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Elsa Tailings Reprocessing Assessment

2008 Closure Studies

Prepared by:

Elsa Reclamation Development Company Ltd.

June 2009

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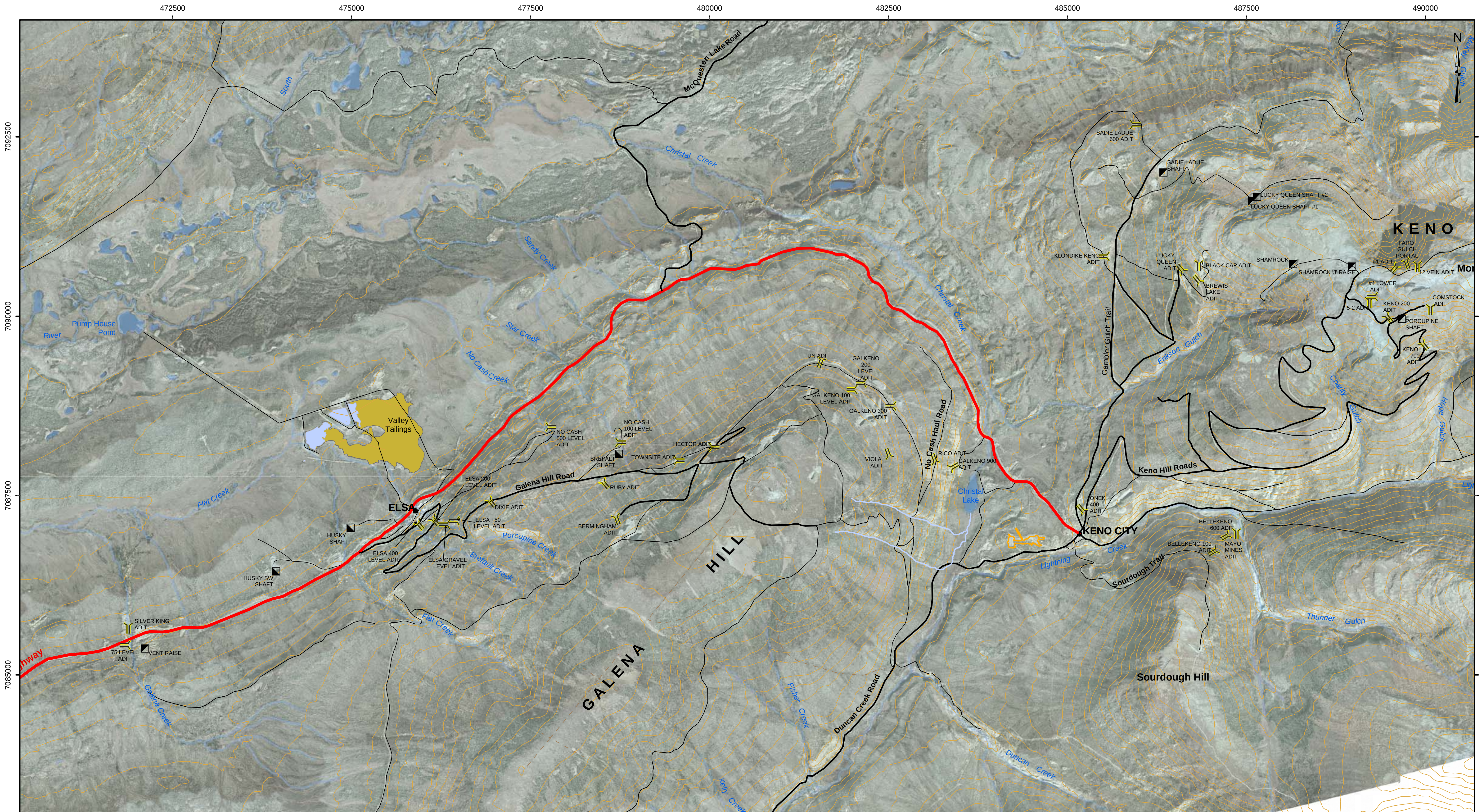
1. PROJECT DESCRIPTION/HISTORY

Between 1920 and 1989, approximately 4.6 million tons of tailings were deposited in the Elsa Tailings Facility, downslope in the upper Flat Creek valley and encompass an area of over 130 ha. The thickness of the tailings ranges from 0.1 m– 0.2 m to over 4 m. The location of the Elsa Tailings Facility is shown in Figure 1. The tailings facility represents the largest single liability in the closure of the Keno Hill Silver District. In 2007, Elsa Reclamation and Development Company Ltd. (ERDC) commissioned SRK Consulting (SRK) to begin assessing various closure options associated with the Elsa Tailings Facility. Under any final closure option selected, consolidation and movement of the tailings is likely involved to varying degrees depending on the final option selected. If the tailings need to be consolidated and re-handled for final closure measures, investigating a reprocessing scenario that can be incorporated into final closure is a logical and prudent step in the closure planning process. Recovery of remaining economic value from the tailings (i.e. silver/gold) offers the potential to significantly offset the final closure costs and result in a reduced footprint for final covering.

The approach to reprocessing silver and gold from the Elsa tailings is not a new concept and there have been a number of assessments completed by previous operators and owners of the district. Previous assessments include considerable drilling, sample collection, and metallurgical testing; all of which is considered credible and reliable support information for this preliminary stage assessment. This report offers a scoping level assessment to the potential economics of reprocessing silver from the Elsa Valley Tailings using conventional reprocessing technologies.



0 500 1,000 2,000 3,000 4,000 Meters

- Adit
- Shaft
- Valley Tailings
- Pit
- Town
- Silver Trail
- Secondary Road
- Limited-use road
- Trail
- Watercourse
- Waterbody

Notes:
This map is for illustrative purposes only.
This is not a legal document.
National Topographic Data Base (NTDB) compiled by Natural Resources Canada at a scale of 1:50,000.
Reproduced under license from Her Majesty the Queen in Right of Canada, permission of Natural Resources Canada.
Projection: UTM Zone 8 NAD83
Scale 1:30,000 when plotted at A0 page size.

ALEXCO

ERDC

KENO HILL SILVER DISTRICT

REGIONAL

VALLEY TAILINGS AREA LOCATION

DRAWN BY: TL	DATE: 27-MAY-2009	FIGURE 1

1.1. PREVIOUS ASSESSMENTS

There is a minimum of four historic reports on the potential for reprocessing of Elsa tailings. A summary of the pertinent findings of each of these reports follows. The full reports are included in Appendix D. Additional reports and testwork are referenced in these reports but copies have not been located.

1.1.1. Internal UKHM1 Office Correspondence – January 1987

- Conceptual report that compiled historic mill records and grades to determine potential metal content in the tailings;
- Based on historic records, estimated ***treatable*** metal content in tailings was 3.2 million tons @ 3.92 opt Ag, 1.17% Pb, 0.79% Zn (12,544,000 ounces Ag);
- No additional sampling, assaying or metallurgical testing was carried out;
- Envisioned an open air gravity pre-concentration and cyanidation recovery process;
- Recommended follow-up sampling and testwork to verify assumptions and potential.

1.1.2. 1987 – 1988 Tailings Drilling Program

- Based on the recommendations from the Jan 1987 report, UKHM initiated an auger drilling program in the tailings;
- Tailings were fully delineated on a 100 ft. x 100 ft. (30 m x 30 m) grid. A total of 5,400 feet of drilling in 379 holes was completed using a Schramm drill;
- Drilling program resulted in total tailings of 4,049,000 tons @ 3.14 opt Ag (12,713,000 ounces Ag). This report also verified the historic mill discharge Ag grade of 3.98 opt (compares favorably to the Jan 1987 report);
- Based on the distribution of tailings in the valley, an initial estimate of selective mining potential indicated 1.0 million tons @ 5.35 opt Ag (5,350,000 ounces Ag);
- No metallurgical testing or economic analysis was completed in this assessment.

1 United Keno Hill Mines Limited

1.1.3. Scoping Testwork on the Recovery of Silver, Lead and Zinc on Tailing Material – December 1995

- First report (found in historic records) summarizing metallurgical testing on tailings material;
- Two grab samples (~50 kg.) were collected and testing included gravity separation and flotation;
- The Ag grades in the samples tested ranged from 2.45 – 3.85 opt with Pb and Zn values both <1%;
- The majority of the lead and zinc is in an oxide form;
- Four separate flotation tests were completed using varying reagent schemes and dosages. The Ag recovery in the flotation tests varied from 23.1% – 33.8%. The poor recovery was attributed to the high oxide forms of the lead and zinc;
- Gravity separation tests included the use of a Falcon concentrator followed by a Knelson concentrator. The overall silver recovery with gravity separation was 4%;
- Results from this cursory metallurgical testing indicates that flotation and gravity separation are not likely process options;
- Given the fact that this testwork was completed on only two samples brings into question the representativeness of the testwork and samples on the overall tailings material.

1.1.4. Investigation into the Reprocessing of Elsa Tailings – Hawthorne March 1996

- Report summarizes historic reports and assessments;
- The report presents contradictory testwork conclusions. As an example, 1995 UKHM testwork on gravity separation returned only a 2% recovery; however, the Hawthorne report discusses the encouraging results from gravity testwork. Additional metallurgical testing is referenced (i.e. UKHM 1988) but copies of this work have not been located;
- Discusses results from Candora Operating Company on favorable heap leach test results (bottle and column tests); however, no copy of the report is available;
- Concludes that approximately 1,700,000 tonnes @ 4.45 opt Ag can be excavated and heap leached;
- Summarizes that at a silver price of US$ 5.50/oz heap leaching is encouraging but there are no supporting capital and operating cost data to support this conclusion.

2. SILVER GRADE CONFIRMATION

Based on Alexco[2]/ERDC experience over the past 3 years in the district, historic information (geological, resource estimation, site conditions) has proved to be reasonably accurate and can be relied upon for estimating current conditions. As part of the valley tailings assessment closure study, 19 test pits were excavated throughout the Valley Tailings in 2007 to collect geochemical information. Representative samples from these test pits were composited and sent to ALS Chemex for full metals analysis. The full suite of analytical results is presented in Table 1. A summary of the depth of tailings encountered in the 19 test pits excavated in 2007 along with the average silver grade for the test pits analysed is presented in Table 2. Not all of the 2007 test pits were assayed. The approach was to select test pits that were within areas identified in previous studies to correlate historic grade distributions with current assay results. Volume and grade are presented in short tons and ounces to accurately compare with historic estimates and units of measurements used at UKHM. Historic records and production at UKHM were based on imperial tons rather than metric tonnes. Likewise grades were based on ounces per ton (opt) rather than grams per tonne (gpt). Going forward, the economic analysis presented in this study is based on metric units for grade and tonnage (tonnes and gpt).

Figure 2 represents an example of one of the numerous test pits that were excavated in 2007 along with the test pit logs completed during the sampling project. Figure 3 presents the location of the test pits that were analysed for silver and other parameters for this study. The area of the outlined perimeter of the tailings shown in Figure 3 is ~752,931 m^2. Using the average depth of tailings sampled in this area (2.1 meters) times an estimated in-situ density of 1.5, the tonnes of tailings potentially available for reprocessing is 2,371,000 tonnes (2,631,520 tons). Table 3 compares the 2007 ERDC sampling results completed for this study against historic silver tailings assessment programs that have been evaluated in the past.

[2] ERDC is a wholly owned subsidiary of Alexco Resource Corp.

