit indicated.

File No. T3356

RESULTS OF ANALYSIS - Water

Sample ID		JM-6	JM-7	JM-8
Sample Date		03 08 25	03 08 25	03 08 25
ALS ID		*11*	*12*	*13*
Dissolved Metals				
Aluminum	D-Al	0.044	0.047	0.048
Antimony	D-Sb	<0.0001	0.0001	0.0001
Arsenic	D-As	0.0001	0.0001	0.0001
Barium	D-Ba	0.0236	0.0332	0.0333
Beryllium	D-Be	<0.0005	<0.0005	<0.0005
Bismuth	D-Bi	<0.0005	<0.0005	<0.0005
Boron	D-B	<0.01	<0.01	<0.01
Cadmium	D-Cd	0.00009	0.00007	0.00007
Calcium	D-Ca	7.99	22.2	22.5
Chromium	D-Cr	<0.0005	<0.0005	<0.0005
Cobalt	D-Co	<0.0001	<0.0001	<0.0001
Copper	D-Cu	0.0004	0.0028	0.0030
Iron	D-Fe	<0.03	0.03	0.03
Lead	D-Pb	0.00016	0.00019	0.00019
Lithium	D-Li	<0.005	<0.005	<0.005
Magnesium	D-Mg	0.6	2.2	2.2
Manganese	D-Mn	0.0149	0.0615	0.0611
Molybdenum	D-Mo	0.00017	0.00040	0.00041
Nickel	D-Ni	<0.0005	<0.0005	<0.0005
Phosphorus	D-P	<0.3	<0.3	<0.3
Potassium	D-K	<2	<2	<2
Selenium	D-Se	<0.001	<0.001	<0.001
Silicon	D-Si	0.57	1.12	1.15
Silver	D-Ag	<0.00001	<0.00001	<0.00001
Sodium	D-Na	<2	<2	<2
Strontium	D-Sr	0.0544	0.182	0.178
Thallium	D-Tl	<0.0001	<0.0001	<0.0001
Tin	D-Sn	<0.0001	<0.0001	<0.0001
Titanium	D-Ti	<0.01	<0.01	<0.01
Uranium	D-U	0.00008	0.00029	0.00029
Vanadium	D-V	<0.001	<0.001	<0.001
Zinc	D-Zn	0.003	0.003	0.002

Results are expressed as milligrams per litre except where noted.
< = Less than the detection limit indicated.

File No. T3356

Appendix 1 - QUALITY CONTROL - Replicates

Water		U-5	U-5
		03 08 24	QC # 352078
Physical Tests			
Hardness	CaCO3	210	211
pH		8.22	8.23
Dissolved Anions			
Alkalinity-Total	CaCO3	144	147
Sulphate	SO4	83	80
Total Metals			
Calcium	T-Ca	60.1	63.4
Magnesium	T-Mg	10.2	10.8

Results are expressed as milligrams per litre except where noted.
< = Less than the detection limit indicated.

File No. T3356

Appendix 1 - QUALITY CONTROL - Replicates

Water		U-5	U-5
		03 08 24	QC # 352078
Dissolved Metals			
Aluminum	D-Al	0.011	0.011
Antimony	D-Sb	0.0004	0.0004
Arsenic	D-As	0.0002	0.0002
Barium	D-Ba	0.0192	0.0191
Beryllium	D-Be	<0.0005	<0.0005
Bismuth	D-Bi	<0.0005	<0.0005
Boron	D-B	<0.01	<0.01
Cadmium	D-Cd	0.00020	0.00019
Calcium	D-Ca	65.0	65.1
Chromium	D-Cr	<0.0005	<0.0005
Cobalt	D-Co	0.0003	0.0004
Copper	D-Cu	0.0063	0.0061
Iron	D-Fe	<0.03	<0.03
Lead	D-Pb	0.00006	<0.00005
Lithium	D-Li	<0.005	<0.005
Magnesium	D-Mg	11.6	11.7
Manganese	D-Mn	0.194	0.189
Molybdenum	D-Mo	0.00044	0.00044
Nickel	D-Ni	<0.0005	<0.0005
Phosphorus	D-P	<0.3	<0.3
Potassium	D-K	<2	3
Selenium	D-Se	<0.001	<0.001
Silicon	D-Si	1.76	1.77
Silver	D-Ag	<0.00001	<0.00001
Sodium	D-Na	<2	<2
Strontium	D-Sr	0.583	0.572
Thallium	D-Tl	<0.0001	<0.0001
Tin	D-Sn	<0.0001	<0.0001
Titanium	D-Ti	<0.01	<0.01
Uranium	D-U	0.00153	0.00151
Vanadium	D-V	<0.001	<0.001
Zinc	D-Zn	0.032	0.032

Results are expressed as milligrams per litre except where noted.
< = Less than the detection limit indicated.