Table 1 Full Suite Sample Analysis

CLIENT	**ALERES - Alexco Resource Corp.**
# of SAMPLES	**33**
DATE RECEIVED	**260808**
DATE COMPLETED	**180908**
PROJECT	**Keno Hill**
CERTIFICATE COMMENT	**ALL:NSS is non-sufficient sample.**
PO NUMBER	**1620-720-034**

	WEI-21	**Au-AA25**	**ME-ICP61**	**ME-ICP61**	**ME-ICP61**	**ME-ICP61**	**ME-ICP61**	**ME-ICP61**	**ME-ICP61**	**ME-ICP61**	**ME-ICP61**	**ME-ICP61**
SAMPLE	**Recvd Wt.**	**Au**	**Ag**	**Al**	**As**	**Ba**	**Be**	**Bi**	**Ca**	**Cd**	**Co**	**Cr**
DESCRIPTION	**kg**	**ppm**	**ppm**	**%**	**ppm**	**ppm**	**ppm**	**ppm**	**%**	**ppm**	**ppm**	**ppm**
DETECTION	0.02	0.01	0.5	0.01	5	10	0.5	2	0.01	0.5	1	1
TP-08-03	4.52	0.1	60.5	1.12	765	180	<0.5	<2	0.28	74.1	3	22
TP-08-04	5.99	0.17	>100	1.09	729	210	<0.5	<2	0.34	178	3	15
TP-08-08	4.8	0.14	80.3	1.2	1250	180	<0.5	<2	0.6	115	4	16
TP-08-10	4.54	0.11	95.8	1.63	1045	230	<0.5	<2	0.53	126	4	27
TP-08-10A	0.06	1.52	21.9	7	8	800	0.9	<2	4.35	102	9	21
UKTP03-A	1.92	0.28	>100	0.82	2870	120	<0.5	2	0.37	117	5	11
UKTP03-B	1.77	0.26	>100	1.27	2000	170	<0.5	5	0.36	93.8	4	19
UKTP03-C	1.34	0.12	>100	4.69	287	950	1.1	<2	1.17	493	14	66
UK-TP03-Z	7.53	0.22	>100	1.89	1705	300	0.5	<2	0.56	186	7	31
UK-TP03-ZZ	<0.02	0.23	>100	1.74	1710	280	0.5	3	0.54	176.5	7	26
UKTP07-A	1.71	0.15	>100	1.12	1635	160	<0.5	<2	0.33	130.5	3	19
UKTP07-C	2.32	0.08	59.2	2.99	865	410	0.7	<2	0.79	169	7	48
UKTP07-D	1.15	0.01	13.8	2.58	160	560	0.7	<2	1.31	103	16	40
UKTP07-Z	5.42	0.11	69.2	2.04	909	310	0.5	<2	0.68	160	6	34
UKTP12-A	1.84	0.06	>100	1.17	660	250	<0.5	<2	0.13	72.3	2	25
UKTP12-H	1.54	0.05	>100	1.15	337	320	<0.5	<2	0.41	80.7	1	20
UKTP12-HH	1.05	<0.01	0.7	0.14	7	10	<0.5	<2	18.6	0.5	2	4
UKTP12-N	0.71	NSS	>100	0.99	122	330	<0.5	<2	1.83	12.5	5	18
UKTP12-Z	13.79	0.09	>100	1.97	532	520	0.5	<2	0.46	80.9	4	36
UKTP14-A	1.62	0.21	>100	0.98	1320	490	<0.5	5	0.28	158	5	17
UKTP14-E	2.52	0.3	>100	0.98	1730	310	<0.5	3	0.47	170	9	15
UKTP14-H	0.77	0.05	>100	3.35	385	1210	0.9	<2	1.49	107.5	14	45
UKTP14-Z	5.61	0.12	>100	1.33	1145	530	<0.5	<2	0.55	132.5	4	24
UKTP14-ZZ	0.06	0.1	>100	4.07	981	300	0.7	343	1.81	6.1	7	35
UKTP15-A	1.84	0.14	83.1	1.11	1620	240	<0.5	6	0.72	103.5	4	18
UKTP15-C	1.85	0.18	>100	0.92	1905	130	<0.5	2	0.5	178	2	16
UKTP15-F	1.28	0.02	28.8	3.94	87	810	1	<2	1.71	12.8	9	54
UKTP15-Z	6.62	0.13	81.5	2.23	1475	370	0.6	4	0.86	157.5	6	39
UKTP17-A	1.82	0.18	>100	0.86	2570	140	<0.5	4	0.24	174	6	16
UKTP17-C	2.19	0.14	>100	1.08	954	160	<0.5	4	0.32	137	<1	15
UKTP17-CC	1.06	<0.01	0.8	0.14	<5	10	<0.5	<2	19.95	0.8	1	3
UKTP17-E	1.61	0.22	19	4.86	65	820	1.1	2	1.34	7.7	18	63
UKTP17-Z	11.86	0.29	>100	1.35	1720	240	<0.5	6	0.44	125.5	6	21
Overall Average	3.21	0.19	47.28	1.93	1048.53	365.15	0.75	29.92	1.95	119.45	6.16	26.64

Table 1 Full Suite Sample
CLIENT
# of SAMPLES
DATE RECEIVED
DATE COMPLETED
PROJECT
CERTIFICATE COMMENT
PO NUMBER

SAMPLE DESCRIPTION	ME-ICP61 Cu ppm	ME-ICP61 Fe %	ME-ICP61 Ga ppm	ME-ICP61 K %	ME-ICP61 La ppm	ME-ICP61 Mg %	ME-ICP61 Mn ppm	ME-ICP61 Mo ppm	ME-ICP61 Na %	ME-ICP61 Ni ppm	ME-ICP61 P ppm	ME-ICP61 Pb ppm	ME-ICP61 S %
DETECTION	1	0.01	10	0.01	10	0.01	5	1	0.01	1	10	2	0.01
TP-08-03	104	7.58	<10	0.33	10	0.28	32900	1	0.02	9	250	7240	1.71
TP-08-04	529	13.35	<10	0.31	10	0.39	62000	<1	0.02	7	310		1.22
TP-08-08	116	11.6	<10	0.32	10	0.43	51100	<1	0.02	12	300	8350	3.26
TP-08-10	179	11.05	<10	0.46	10	0.39	47600	1	0.03	11	350		2.41
TP-08-10A	2680	4.56	20	0.66	10	1.34	1235	3	1.59	7	800		1.74
UKTP03-A	121	19.05	<10	0.23	10	0.33	76200	<1	0.01	16	290	5810	6.6
UKTP03-B	191	16.4	<10	0.36	10	0.45	59900	<1	0.02	12	260		4.74
UKTP03-C	117	4.31	10	0.97	30	0.66	10850	1	0.68	54	910	660	1.03
UK-TP03-Z	140	14.45	<10	0.45	10	0.46	52800	<1	0.17	24	410	6670	4.59
UK-TP03-ZZ	137	14.85	<10	0.42	10	0.45	55000	<1	0.15	22	370	6760	4.8
UKTP07-A	114	11.3	<10	0.32	10	0.29	38500	<1	0.02	13	300	6900	2
UKTP07-C	304	9.63	10	0.86	10	0.32	43400	<1	0.07	16	450		2.4
UKTP07-D	108	3.2	10	0.55	10	0.6	24400	2	0.3	48	750	1640	2.16
UKTP07-Z	207	9.01	10	0.57	10	0.32	36400	1	0.07	18	400		2.11
UKTP12-A	244	5.59	<10	0.33	10	0.09	19400	1	0.03	8	320	7820	0.23
UKTP12-H	100	5.83	<10	0.34	10	0.16	30200	1	0.02	5	260	8960	0.45
UKTP12-HH	1	0.15	<10	0.02	<10	10.85	223	<1	<0.01	3	290	30	0.03
UKTP12-N	198	2.57	<10	0.15	<10	0.18	2820	2	0.09	42	1590	460	0.82
UKTP12-Z	173	6.81	10	0.57	10	0.22	27100	1	0.05	13	410	9240	1.15
UKTP14-A	320	12.8	<10	0.27	10	0.26	52800	<1	0.04	15	240		3.05
UKTP14-E	232	13.3	<10	0.3	10	0.36	44400	<1	0.03	26	220		6.63
UKTP14-H	448	4.79	10	0.79	10	0.47	4150	2	0.22	43	950	675	0.58
UKTP14-Z	174	9.88	<10	0.39	10	0.29	38800	<1	0.05	16	310	9060	2.82
UKTP14-ZZ	5190	7.86	10	1.02	10	0.84	8100	567	1	13	1450	668	0.75
UKTP15-A	116	10.1	<10	0.33	10	0.3	40000	1	0.04	15	240	6180	3.13
UKTP15-C	149	11.85	10	0.26	10	0.37	48700	<1	0.03	15	200	9900	3.93
UKTP15-F	106	3.35	10	0.82	20	0.55	3520	1	0.55	28	910	1295	0.74
UKTP15-Z	152	10.1	10	0.6	10	0.4	40400	<1	0.12	19	400	9500	2.77
UKTP17-A	148	13.25	<10	0.25	10	0.26	52300	<1	0.03	17	210		4.78
UKTP17-C	499	14.05	<10	0.31	10	0.34	59500	<1	0.04	8	310		1.7
UKTP17-CC	5	0.15	<10	0.03	<10	11.1	265	<1	0.01	1	270	62	0.03
UKTP17-E	53	2.94	10	1.07	20	0.67	7150	<1	0.76	38	830	485	1.22
UKTP17-Z	276	12.15	<10	0.36	10	0.36	46700	<1	0.1	17	390		3.57
Overall Average	413.06	9.03	10.83	0.46	11.33	1.05	33903.42	41.79	0.20	18.52	483.33	4925.68	2.40

Table 1 Full Suite Sample
CLIENT
# of SAMPLES
DATE RECEIVED
DATE COMPLETED
PROJECT
CERTIFICATE COMMENT
PO NUMBER

	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	Pb-OG62	Ag-OG62	Zn-OG62
SAMPLE	**Sb**	**Sc**	**Sr**	**Th**	**Ti**	**Tl**	**U**	**V**	**W**	**Zn**	**Pb**	**Ag**	**Zn**
DESCRIPTION	**ppm**	**ppm**	**ppm**	**ppm**	**%**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**%**	**ppm**	**%**
DETECTION	5	1	1	20	0.01	10	10	1	10	2	0.01	0.5	0.01
TP-08-03	210	2	13	<20	0.07	10	<10	22	<10	5140		60.5	5140
TP-08-04	603	3	19	<20	0.1	20	<10	27	<10	8240	2.72	609	8240
TP-08-08	265	3	20	<20	0.06	20	<10	23	<10	7070		80.3	7070
TP-08-10	350	4	22	<20	0.1	20	<10	33	<10	7540	1.14	95.8	7540
TP-08-10A	32	11	504	<20	0.24	<10	<10	102	<10	>10000	1.24	21.9	20700
UKTP03-A	265	3	11	<20	0.06	20	<10	18	<10	8590		145	8590
UKTP03-B	313	3	19	<20	0.08	20	<10	25	<10	6320	1.26	155	6320
UKTP03-C	31	10	132	<20	0.31	10	<10	89	<10	5500		126	5500
UK-TP03-Z	220	5	42	<20	0.12	20	<10	39	<10	6830		148	6830
UK-TP03-ZZ	226	4	38	<20	0.11	20	<10	35	<10	6970		143	6970
UKTP07-A	190	3	14	<20	0.06	10	<10	23	<10	4570		104	4570
UKTP07-C	535	6	40	<20	0.14	10	<10	57	<10	8210	1.71	59.2	8210
UKTP07-D	47	6	72	<20	0.16	10	<10	56	<10	7150		13.8	7150
UKTP07-Z	322	4	31	<20	0.11	10	<10	41	<10	6470	1.13	69.2	6470
UKTP12-A	358	3	28	<20	0.08	10	<10	25	<10	2290		154	2290
UKTP12-H	277	3	27	<20	0.09	10	<10	26	<10	3690		114	3690
UKTP12-HH	<5	1	106	<20	0.01	<10	<10	3	<10	20		0.7	20
UKTP12-N	14	3	74	<20	0.04	<10	<10	22	<10	579		109	579
UKTP12-Z	310	4	36	<20	0.11	10	<10	41	<10	4250		116	4250
UKTP14-A	336	3	23	<20	0.09	10	<10	23	<10	9810	1.36	258	9810
UKTP14-E	303	3	19	<20	0.07	<10	<10	22	<10	>10000	1.39	282	12800
UKTP14-H	22	8	93	<20	0.15	<10	<10	73	<10	4920		194	4920
UKTP14-Z	270	4	27	<20	0.09	10	10	29	<10	8010		153	8010
UKTP14-ZZ	1830	10	175	<20	0.15	<10	10	69	<10	465		402	465
UKTP15-A	203	3	17	<20	0.06	<10	10	23	<10	7250		83.1	7250
UKTP15-C	303	2	14	<20	0.05	<10	<10	17	<10	>10000		140	12900
UKTP15-F	44	9	132	<20	0.24	<10	<10	79	<10	855		28.8	855
UKTP15-Z	299	5	42	<20	0.11	<10	<10	44	<10	9630		81.5	9630
UKTP17-A	276	2	12	<20	0.05	<10	10	17	<10	>10000	1.15	131	12000
UKTP17-C	660	3	14	<20	0.09	10	10	26	<10	6740	3.45	559	6740
UKTP17-CC	<5	<1	107	<20	0.01	<10	<10	2	<10	31		0.8	31
UKTP17-E	19	10	135	<20	0.3	<10	<10	89	<10	4080		19	4080
UKTP17-Z	420	4	29	<20	0.09	<10	10	30	<10	8470	1.74	299	8470
Overall Average	308.16	4.59	63.24	<20	0.11	13.68	10.00	37.88	<10	5506.55	1.66	150.17	6608.79
												4.38	**opt**