Appendix 2 - METHODOLOGY

Outlines of the methodologies utilized for the analysis of the samples submitted are as follows

Conventional Parameters in Water

These analyses are carried out in accordance with procedures described in "Methods for Chemical Analysis of Water and Wastes" (USEPA), "Manual for the Chemical Analysis of Water, Wastewaters, Sediments and Biological Tissues" (BCMOE), and/or "Standard Methods for the Examination of Water and Wastewater" (APHA). Further details are available on request.

pH in Water

This analysis is carried out using procedures adapted from APHA Method 4500-H "pH Value". The pH is determined in the laboratory using a pH electrode.

Recommended Holding Time:
Sample: 2 hours
Reference: APHA
For more detail see ALS Environmental "Collection & Sampling Guide"

Alkalinity in Water by Colourimetry

This analysis is carried out using procedures adapted from EPA Method 310.2 "Alkalinity". Total Alkalinity is determined using the methyl orange colourimetric method.

Recommended Holding Time:
Sample: 14 days
Reference: APHA
For more detail see ALS Environmental "Collection & Sampling Guide"

Sulphate in Water

This analysis is carried out using procedures adapted from APHA Method 4500-SO4 "Sulphate". Sulphate is determined using the turbidimetric method.

Recommended Holding Time:
Sample: 28 days
Reference: APHA
For more detail see ALS Environmental "Collection & Sampling Guide"

Metals in Water

This analysis is carried out using procedures adapted from "Standard Methods for the Examination of Water and Wastewater" 20th Edition 1998 published by the American Public Health Association, and with procedures adapted from "Test Methods for Evaluating Solid Waste" SW-846 published by the United States Environmental Protection Agency (EPA). The procedures may involve preliminary sample treatment by acid digestion, using either hotplate or microwave oven, or filtration (EPA Method 3005A). Instrumental analysis is by atomic absorption/emission spectrophotometry (EPA Method 7000 series), inductively coupled plasma - optical emission spectrophotometry (EPA Method 6010B), and/or inductively coupled plasma - mass spectrometry (EPA Method 6020).

Recommended Holding Time:

Sample:	6 months
Reference:	EPA
For more detail see:	ALS "Collection & Sampling Guide"

Solids in Water

This analysis is carried out using procedures adapted from APHA Method 2540 "Solids". Solids are determined gravimetrically. Total dissolved solids (TDS) and total suspended solids (TSS) are determined by filtering a sample through a glass fibre filter, TDS is determined by evaporating the filtrate to dryness at 180 degrees celsius, TSS is determined by drying the filter at 104 degrees celsius. Total solids are determined by evaporating a sample to dryness at 104 degrees celsius. Fixed and volatile solids are determined by igniting a dried sample residue at 550 degrees celsius.

Recommended Holding Time:
Sample: 7 days
Reference: APHA
For more detail see ALS Environmental "Collection & Sampling Guide"

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End of Report

CHAIN OF CUSTODY / ANALYTICAL REQUEST FORM

CLIENT: SKYLINE GOLD CORPORATION
ADDRESS: 910 - 925 West Georgia Street
CITY: Vancouver PROV.: B.C. POSTAL CODE: V6C 3L2
CONTACT: David Yeager SAMPLER: David Yeager
TELEPHONE: 604-683-6049 FAX: 604-681-5636
PROJECT NAME/NO.: Johnny Mountain
P.O. NO.: DAY-ALS-1 QUOTE NO.:
DATE SUBMITTED: Aug. 27/2003 ALS CONTACT: Fredrick Chen