TABLE 2. 2007 VALLEY TAILINGS TEST PIT AVERAGE DEPTHS AND GRADE

TEST PIT ID	TEST PIT DEPTH (M)	TEST PIT Ag GRADE (GPT)	TEST PIT Ag GRADE (OPT)	HISTORIC GRADE RANGES WITHIN TEST PIT LOCATION
TP-01-07	4.80			
TP-02-07	0.90			
TP-03-07	0.30	143.4	4.18	2-4
TP-04-07	2.30			
TP-05-07	2.70			
TP-06-07	1.40			
TP-07-07	0.75	61.55	1.80	2-4
TP-08-07	1.30	173.5	5.06	0-2
TP-10-07	2.00			
TP-11-07	3.00			
TP-12-07	4.00	98.74	2.88	2-4
TP-13-07	1.60			
TP-14-07	0.35	257.8	7.52	2-4
TP-15-07	1.80	83.35	2.43	+4
TP-16-07	1.40			
TP-17-07	0.80	201.76	5.89	+4
TP-18-07	2.07			
TP-19-07	4.70			
Simple Average			4.25	Opt
Hole Depth Weighted Average			3.49	Opt

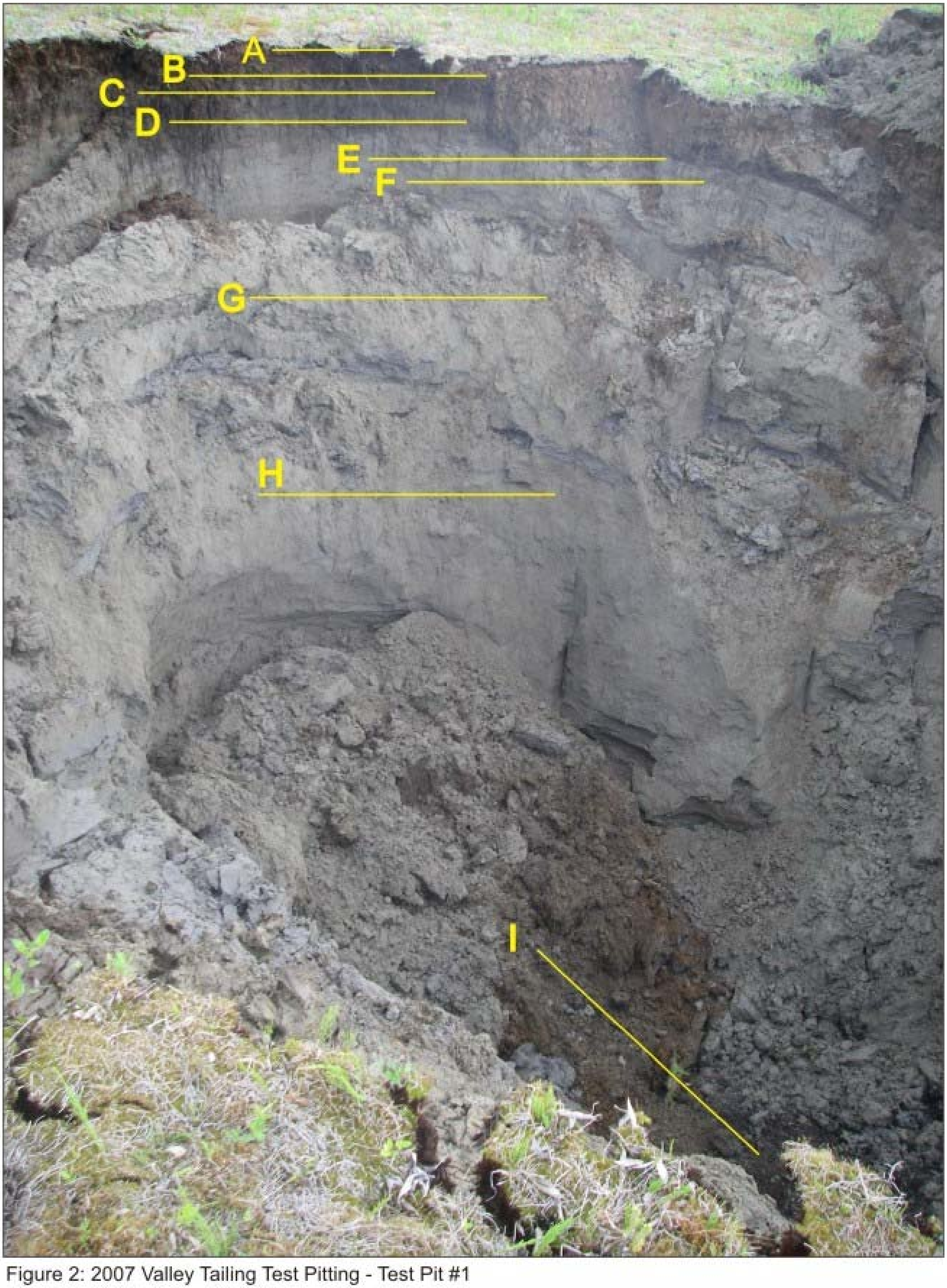
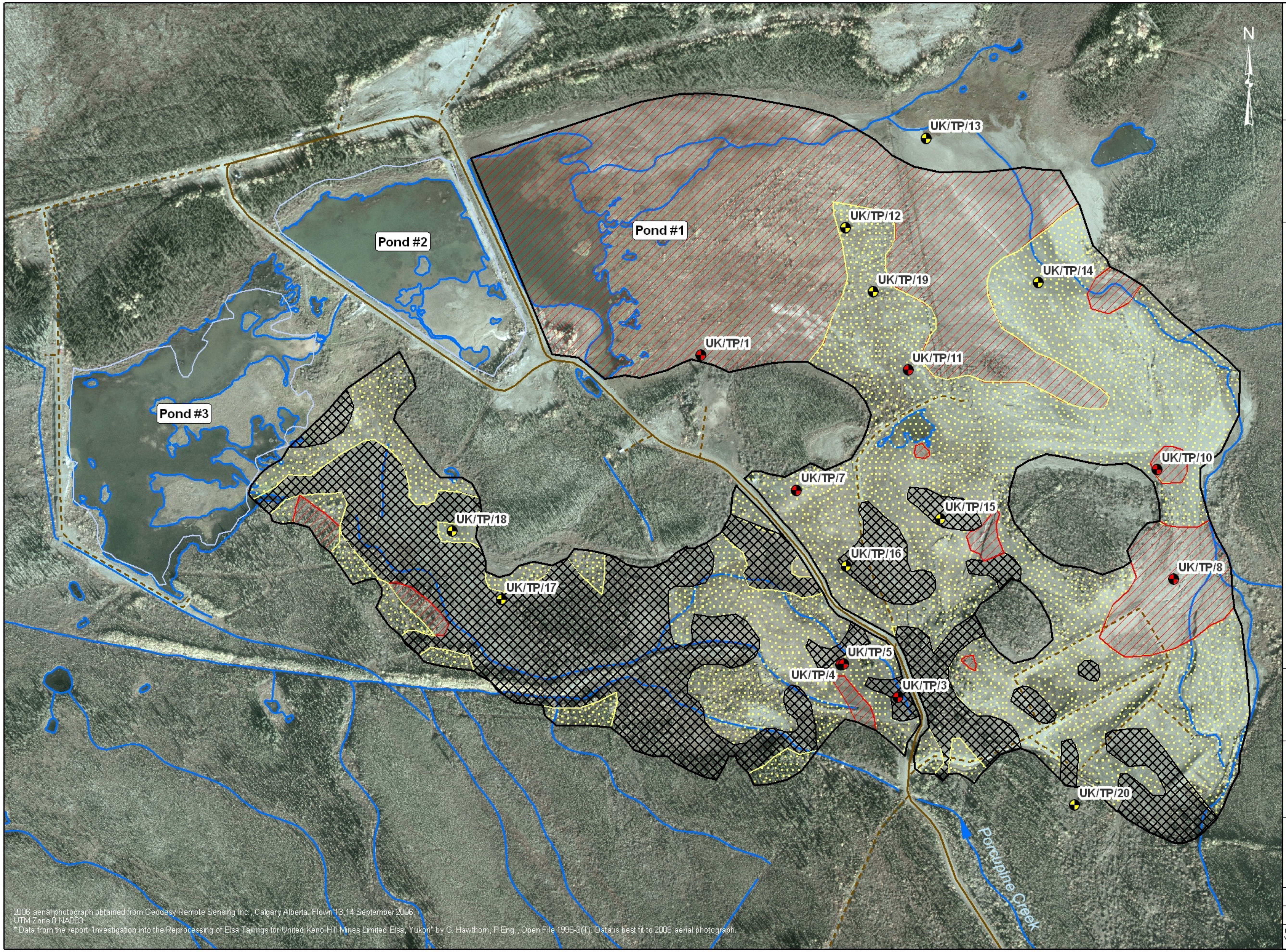


Figure 2: 2007 Valley Tailing Test Pitting - Test Pit #1

Keno Hill Silver District

ERDC



Legend

Public Road
Exploration roads
Trails
Cutlines
Contour
Surface water (actual and potential)
Pond*
Tailings Outline*

Valley Tailings Grade (oz. Ag/ton)*

2-4
0-2
+4

Test Pits

Est 1995
Est 2007

Figure 3

Valley Tailings Site - Grades

Scale:

1:5,000
(when plotted on 11x17 inch sheet)

0 25 50 100 150 200 250 m

ACCESS Consulting Group

Drawn by: HD	Checked by: BT

Date: June 2008

Our File: Dawson\D\Project\AllProjects\ALEX-05-01\gis\mxd\UKHM\ValleyTailings\TailingsGrade\ValleyTailingsGrades.mxd

TABLE 3. HISTORIC TAILINGS ASSESSMENT COMPARISON

STUDY DATE	TONS TAILINGS	SILVER OPT	POTENTIAL SILVER OZ.
1970	2,156,175	1.91	4,118,294
Jan 1987	3,200,000	3.92	12,544,000
1988	4,049,000	3.14	12,713,860
1988	1,000,000	5.35	5,350,000
1996	1,700,000	4.45	7,565,000
Historic Production Records	4,049,670	3.98	16,117,687
2007/08	2,631,520	4.25	11,183,960

Note that in all of the tailings reprocessing studies completed in the past, the estimated volume of tailings for reprocessing is significantly less than what was deposited from historic production records. This is due to many of the areas of the tailings facility having a very thin layer of tailings that would not be efficient or possible to retrieve and rehandle under a reprocessing program. This same problem will be an issue in a stand alone closure option of reconsolidation and covering.