ALS Environmental

1988 Triumph Street
Vancouver, BC Canada V5L 1K5
TEL: 604-253-4188, 1-800-665-0243
FAX: 604-253-6700

#2 - 21 Highfield Circle SE
Calgary, AB Canada T2G 5N6
TEL: 403-214-5431, 1-866-722-6231
FAX: 403-214-5430

www.alsenviro.com

ANALYSIS REQUESTED

LAB USE ONLY: T3356

Sample Identification	Y	M	D	Time	Matrix	pH	SO4	Alkalinity	Total Hardness	Total Suspended Solids		Dissolved Metals 30 element ICPMS include Iron					Notes
U-1	03	08	24	: AM PM	Water	✓	✓	✓	✓			✓					
U-2	03	08	24	: AM PM	"	✓	✓	✓	✓			✓					
U-3	03	08	24	: AM PM	"	✓	✓	✓	✓			✓					
U-4	03	08	24	: AM PM	"	✓	✓	✓	✓			✓					
U-5	03	08	24	: AM PM	"	✓	✓	✓	✓			✓					
U-6	03	08	24	: AM PM	"	✓	✓	✓	✓			✓					
JM-1	03	08	25	: AM PM	"	✓	✓	✓	✓			✓					
JM-2	03	08	25	: AM PM	"	✓	✓	✓	✓			✓					
JM-3	03	08	25	: AM PM	"	✓	✓	✓	✓			✓					
JM-4	03	08	25	: AM PM	"	✓	✓	✓	✓			✓					
JM-6	03	08	25	: AM PM	"	✓	✓	✓	✓	✓		✓					
JM-7	03	08	25	: AM PM	"	✓	✓	✓	✓	✓		✓					
JM-8	03	08	25	: AM PM	"	✓	✓	✓	✓			✓					
Travel Blank JM-9	03	08	15	: AM PM	"	✓	✓	✓	✓			✓					
				: AM PM													
				: AM PM													

FOR LAB USE ONLY

TURN AROUND REQUIRED:
☑ ROUTINE (7 - 10 WORKING DAYS)
☐ RUSH (SPECIFY DATE): ____________ (SURCHARGES MAY APPLY)
SPECIAL INSTRUCTIONS (BILLING DETAILS, QC REPORTING, ETC.):

RELINQUISHED BY:	DATE / TIME	RECEIVED BY:	DATE / TIME
DAY	DATE: TIME:	MAR	DATE: 28/08/03 TIME: 10:25
	DATE: TIME:		DATE: TIME:

FOR LAB USE ONLY

COOLER SEAL INTACT UPON RECEIPT? ☐ YES ☐ NO ☐ N/A

SAMPLE TEMPERATURE UPON RECEIPT: ____ °C
FROZEN? ☐ YES ☐ NO

REPORT COPY

ALS Chemex
EXCELLENCE IN ANALYTICAL CHEMISTRY
ALS Canada Ltd.
212 Brooksbank Avenue
North Vancouver BC V7J 2C1 Canada
Phone: 604 984 0221 Fax: 604 984 0218

SKYLINE GOLD CORPORATION
910 - 925 W. GEORGIA ST.
VANCOUVER BC V6C 3L2

Date : 1-Oct-2003
Account: BQL

CERTIFICATE VA03033021

Project : Johnny Mtn.

P.O. No: DAY-CHE-01

This report is for 26 samples submitted to our lab in North Vancouver, BC, Canada on 10-Sep-2003.

The following have access to data associated with this certificate:

DAVID YEAGER		

SAMPLE PREPARATION

ALS CODE	DESCRIPTION
WEI-21	Received Sample Weight
LOG-22	Sample login - Rcd w/o BarCode
PUL-31	Pulverize split to 85% <75 um

ANALYTICAL PROCEDURES

ALS CODE	DESCRIPTION	INSTRUMENT
Au-AA23	Au 30g FA-AA finish	AAS
Au-GRA21	Au 30g FA-GRAV finish	WST-SIM

To: **SKYLINE GOLD CORPORATION**
ATTN: DAVID YEAGER
910 - 925 W. GEORGIA ST.
VANCOUVER BC V6C 3L2

This is the Final Report and supersedes any preliminary report with this certificate number. Results apply to samples as submitted. All pages of this report have been checked and approved for release.

Signature:

ALS Chemex

EXCELLENCE IN ANALYTICAL CHEMISTRY

ALS Canada Ltd.
212 Brooksbank Avenue
North Vancouver BC V7J 2C1 Canada
Phone: 604 984 0221 Fax: 604 984 0218

SKYLINE GOLD CORPORATION
910 - 925 W. GEORGIA ST.
VANCOUVER BC V6C 3L2

Date : 1-Oct-2003
Account: BQL

Project : Johnny Mtn.