The tonnage estimate presented in the current 2007/08 estimate is generally consistent with previous assessments and estimates made by UKHM. There are obvious variances in both grade and tonnage across the various historic studies and assessments, but the results are within expected variance given the differences in sampling and estimating approaches. These variances are primarily attributed to differences in study design and approach (i.e. grid drilling vs. test pits vs. historic records) and not necessarily reflective of a high level of uncertainty in the amount of silver available for reprocessing and recovery.

In addition to the silver grade of the historic tailings, an appreciable amount of manganese is present, with an approximate overall grade of 3.4%. Current manganese prices are within the $US 2,200/ton range. Although not within the scope of this assessment, the opportunity to recover manganese should be investigated in future studies.

The assayed tailings returned an average gold grade of 0.19 gpt. Although low grade in nature, gold would be recovered in a potential cyanide process and will enhance the overall project economics. Since the gold will be recovered in a cyanide process similar to silver, for the purposes of the economic analysis, a gold grade of 0.15 gpt has been assumed (discounted 20% from average test pit grades) and a conservative recovery of 50% assumed which is lower than the bottle roll tests for silver.

3. PROCESSING OPTIONS

There are a number of processing options to consider for reprocessing the Elsa tailings. Each of these options has inherent pros and cons associated with the particular technology and approach. Although some of these options can be eliminated without further consideration, Table 4 presents the various technologies that could reasonably be investigated for further consideration as a reprocessing technology. A detailed assessment including testwork on all of these options would have to be completed in order to select a final preferred option.

There are other technologies that have not been included in this initial assessment as shown in Table 4. For example, thiosulfate leaching the tailings in a vat approach is a process that on paper would be technically possible but from a practical standpoint is not currently being considered due to the increased risk and technical research and development that would be required to pursue a process that is not “off the shelf”. This process flowsheet (thiosulfate leaching in a vat) was attempted on a silver tailings reprocessing project (Baronne Project, Mexico) in 2004/05 without commercial success.

Pre-concentration of the tailings followed by cyanidation is another example of a process approach that could be considered during an optimization step. At this stage of the assessment, cyanidation of the tailings using conventional heap leach technology is used as the basis for reprocessing. This does not suggest that cyanidation is the current preferred option but it is the most well know and commercially established process for recovery of gold and silver and is therefore used as the basis for this assessment. There are a number of variations to this process that will require further testwork and assessment. Cyanidation of the Elsa tailings was a practice used historically in the Elsa mill to recover oxide silver.

Vat leaching should be one of the primary process options to consider for future studies and assessments. An example of a tailings vat leach system is shown in Figure 4. There are a number of advantages to a vat leach approach over heap leaching tailings, including faster leaching kinetics, reduced solution volumes, reduced reagent consumption, reduced upsets to seasonal operating conditions, smaller process footprint, etc. Not enough information is presently available to present vat leaching as the basis for the economic assessment but vat leaching should remain as a high priority candidate for a reprocessing technology and approach.

Figure 5 shows a typical CIP/CIL circuit that is being used to recover silver. Figure 6 shows heap leaching at the Brewery Creek Mine near Dawson City, Yukon. Heap leaching was used successfully at Brewery Creek on a year round basis and has demonstrated that this is a proven and viable technology in similar operating and climate conditions.

Overall, heap leaching, vat leaching and CIP/CIL are likely the process options that warrant ongoing consideration and development.

TABLE 4. TAILINGS REPROCESSING OPTIONS

PROCESS OPTION	DESCRIPTION	PROS	CONS	DATA DEFICIENCY
Cyanide Heap Leaching	Excavate tailings using scrapers and/or excavator and trucks and deposit onto liner system. Silver recovered from cyanide irrigation on heap and recovered in Merrill Crowe recovery plant. Tailings will require lime/cement agglomeration and/or mixing with competent rock (mineralized waste rock) to provide heap stability and proper percolation	Proven technology. Successfully used in Yukon (Brewery Creek). Generally lower capital and operating costs. Compatible with final closure options and could result in consolidated tailings over a lined system.	Likely to require agglomeration or mixing with coarse waste rock to provide suitable percolation rates. Shorter process season Shutdown in winter. Requires larger holding ponds for solution management.	Recovery testwork. Heap Geotechnical.
Cyanide Vat Leaching	Similar to heap leaching but tailings are deposited into concrete/lined vats and then cyanide solution is percolated through the tailings and recovered in Merrill Crowe recovery plant	Proven technology. Compatible with final closure options. Smaller holding ponds for solution management. Easier to start and stop seasonally. Faster leaching times than heap leaching. Reduced reagent consumption.	Requires additional material rehandling to unload vats. More complicated than heap leaching due to vat rotation timing and limitations on leaching time available. Each vat would require neutralization prior to unloading. Shorter process season shutdown in winter.	Recovery testwork. Recovery cycle times. Vat design.
CIP/CIL Cyanidation	Agitated tanks are used to suspend and agitate a tailings slurry, cyanide solution is added to the tanks and the gold and silver leached within the tanks. In some instances, carbon is added in the leach tanks at the same time leaching occurs (CIL) and the carbon is “stripped” to remove the gold and silver, electrowinning then recovers the precious metals and a dore product is produced.	Proven technology. Smaller volumes of solution management. Easy to start and stop seasonally. Fast leaching times.	Higher capital due to larger tanks, filters. Requires filtering pulp to produce dry stackable tails as end product.	Recovery testwork.

Table 4 (cont'd)

PROCESS OPTION	DESCRIPTION	PROS	CONS	DATA DEFICIENCY
Flotation	Similar process used historically as well as future operations. Silver in sulphide forms is floated and recovered for sale and transportation as a concentrate	Proven technology. Possible to operate year round.	Silver in oxide forms will result in lower recoveries due to poor flotation. Low throughput (<1,000 tpd) will result in long extended project life and reduced economics.	Recovery testwork
Flotation/cyanidation	Similar as flotation above with additional step of cyanidation of the flotation produce followed by Merrill Crowe recovery and dore product. UKHM operated similar circuit in the past	Proven technology. Improved recoveries. Possible to operate year round.	Silver in oxide forms will result in lower recoveries due to poor flotation. Low throughput (<1,000 tpd) will result in long extended project life and reduced economics.	Recovery testwork
Gravity Separation	Gravity recovery would consist of rehandling the tailings in the same manner (i.e. scrapers) repulping and then processing through a Falcon and/or Knelson concentrator approach	Proven technology. Possible process synergies with future milling operations. Possible to operate year round.	Only recovers gravity silver but does not recover leachable silver associated with oxides.	Recovery testwork

FIGURE 4. EXAMPLE OF TAILINGS VAT LEACH SYSTEM

FIGURE 5. EXAMPLE OF CIP/CIL SILVER RECOVERY CIRCUIT

FIGURE 6. EXAMPLE OF HEAP LEACH (BREWERY CREEK MINE, YUKON)

3.1. Metallurgical Testing

Based on a review of the historic testwork completed on previous Elsa tailings reprocessing studies, the possible processing options presented and direct operating experience, metallurgical testwork was completed using standard cyanidation bottle roll techniques on the samples that were collected by ERDC in 2007 (Table 2). Bottle roll tests are a standard approach to determine overall recovery of silver and gold using cyanide as a lixiviant. Bottle roll tests are not sufficient to prepare final detailed designs and operating parameters but they do provide a good indication of potential recovery levels that can be used for scoping purposes.

Six samples from the 2007 test pits were sent to Process Research Associates (Vancouver, BC) for standard cyanide bottle roll tests. Table 5 presents the results of the bottle roll tests. Appendix A presents the detailed test results for the bottle roll tests.

Table 5. Metallurgical Testing Summary

Sample ID	Head Grade (gpt)	Ag Recovery %
C1	142.8	72.7
C2	93.0	58.7
C3	118.0	59.1
C4	246.9	41.6
C5	88.3	42.8
C6	321.3	67.6
Average	168.3	57.1

The average recovery of the six samples tested was 57%. There does not appear to be any relationship between recovery and grade that would lead to a possible conclusion that the lower recoveries are associated with higher grade sulphides.

3.2. PROCESS DESCRIPTION

For the purpose of the tailings reprocessing assessment, a conventional cyanidation heap leach process has been used as the basis for determining order of magnitude economics. It is important to note that a final selection on the preferred processing option can only be made after more detailed assessment and engineering. A conventional heap leach approach is used for determining potential economics and viability of reprocessing tailings and a detailed engineering process trade off study would be required.

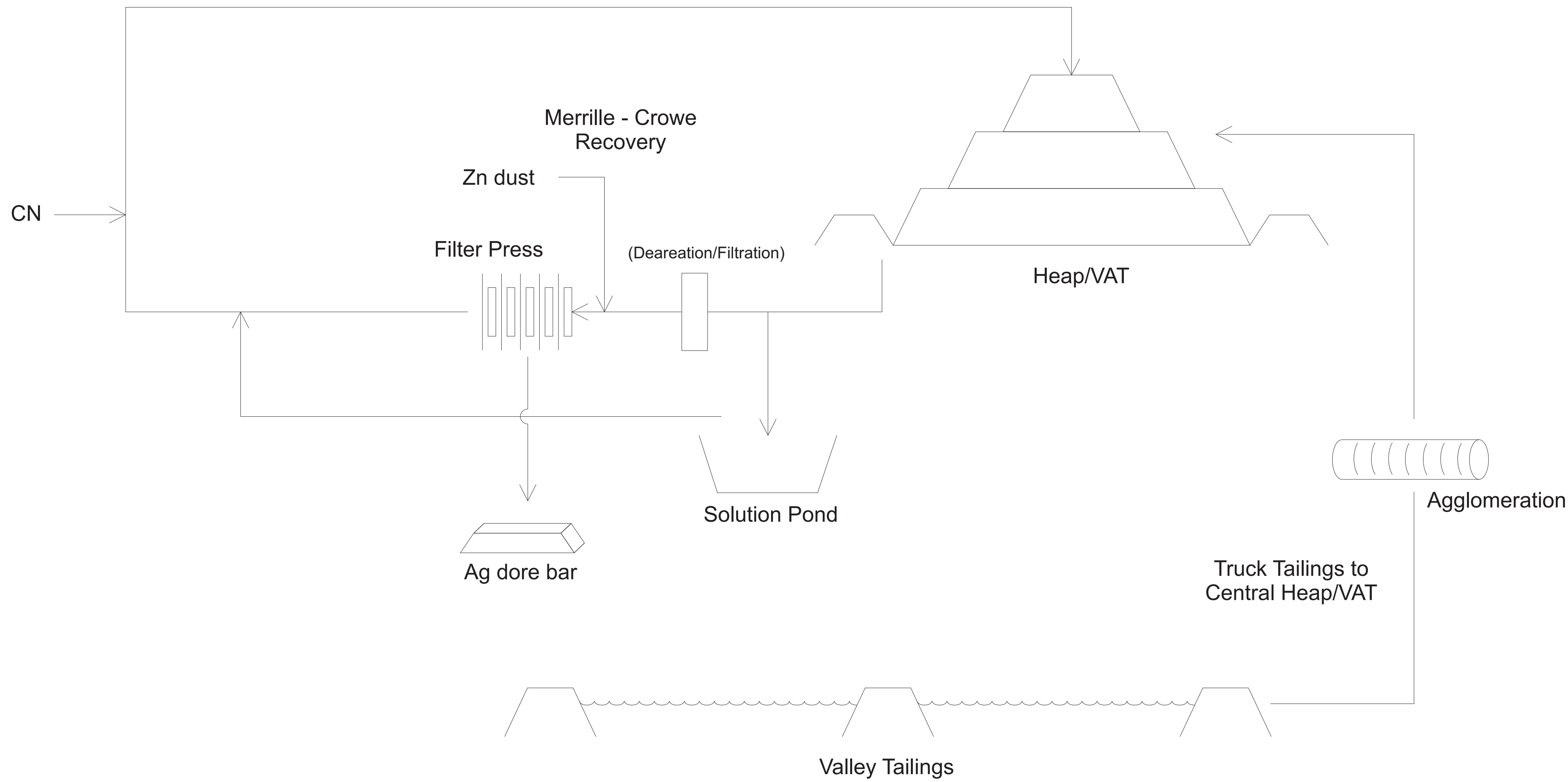
Heap leaching for recovery of gold and silver is an industry proven technology that has been used successfully in the Yukon (Brewery Creek Mine). For the basis of this initial assessment, the process would consist of stacking tailings (likely with grasshopper conveyors) on a lined leach pad and recirculating a weak alkaline cyanide solution over the stacked tailings to recover silver and gold. As discussed, vat leaching is also a likely candidate for reprocessing and is similar to heap leaching but numerous smaller vats are used rather than a single leach pad. Agglomerations of the tailings with cement or lime would likely be required to provide both alkalinity and acceptable permeability for the leach solution. Alternatively, mineralized waste rock could possibly be used as a substrate for mixing tailings. The leach solution is collected in a series of pipes overlain on the liner system and directed to a recovery circuit. A Merrille-Crowe recovery system would be used to recover gold and silver which would be refined on site and a dore product produced that is shipped directly to a refiner.

The Merrille-Crowe Process is a separation technique for removing gold and silver from a cyanide solution. The solution is clarified in special filters, usually coated with diatomaceous earth to produce a clarified solution. Oxygen is then removed by passing through a vacuum deaeration column. Zinc dust is then added to the clarified, deaerated solution which precipitates the gold and silver, zinc having a higher affinity for the cyanide ion than gold and silver. The gold/silver precipitate is then filtered out of the solution, mixed with fluxes and smelted to form crude and impure bars which are sent to a refinery for further separation of the precious metals. A simplified process flowsheet for a typical heap leach reprocessing option is shown in Figure 7.

Given the operational restraints of stacking material on a leach pad in northern winter conditions, the study assumes that tailings rehandling, stacking and leaching would occur over a 150 day period each year (~ May – Sept.). The other important operational restraint that will require attention is the spring freshet period (late April – May) where a large portion of the current tailings becomes flooded with runoff water and will require material handling and water diversion measures in order to maintain a reasonable productivity for material handling.

The general operating parameters used in the heap leaching reprocessing analysis are presented in Table 6.

Figure 7 - Simplified Tailings Reprocessing Flowsheet



4. ECONOMIC ANALYSIS

The economic evaluation indicates a base case pre-tax internal rate of return of 17.7% and a pre-tax net present value of Cdn $7.7 million at a discount rate of 8.0% for the reprocessing project. The summary project and economic information is shown in Table 6.

The pre-tax base case financial model is calculated within the following parameters:

- project life of 4.5 years, 3,600 tpd over 150 day operating season;
- base case metals prices $US: Ag $13/ounce, Au $900/ounce;
- 57% silver recovery;
- US/Canadian exchange rate: 0.85;
- closure and reclamation costs included;
- the model was prepared on a pre-tax basis.

4.1. NPV AND IRR SUMMARY

This study presents the predicted NPV and IRR for the project and a sensitivity analysis of key variables including silver grade, silver recovery, metal prices, capital and operating costs, total resource tonnes and tonnes processed each season. Initial and sustaining capital has been assumed on a year-by-year basis for the life of the Project.

The initial capital includes all capital expenditure prior to first production of silver from the process plant; sustaining capital includes all subsequent capital expenditure, including equipment replacement based on predicted equipment life. An overall contingency of 30% is included in the capital cost estimated, given the preliminary nature of the project understanding. A discounted cash flow rate of 8.0% was assumed.

The detailed capital and operating cost calculations and assumptions are presented in Appendix B and Appendix C. The cash flow analysis is shown in Table 7.

Spider charts for the sensitivity cases are presented in Figure 8 and Figure 9. It is observed that NPV is most sensitive to silver recovery, price and grade and less sensitive to resource tonnes, capital and operating costs. IRR has the same sensitivities as the NPV. The IRR is least sensitive to the total tonnes processed each season.

TABLE 6. ECONOMIC ASSESSMENT SUMMARY DATA

OPERATING PARAMETER	DATA	UNIT
Tonnes Reprocessed	2,371,000	tonnes
Ag Grade	150	gpt
Ag Recovery	57	%
Au Grade	0.15	gpt
Au Recovery	50	%
Daily Reprocessing Rate	3,666	tonnes/day
Seasonal Reprocessing Rate	550,000	tonnes/season
Project Life	4.5	years
Capital Costs	$28,035,000	$ Cdn
Reclamation Costs	$6,250,000	$ Cdn
Operating Costs	$19.83	$/tonne tailings
NPV	$6,350,000	$Cdn @ 8% DR
IRR	17.7	%

Table 7 Economic Analysis

NPV @	0%	$13,668,274	**VALLEY TAILINGS REPROCESSING**
	5%	$ 8,828,178	**TONNES/YEAR**
	8%	$ 6,352,519	**550,000**
	15%	$ 1,545,934	**TONNES/DAY**
IRR	17.7%		**3,667**

	Development Period -3	-2	-1	**Year** 1	**Year** 2	**Year** 3	**Year** 4	**Year** 5
				1	2	3	4	5
Production Statistics								
Tailings Tonnes Reprocessed				550,000	550,000	550,000	550,000	171,000
Cumulative Tonnes Reprocessed				550,000	1,100,000	1,650,000	2,200,000	2,371,000
Total Tonnes Mined				550,000	550,000	550,000	550,000	171,000
Silver Grade gpt			150.00	150.000	150.000	150.000	150.000	150.000
Gold Grade gpt			0.15	0.150	0.150	0.150	0.150	0.150
Silver Ounces Delivered				2,652,435	2,652,435	2,652,435	2,652,435	824,666
Gold Ounces Delivered				2,652	2,652	2,652	2,652	825
Cumul Silver Ounces Delivered				2,652,435	5,304,869	7,957,304	10,609,738	11,434,404
Cumul Gold Ounces Delivered				2,652	5,305	7,957	10,610	11,434
Overall Silver Recovery %			57%	57.0%	57.0%	57.0%	57.0%	57.0%
Overall Gold Recovery %			50%	50%	50%	50%	50%	50%
Silver Ounces Recovered				1,511,888	1,511,888	1,511,888	1,511,888	470,060
Gold Ounces Recovered				1,326	1,326	1,326	1,326	412
Cumul Silver Ounces Recovered				1,511,888	3,023,775	4,535,663	6,047,551	6,517,611
Cumul Gold Ounces Recovered				1,326	2,652	3,979	5,305	5,717
Operating Costs $/tonne								
Process/Manpower	tonne/ore			$ 12.018	$ 12.018	$ 12.018	$ 12.018	$ 12.018
Mining	tonne/ore			$ 3.454	$ 3.454	$ 3.454	$ 3.454	$ 3.454
G&A	tonne/ore			$ 1.198	$ 1.198	$ 1.198	$ 1.198	$ 1.198
Environmental	tonne/ore			$ 2.636	$ 2.636	$ 2.636	$ 2.636	$ 2.636
Power Supply	tonne/ore			$ 0.527	$ 0.527	$ 0.527	$ 0.527	$ 0.527
Operating Costs $/tonne				$ 19.834	$ 19.834	$ 19.834	$ 19.834	$ 19.834
Operating Costs $								
Process/Manpower			$28,495,456	$ 6,610,081	$ 6,610,081	$ 6,610,081	$ 6,610,081	$ 2,055,134
Mining			$ 8,189,543	$ 1,899,725	$ 1,899,725	$ 1,899,725	$ 1,899,725	$ 590,642
G&A			$ 2,839,435	$ 658,663	$ 658,663	$ 658,663	$ 658,663	$ 204,784
Environmental			$ 6,250,818	$ 1,450,000	$ 1,450,000	$ 1,450,000	$ 1,450,000	$ 450,818
Power Supply			$ 1,250,357	$ 290,045	$ 290,045	$ 290,045	$ 290,045	$ 90,178
Operating Cash Costs				$10,908,513	$10,908,513	$10,908,513	$10,908,513	$ 3,391,556
Cash Cost/Ounce				$ 7.22	$ 7.22	$ 7.22	$ 7.22	$ 7.22
Revenues								
Silver Price ($ US)	$Cdn:US	0.85	$ 13.00	$ 15.29	$ 15.29	$ 15.29	$ 15.29	$ 15.29
Gold Price ($US)			$ 900	$ 1,059	$ 1,059	$ 1,059	$ 1,059	$ 1,059
Revenue ($Cdn)				$24,527,219	$24,527,219	$24,527,219	$24,527,219	$ 7,625,735
Operating Expenses ($Cdn)				$10,908,513	$10,908,513	$10,908,513	$10,908,513	$ 3,391,556
Refining			$ 0.50	$ 755,944	$ 755,944	$ 755,944	$ 755,944	$ 235,030
Cash Flow			$55,450,197	$12,862,762	$12,862,762	$12,862,762	$12,862,762	$ 3,999,150
Other Costs								
Initial Capital			$28,035,188					
Equipment Salvage Value	10%		$ 2,803,519					
Sustaining Capital				$ 300,000	$ 300,000	$ 300,000	$ 300,000	$ 300,000
Sunk Costs Credit from INAC			$ -					
Loan Repayment								
IMA Royalty NSR				$ -	$ -	$ -	$ -	$ -
Federal Royalty NSR				$ -	$ -	$ -	$ -	$ -
Pre-Tax Cash Flow				$12,562,762	$12,562,762	$12,562,762	$12,562,762	$ 3,699,150
Depreciation	4.31			$10,344,984	$ 6,527,685	$ 4,118,969	$ 2,599,070	$ 1,640,013
Tax Base				$ 2,217,778	$ 6,035,077	$ 8,443,793	$ 9,963,692	$ 2,059,137
Tax 35%				$ 776,222	$ 2,112,277	$ 2,955,327	$ 3,487,292	$ 720,698
After Tax Net Cash Flow				$11,786,540	$10,450,485	$ 9,607,434	$ 9,075,469	$ 2,978,452
Net Profits Interest	5%			$ 589,327	$ 522,524	$ 480,372	$ 453,773	$ 148,923
After Tax/Interest Profit			-$28,035,188	$11,197,213	$ 9,927,961	$ 9,127,063	$ 8,621,696	$ 2,829,529
Cumulative Cash Flow			-$28,035,188	-$16,837,975	-$ 6,910,014	$ 2,217,048	$10,838,744	$13,668,274