CERTIFICATE OF ANALYSIS VA03033021

Sample Description	Method / Analyte / Units / LOR	WEI-21 Recvd Wt kg 0.02	Au-AA23 Au ppm 0.005	Au-GRA21 Au ppm 0.05
40N 160E-A		0.36	1.770	
40N 160E-B		0.36	2.37	
40N 180E-A		0.32	5.48	
40N 180E-B		0.36	2.43	
40N 200E-A		0.28	5.95	
40N 200E-B		0.34	5.15	
40N 220E-A		0.34	2.86	
40N 220E-B		0.32	3.55	
40N 240E-A		0.36	1.870	
40N 240E-B		0.36	2.69	
40N 260E-A		0.34	1.135	
40N 260E-B		0.40	1.915	
40N 280E-A		0.32	2.44	
40N 280E-B		0.42	1.970	
10N 200E-A		0.32	>10.0	13.65
10N 200E-B		0.36	>10.0	10.85
60N 200E-A		0.42	1.550	
60N 200E-B		0.34	3.15	
60N 220E-A		0.32	1.160	
60N 220E-B		0.36	0.949	
60N 240E-A		0.32	1.510	
60N 240E-B		0.36	1.605	
60N 260E-A		0.30	1.110	
60N 260E-B		0.36	1.470	
60N 280E-A		0.36	0.761	
60N 280E-B		0.34	0.965	

BC RESEARCH Inc.

BC Research Inc., BC Research and Innovation Complex, 3650 Wesbrook Mall, Vancouver, BC, Canada V6S 2L2
Telephone: 604 224-4331 • Facsimile: 604 224-0540 • Email: info@bcresearch.com • Website: www.bcresearch.com

File No: 2-21-900
September 18, 2003

Mr. David Yeager
Skyline Gold Corporation
Suite 910 Cathedral Place
925 West Georgia St.
Vancouver, BC
Canada V6C 3L2

Dear David:

Subject: **ABA Results & Invoice for Johnny Mountain Samples Rec'd August 29/03**

Attached in Table 1 are the results of acid base accounting of 27 rock samples. Sulphur speciation and carbonate carbon were also determined. QA/QC for the analysis is shown in Tables 2a, 2b and 2c.

The ABA analysis was carried out according to Sobek A. et al., EPA-600/2-78-054, March, 1978. Sulphate sulphur was determined by extracting the sulphate from the sample with 3N hydrochloric acid. Carbonate carbon was determined by treating the sample with hydrochloric acid to evolve the carbon dioxide gas which was collected and analysed with a Leco carbon/sulphur analyser. The results are expressed as percent carbon dioxide.

Cost detail is outlined in Table 3. Our invoice for $2,700.00, plus GST, is enclosed. Thank you for using BCRI.

Sincerely,

Rik Vos
ARD and Extractive Metallurgy Group
Process and Analysis Division

Table 3
Cost Detail

Task	Unit Price	Number	Total Price
Sample Preparation	$7	27	$189.00
Sobek ABA	$60	27	$1,620.00
SO_4-S	$20	27	$540.00
CO_2	$13	27	$351.00
			Grand Total = $2,700.00

BC RESEARCH Inc.

BC Research Inc., BC Research and Innovation Complex, 3650 Wesbrook Mall, Vancouver, BC, Canada V6S 2L2
Telephone: 604 224-4331 • Facsimile: 604 224-0540 • Email: info@bcresearch.com • Website: www.bcresearch.com

Table 1: ABA Results by Standard Sobek Method for Johnny Mountain Samples - August 29, 2003