FIGURE 8 – NPV SENSITIVITY ANALYSIS

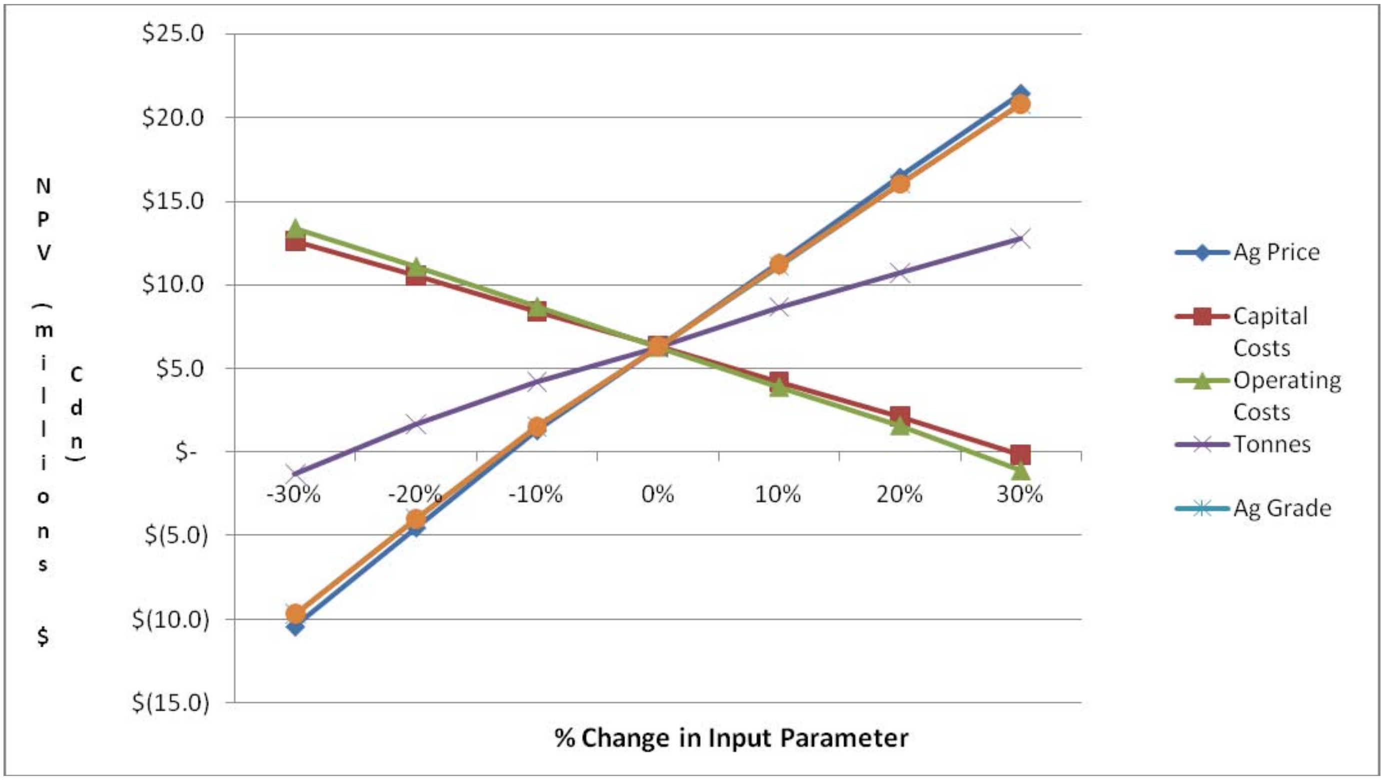
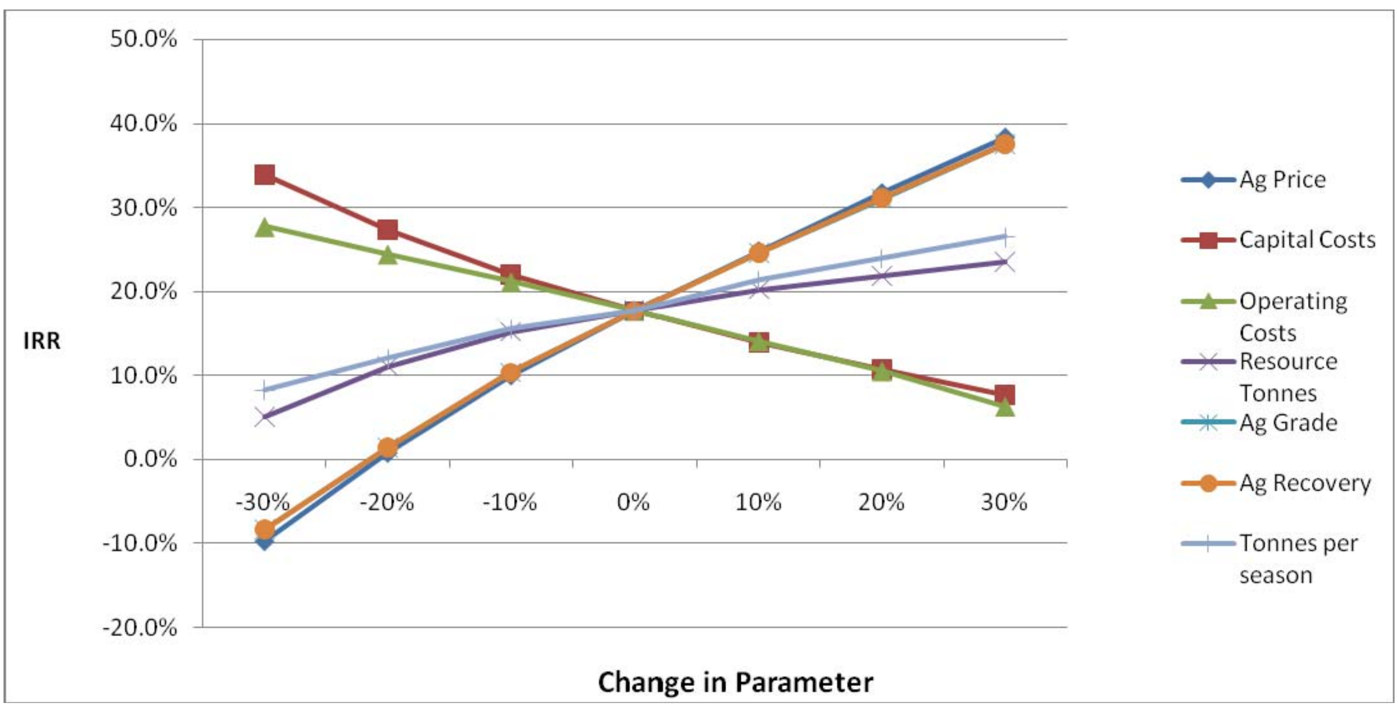


FIGURE 9. IRR SENSITIVITY ANALYSIS



5. COMPATIBILITY WITH FINAL CLOSURE OPTIONS

SRK has completed initial closure options for the valley tailings. The two basic options presented are covering the tailings in place with a nominal 0.5 m cover or consolidating the tailings, adding lime and then covering the larger consolidated pile with a nominal 0.5 m cover. Both closure options are shown in Figures 10 and 11. These closure options were part of a March 2009 risk assessment for overall closure options.

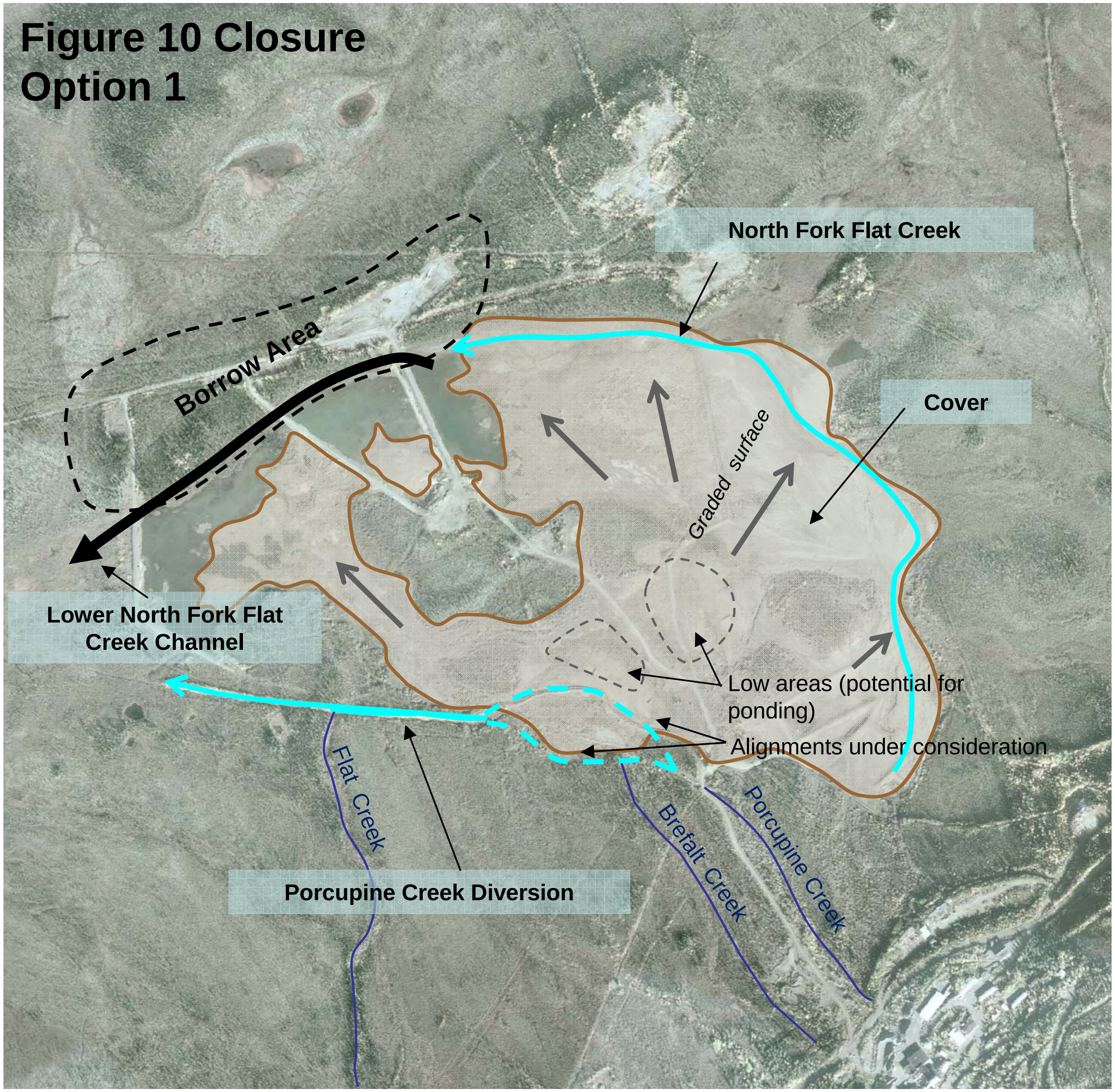
The potential heap leach reprocessing approach is compatible with the closure option of consolidating the tailings into a larger pile. The end result of a heap leach tailings reprocessing system would be a larger consolidated pile of tailings that have been amended with lime to increase the overall alkalinity. The tailings would be constructed over a liner which would provide further long-term water management advantages and along with the addition of lime will add further alkalinity and more robust geochemical stability. Once the tailings have been leached, the pile will have to be neutralized of residual cyanide prior to covering and revegetation. Neutralization of heap leach piles is an industry proven practice and was recently completed at the Brewery Creek Mine. Final drainage from the Brewery Creek heap meets direct discharge criteria for release into the receiving environment. Figure 12 shows the final reclaimed heap at Brewery Creek as an example of what a reprocessed/reclaimed heap would result in. Within the economic analysis, a total of $6,235,000 is included for neutralization, cover and revegetation of the pile once reprocessing and metals recovery is complete. This amounts to approximately $2.63/tonne of material. As a comparison, the costs to neutralize, cover and revegetate the Brewery Creek heap was approximately $0.35/tonne of material (2003 dollars).

Costs to load and haul the tailings to a central heap leach facility are included in the economics along with costs for the addition of 4 kg/tonne of lime to increase the alkalinity necessary for cyanidation. These costs are also included in the SRK analysis to complete the closure option of consolidation and covering. As a comparison, Table 8 summarizes the costs included in the reprocessing study that are also included in the stand alone closure option of consolidation and covering.

TABLE 8. REPROCESSING AND CLOSURE OPTIONS COST COMPARISON

PARAMETER	**REPROCESSING**	**STAND ALONE CLOSURE (SRK)**
Load/Haul/Place	\$5.26 / m^3	\$5.00 / m^3
Lime Amendment	\$3.45 / m^3	\$5.71 / m^3
Recontour	\$0.03 / m^3	Na
Place Soil Cover (\$ per 148,000 m^3 soil placed)	\$5.00 / m^3	\$5.00 / m^3
Revegetate	\$0.45 / m^2	\$0.45 / m^2

The costs for the consolidation closure option are fully captured in the tailings reprocessing costs resulting in the reprocessing approach representing a true stand alone option for closure.



Key Features

- Regrade and cover tailings surface
- Upgrade current water management plan
- Flatten profile of Dam #1, 2, 3
- Construct Lower North Fork Flat Creek Channel through borrow area

North Fork Flat Creek

- Establish stable channel to pass design flows
  - Line as necessary
  - Armour as necessary
  - Stabilize slopes and contour to promote drainage towards channel
  - Channel to follow edge of VTF to encourage natural revegetation

Porcupine Creek Diversion

- Remove tailings from streambed
- Select final alignment
- Upgrade channel to pass design flows
  - Line as necessary
  - Armour as necessary
  - Recontour and stabilize channel side slopes

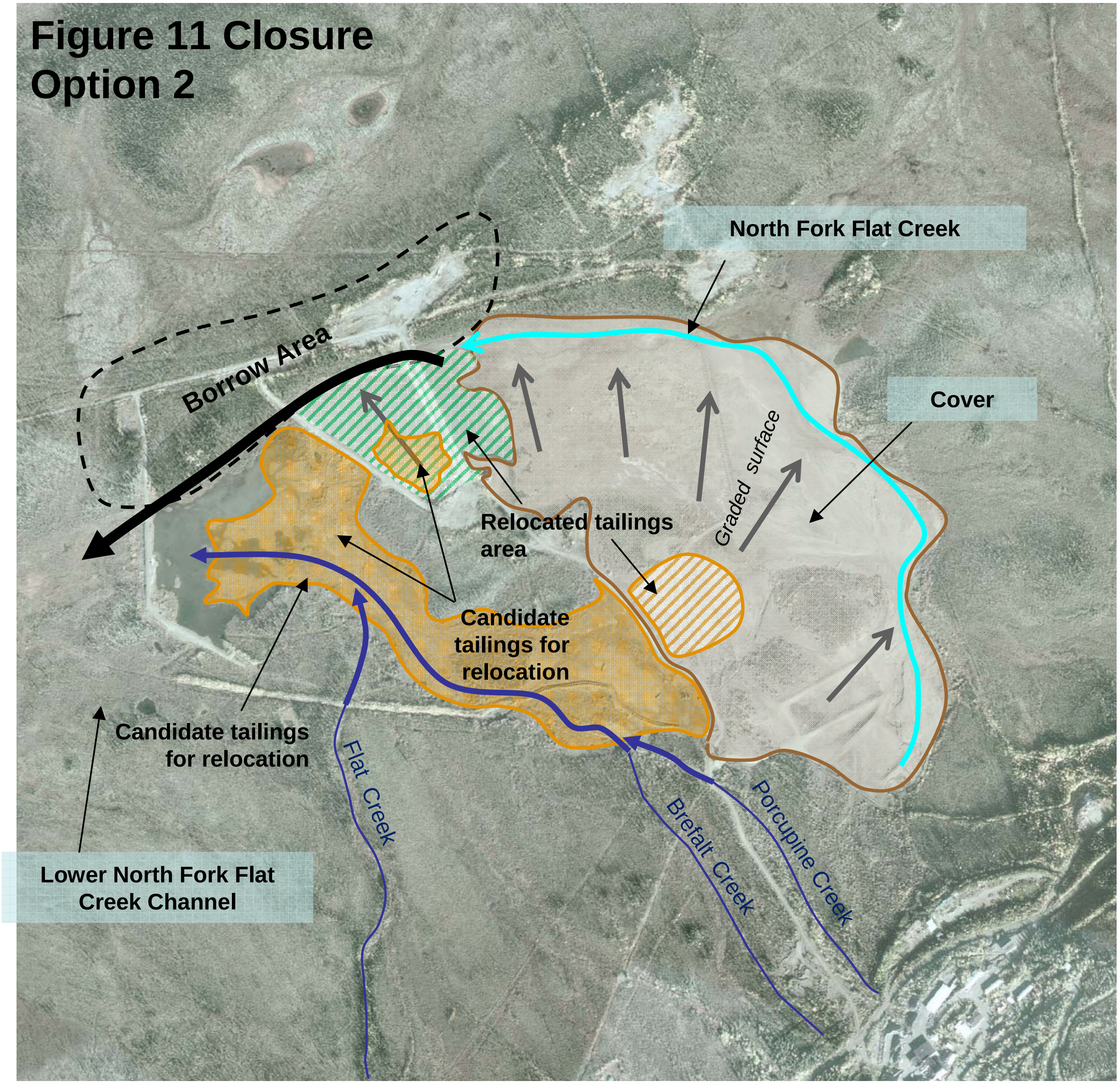
Cover

- Cover placement to isolate tailings from the environment and provide a medium for vegetative growth

Lower North Fork Flat Creek Channel

- Construct channel through borrow area

Figure 11 Closure Option 2



Key Features

- Regrade and cover tailings surface
- Flatten profile of Dams #1, 2, 3
- Construct Lower North Fork Flat Creek Channel through borrow area
- Abandon Porcupine Creek Diversion and develop stable channel over original ground
- Partial tailings relocation

North Fork Flat Creek

- Establish stable channel to pass design flows
  - Line as necessary
  - Armour as necessary
  - Stabilize slopes and contour to promote drainage towards channel
  - Channel to follow edge of VTF to encourage natural revegetation

Flat Creek, Brefalt Creek and Porcupine Creek

- Establish stable channel
  - Line as necessary
  - Armour as necessary
  - Stabilize slopes and contour to promote drainage towards channel

Original Ground Exposed by Tailings Relocation

- Establish vegetation in exposed soil
- Leave exposed soil
- Establish stable water features
  - Ponds
  - Wetlands
  - Channels

Cover

- Cover placement to isolate tailings from the environment and provide a medium for vegetative growth

Relocated Tailings Area

- Consolidate tailings and place appropriate cover

Lower North Fork Flat Creek Channel

- Construct channel through borrow area

FIGURE 12. EXAMPLE OF NEUTRALIZED RECLAIMED HEAP (BREWERY CREEK, YUKON)

6. CONCLUSIONS AND RECOMMENDATIONS

Tailings reprocessing projects are generally a function of effective materials handling and management. A successful Elsa tailings reprocessing project will be no exception and a key variable will be cost effective and efficient movement of the tailings. Some of the general conclusions and recommendations drawn from this scoping level study include:

- Reprocessing the valley tailings for silver recovery has the potential to be a positive economic project even on a stand alone basis where no cost credit or offset for the costs of rehandling of tailings under the closure option are included;
- Use of cyanide for tailings reprocessing may be present additional permitting challenges but the use of cyanide in a northern heap leach operation at Brewery Creek has been demonstrated to be successful from an environmental protection standpoint. Consultation and support from the First Nation of Na-cho Nyak Dun and other stakeholders is critical to a potential tailings reprocessing option;
- The reprocessing of silver from the Valley Tailings can be accomplished in a manner that is consistent with current closure planning options and could provide significant offsets in final closure costs for the tailings;
- Material handling and management is one of the key considerations in any tailings reprocessing scenario. Tailings trafficability test planned for 2009 should help address materials and equipment handling;
- Spring freshet conditions and short operating seasons present challenges in the material handling and sequencing required to obtain reasonable daily productivity levels necessary for an economic project;
- Additional sampling, testwork and engineering are obviously required before advancing the reprocessing option;
- Given the potential benefits of reprocessing the tailings, additional work should proceed given the status of the closure planning schedule. Recommended course of follow-up includes:
  - Sample and resource estimation program as per NI 43-101 requirements;
  - Metallurgical test program to select preferred process flowsheet;
  - Advanced engineering desk top review to determine process criteria and fatal flaws.

Appendix A

Metallurgical Testwork Results

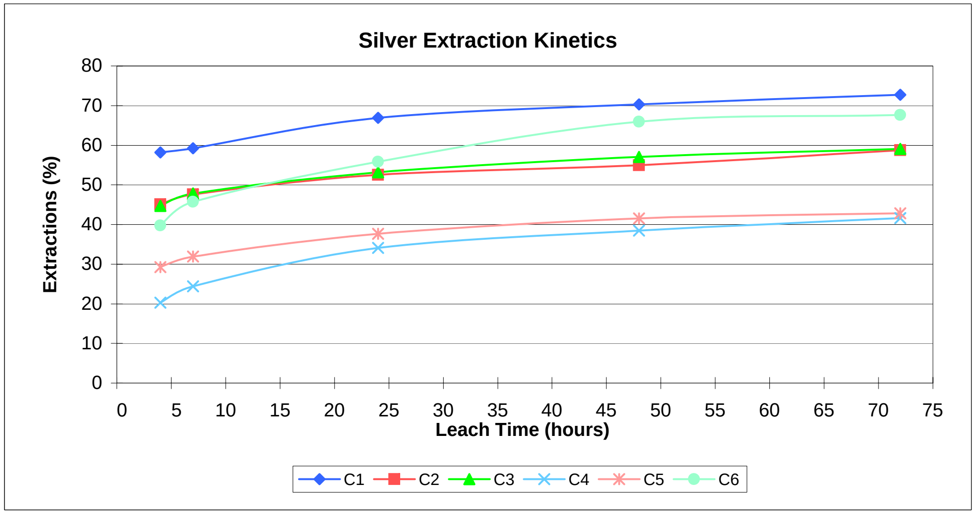
CYANIDATION TEST SUMMARY

Client: Alexco
Test: C1 through C6
Sample: Historic tailing samples from Keno Hill

Date: 17-Feb-09
Project: 0901302

Objective: To determine the Ag extraction by direct cyanidation of historic tailing samples in 1 g/L NaCN

Test No	Sample ID	Test Conditions pH	Measured Head (as-received) Ag (g/t)	Calculated Head Ag (g/t)	72-h Extraction Ag (%)	Residue Grade Ag (g/t)	Consumption (kg/t) NaCN	Lime
C1	UKTP-03 A+B+C	10.5-11.0	118.7	142.8	72.7	39.00	0.76	7.0
C2	UKTP-07 A+D	10.5-11.0	81.2	93.0	58.7	38.40	0.81	16.3
C3	UKTP-12 A+H+N	10.5-11.0	107.5	118.0	59.1	48.30	0.82	2.4
C4	UKTP-14 A+E+H	10.5-11.0	213.9	246.9	41.6	144.20	1.03	2.5
C5	UKTP-15 A+C+F	10.5-11.0	80.0	88.3	42.8	50.50	0.76	8.2
C6	UKTP-17 A+C+E	10.5-11.0	267.5	321.3	67.6	104.10	0.91	10.2



CYANIDATION TEST REPORT

Client: Alexco
Test: C1
Sample: UKTP-03 A+B+C

Date: 17-Feb-09
Project: 0901302

Objective: To determine the Ag extraction by direct cyanidation of historic tailing samples

TEST CONDITIONS

Solids:	1,942	g
Solution:	3,000	g
Solids:	39	%
Grind Size -74um:	N/A	%
Initial NaCN:	1.0	g/L
Target pH:	10.5-11.0	
Test Duration:	72	hours

TEST DESCRIPTION

- sample repulped to 40% solids
- adjusted to and maintained pH 10.5-11.0
- adjusted to 1.0g/L NaCN
- sampled at 4, 7, 24, & 48 hours
- test ends after 72 hours
- filter and displacement wash with hot cyanide solution followed by two hot water displacement washes
- solution and solids assay for Ag

HEAD GRADE

	Ag	
Calculated Total:	142.8	g/t
Measured Total:	118.7	g/t

LEACH TEST DATA

Time	NaCN		Lime	pH		dO_2	Slurry Weight	Solution Vol.	Solution Assay Vol.	Solution Ag	
(hours)	(g/L)	(g)	(g)	before	after	(mg/L)	(g)	(mL)	(mL)	(mg/L)	(mg)
0	1.00	3.0	6.50	6.0	10.5		4,942	3,000			
2	0.80	0.6	2.00	9.6	10.7	7.6			5		
4	0.89	0.3	1.00	9.8	10.6		5,106	3,164	30	50.9	161.4
7	0.92	0.2	1.00	10.0	10.8		5,094	3,152	30	51.5	164.4
24	0.93	0.2	1.00	9.8	10.8	8.1	5,058	3,116	30	58.3	185.5
30	0.99	0.0	1.00	10.2	11.0				5		
48	0.98	0.1	0.50	10.0	10.6	8.0	5,046	3,104	30	61.6	195.0
54	1.00		0.50	10.0	10.7				5		
72	0.98				9.9	8.9	5,004	3,062		64.5	201.6
Total		4.5	13.50								