Sample	Paste pH	CO2 (Wt.%)	CaCO3 Equiv. (Kg CaCO3/Tonne)	Total Sulphur (Wt.%)	Sulphate Sulphur (Wt.%)	Sulphide Sulphur* (Wt.%)	Maximum Potential Acidity** (Kg CaCO3/Tonne)	Neutralization Potential (Kg CaCO3/Tonne)	Net Neutralization Potential (Kg CaCO3/Tonne)	Fizz Rating
5784	7.8	1.41	32.0	1.52	<0.01	1.52	47.5	53.2	5.7	moderate
5785	7.2	1.05	23.9	1.56	<0.01	1.56	48.8	41.8	-7.0	moderate
5786	7.2	0.93	21.1	3.41	0.01	3.40	106.3	49.4	-56.9	moderate
5787	8.1	1.59	36.1	0.62	<0.01	0.62	19.4	46.8	27.4	moderate
5788	8.0	1.61	36.6	0.71	<0.01	0.71	22.2	57.0	34.8	moderate
5789	8.0	1.37	31.1	0.31	<0.01	0.31	9.7	48.1	38.4	moderate
5790	8.1	1.31	29.8	0.23	<0.01	0.23	7.2	46.8	39.6	moderate
5791	8.3	1.45	33.0	0.17	<0.01	0.17	5.3	49.4	44.1	moderate
5792	8.3	1.39	31.6	0.09	<0.01	0.09	2.8	50.6	47.8	moderate
5793	8.2	1.2	27.3	0.09	<0.01	0.09	2.8	46.8	44.0	moderate
5794	8.4	1.48	33.6	0.20	<0.01	0.20	6.3	49.4	43.2	moderate
5795	8.1	0.99	22.5	0.04	<0.01	0.04	1.3	39.2	38.0	moderate
5796	7.4	1.13	25.7	0.07	<0.01	0.07	2.2	26.6	24.4	moderate
5797	7.0	0.23	5.2	0.03	<0.01	0.03	0.9	13.5	12.6	slight
5798	6.3	<0.01	<0.2	0.02	<0.01	0.02	0.6	4.8	4.2	none
5799	8.3	2.3	52.3	2.12	<0.01	2.12	66.3	75.9	9.7	moderate
5800	8.3	2.23	50.7	3.88	<0.01	3.88	121.3	67.1	-54.2	moderate
314451	7.7	0.77	17.5	0.61	<0.01	0.61	19.1	24.8	5.7	slight
314452	6.3	0.23	5.2	0.28	0.01	0.27	8.4	13.3	4.9	slight
314453	6.1	0.46	10.5	1.67	0.02	1.65	51.6	15.5	-36.1	slight
314454	6.3	0.68	15.5	1.92	0.02	1.90	59.4	18.0	-41.4	slight
314455	5.7	0.4	9.1	4.18	0.05	4.13	129.1	13.3	-115.8	slight
314456	5.4	0.16	3.6	0.92	0.02	0.90	28.1	7.0	-21.1	none
314457	5.2	0.32	7.3	4.95	0.08	4.87	152.2	12.5	-139.7	slight
314458	7.1	0.76	17.3	0.52	<0.01	0.52	16.3	22.3	6.1	slight
314459	7.0	0.67	15.2	0.27	<0.01	0.27	8.4	18.8	10.4	slight
314460	4.9	0.39	8.9	3.96	0.19	3.77	117.8	8.5	-109.3	slight

*Based on difference between total sulphur and sulphate-sulphur
**Based on sulphide-sulphur

BC Research Inc., BC Research and Innovation Complex, 3650 Wesbrook Mall, Vancouver, BC, Canada V6S 2L2
Telephone: 604 224-4331 • Facsimile: 604 224-0540 • Email: info@bcresearch.com • Website: www.bcresearch.com

Table 2a: QA/QC for NP Determination (Sobek Method)

Sample	Neutralisation Potential (kgCaCO3/Tonne)	Neutralisation Potential (kgCaCO3/Tonne)
Duplicates - NP		
5788	57.0	57.0
5796	26.6	25.3
314454	18.0	18.8
KZK-1 Reference (NP = 59.0)	59.5	-

Table 2b: QA/QC for Sulphur Speciation

Sample	Sulphur (Wt.%)	Sulphur (Wt.%)
Duplicates - total sulphur		
5791	0.17	0.18
5798	0.02	<0.02
314456	0.92	0.93
Inhouse Std. (0.11%)	0.12	-
Std. CSB (5.3%)	5.36	-
Duplicates - sulphate sulphur		
5789	<0.01	<0.01
5798	<0.01	<0.01
314453	0.02	0.02
BCRI 0.23% SO4-S Ref.	0.25	-

Table 2c: QA/QC for CO2 Determination

Sample	CO2 (Wt.%)	CO2 (Wt.%)
Duplicates - CO2		
5791	1.45	1.42
5798	<0.01	<0.01
314456	0.16	0.17
Std. CSB (1.50% CO2)	1.49	-