SOLIDS

Time	Residue Weight	Residue Ag	
(hours)	(g)	(g/t)	(mg)
72	1,942	39.0	75.7

CYANIDATION RESULTS

Time	Distribution Ag	Reagent Consumption NaCN	Reagent Consumption $Ca(OH)_2$	Reducing Power 0.1 N $KMnO_4$/L
(hours)	(%)	(kg/t)	(kg/t)	(mL)
4	58.2	0.40		
7	59.3	0.53		
24	66.9	0.66		
48	70.3	0.70		
72	72.7	**0.76**	**6.95**	**110**
Residue	27.3			
Total	100.0			

CYANIDATION TEST REPORT

Client: Alexco
Test: C2
Sample: UKTP-07 A+D

Date: 17-Feb-09
Project: 0901302

Objective: To determine the Ag extraction by direct cyanidation of historic tailing samples

TEST CONDITIONS

Solids:	1,903	g
Solution:	3,000	g
Solids:	39	%
Grind Size -74um:	N/A	%
Initial NaCN:	1.0	g/L
Target pH:	10.5-11.0	
Test Duration:	72	hours

TEST DESCRIPTION

- sample repulped to 40% solids
- adjusted to and maintained pH 10.5-11.0
- adjusted to 1.0g/L NaCN
- sampled at 4, 7, 24, & 48 hours
- test ends after 72 hours
- filter and displacement wash with hot cyanide solution followed by two hot water displacement washes
- solution and solids assay for Ag

HEAD GRADE

	Ag	
Calculated Total:	93.0	g/t
Measured Total:	81.2	g/t

LEACH TEST DATA

Time	NaCN		Lime	pH		dO_2	Slurry Weight	Solution			
								Vol.	Assay Vol.	Ag	
(hours)	(g/L)	(g)	(g)	before	after	(mg/L)	(g)	(mL)	(mL)	(mg/L)	(mg)
0	1.00	3.0	21.00	5.7	10.6		4,903	3,000			
2	0.74	0.8	2.50	9.7	10.5	1.4			5		
4	0.84	0.5	1.50	10.1	10.7		5,297	3,394	30	23.5	79.9
7	0.92	0.2	1.00	10.2	10.8		5,291	3,388	30	24.6	84.2
24	0.94	0.2	1.50	9.7	10.6	1.5	5,257	3,354	30	27.2	93.0
30	0.99	0.0	1.00	10.0	10.7				5		
48	0.98	0.1	1.50	9.6	10.6	3.9	5,245	3,342	30	28.6	97.3
54	1.00		1.00	9.9	10.7				5		
72	0.97				9.8	8.0	5,239	3,336		30.6	104.0
Total		4.8	31.00								

SOLIDS

Time	Residue		
	Weight	Ag	
(hours)	(g)	(g/t)	(mg)
72	1,903	38.4	73.1

CYANIDATION RESULTS

Time	Distribution	Reagent Consumption		Reducing Power
	Ag	NaCN	$Ca(OH)_2$	0.1 N $KMnO_4$/L
(hours)	(%)	(kg/t)	(kg/t)	(mL)
4	45.1	0.49		
7	47.6	0.60		
24	52.5	0.71		
48	55.0	0.75		
72	58.7	**0.81**	**16.29**	**135**
Residue	41.3			
Total	100.0			

CYANIDATION TEST REPORT

Client: Alexco
Test: C3
Sample: UKTP-12 A+H+N

Date: 17-Feb-09
Project: 0901302

Objective: To determine the Ag extraction by direct cyanidation of historic tailing samples

TEST CONDITIONS

Solids:	1,963	g
Solution:	3,000	g
Solids:	40	%
Grind Size -74um:	N/A	%
Initial NaCN:	1.0	g/L
Target pH:	10.5-11.0	
Test Duration:	72	hours

TEST DESCRIPTION

- sample repulped to 40% solids
- adjusted to and maintained pH 10.5-11.0
- adjusted to 1.0g/L NaCN
- sampled at 4, 7, 24, & 48 hours
- test ends after 72 hours
- filter and displacement wash with hot cyanide solution followed by two hot water displacement washes
- solution and solids assay for Ag

HEAD GRADE

	Ag	
Calculated Total:	118.0	g/t
Measured Total:	107.5	g/t

LEACH TEST DATA

Time	NaCN		Lime	pH		dO_2	Slurry Weight	Solution			
								Vol.	Assay Vol.	Ag	
(hours)	(g/L)	(g)	(g)	before	after	(mg/L)	(g)	(mL)	(mL)	(mg/L)	(mg)
0	1.00	3.0	1.00	8.0	10.6		4,963	3,000			
2	0.80	0.6	1.00	9.8	11.0	11.9			5		
4	0.90	0.3	0.50	10.4	11.1		5,032	3,069	30	33.7	103.6
7	0.96	0.1		10.7			5,010	3,047	30	36.0	110.9
24	0.94	0.2	0.50	10.1	10.7	9.1	4,980	3,017	30	40.1	123.2
30	0.98	0.1	0.50	10.4	11.1				5		
48	0.97	0.1	0.40	10.0	10.8	8.9	4,994	3,031	30	42.4	132.2
54	0.98	0.1	0.90	10.3	11.1				5		
72	0.93				10.1	9.5	4,978	3,015		43.7	136.9
Total		4.4	4.80								

SOLIDS

Time	Residue		
	Weight	Ag	
(hours)	(g)	(g/t)	(mg)
72	1,963	48.3	94.8

CYANIDATION RESULTS

Time	Distribution	Reagent Consumption		Reducing Power
	Ag	NaCN	$Ca(OH)_2$	0.1 N $KMnO_4$/L
(hours)	(%)	(kg/t)	(kg/t)	(mL)
4	44.7	0.43		
7	47.8	0.50		
24	53.2	0.60		
48	57.0	0.67		
72	59.1	**0.82**	**2.45**	**85**
Residue	40.9			
Total	100.0			

CYANIDATION TEST REPORT

Client: Alexco
Test: C4
Sample: UKTP-14 A+E+H

Date: 17-Feb-09
Project: 0901302

Objective: To determine the Ag extraction by direct cyanidation of historic tailing samples

TEST CONDITIONS

Solids:	1,968	g
Solution:	3,000	g
Solids:	40	%
Grind Size -74um:	N/A	%
Initial NaCN:	1.0	g/L
Target pH:	10.5-11.0	
Test Duration:	72	hours

TEST DESCRIPTION

- sample repulped to 40% solids
- adjusted to and maintained pH 10.5-11.0
- adjusted to 1.0g/L NaCN
- sampled at 4, 7, 24, & 48 hours
- test ends after 72 hours
- filter and displacement wash with hot cyanide solution followed by two hot water displacement washes
- solution and solids assay for Ag

HEAD GRADE

	Ag	
Calculated Total:	246.9	g/t
Measured Total:	213.9	g/t

LEACH TEST DATA

Time	NaCN		Lime	pH		dO_2	Slurry Weight	Solution			
								Vol.	Assay Vol.	Ag	
(hours)	(g/L)	(g)	(g)	before	after	(mg/L)	(g)	(mL)	(mL)	(mg/L)	(mg)
0	1.00	3.0	1.00	7.5	10.6		4,968	3,000			
2	0.74	0.8	1.00	9.3	10.7	7.5			5		
4	0.90	0.3	0.50	10.0	10.6		5,026	3,058	30	32.2	98.6
7	0.98	0.1	0.50	10.2	11.0		4,996	3,028	30	38.8	118.6
24	0.91	0.3	0.50	9.8	10.9	8.8	4,980	3,012	30	54.3	165.8
30	0.96	0.1	0.50	10.1	11.1				5		
48	0.95	0.2	0.40	9.7	10.8	8.7	4,986	3,018	30	60.5	186.8
54	0.98	0.1	0.50	9.9	11.0				5		
72	0.90				9.7	9.1	4,974	3,006		65.1	202.1
Total		4.7	4.90								

SOLIDS

Time	Residue		
	Weight	Ag	
(hours)	(g)	(g/t)	(mg)
72	1,968	144.2	283.8

CYANIDATION RESULTS

Time	Distribution	Reagent Consumption		Reducing Power
	Ag	NaCN	$Ca(OH)_2$	0.1 N $KMnO_4$/L
(hours)	(%)	(kg/t)	(kg/t)	(mL)
4	20.3	0.52		
7	24.4	0.57		
24	34.1	0.71		
48	38.5	0.84		
72	41.6	**1.03**	**2.49**	**185**
Residue	58.4			
Total	100.0			

CYANIDATION TEST REPORT

Client: Alexco
Test: C5
Sample: UKTP-15 A+C+F

Date: 17-Feb-09
Project: 0901302

Objective: To determine the Ag extraction by direct cyanidation of historic tailing samples

TEST CONDITIONS

Solids:	1,921	g
Solution:	3,000	g
Solids:	39	%
Grind Size -74um:	N/A	%
Initial NaCN:	1.0	g/L
Target pH:	10.5-11.0	
Test Duration:	72	hours

TEST DESCRIPTION

- sample repulped to 40% solids
- adjusted to and maintained pH 10.5-11.0
- adjusted to 1.0g/L NaCN
- sampled at 4, 7, 24, & 48 hours
- test ends after 72 hours
- filter and displacement wash with hot cyanide solution followed by two hot water displacement washes
- solution and solids assay for Ag

HEAD GRADE

	Ag	
Calculated Total:	88.3	g/t
Measured Total:	80.0	g/t

LEACH TEST DATA

Time	NaCN		Lime	pH		dO_2	Slurry Weight	Solution Vol.	Solution Assay Vol.	Solution Ag	
(hours)	(g/L)	(g)	(g)	before	after	(mg/L)	(g)	(mL)	(mL)	(mg/L)	(mg)
0	1.00	3.0	9.00	6.7	10.7		4,921	3,000			
2	0.72	0.8	2.00	9.4	10.7	4.2			5		
4	0.90	0.3	1.00	9.8	10.7		5,033	3,112	30	15.9	49.7
7	0.99	0.0	0.70	10.1	10.7		5,023	3,102	30	17.3	54.1
24	0.95	0.2	1.00	9.6	10.7	7.4	4,981	3,060	30	20.5	63.9
30	1.00		0.50	10.0	10.6				5		
48	0.98	0.1	1.00	9.6	10.7	8.0	4,997	3,076	30	22.3	70.4
54	1.00		0.50	10.0	10.7				5		
72	0.96				9.8	8.3	4,969	3,048		23.0	72.6
Total		4.4	15.70								

SOLIDS

Time	Residue Weight	Residue Ag	
(hours)	(g)	(g/t)	(mg)
72	1,921	50.5	97.0

CYANIDATION RESULTS

Time	Distribution Ag	Reagent Consumption NaCN	Reagent Consumption $Ca(OH)_2$	Reducing Power 0.1 N $KMnO_4$/L
(hours)	(%)	(kg/t)	(kg/t)	(mL)
4	29.3	0.54		
7	31.9	0.56		
24	37.7	0.66		
48	41.5	0.68		
72	42.8	**0.76**	**8.17**	**140**
Residue	57.2			
Total	100.0			

CYANIDATION TEST REPORT

Client: Alexco
Test: C6
Sample: UKTP-17 A+C+E

Date: 17-Feb-09
Project: 0901302

Objective: To determine the Ag extraction by direct cyanidation of historic tailing samples

TEST CONDITIONS

Solids:	1,951	g
Solution:	3,000	g
Solids:	39	%
Grind Size -74um:	N/A	%
Initial NaCN:	1.0	g/L
Target pH:	10.5-11.0	
Test Duration:	72	hours

TEST DESCRIPTION

- sample repulped to 40% solids
- adjusted to and maintained pH 10.5-11.0
- adjusted to 1.0g/L NaCN
- sampled at 4, 7, 24, & 48 hours
- test ends after 72 hours
- filter and displacement wash with hot cyanide solution followed by two hot water displacement washes
- solution and solids assay for Ag

HEAD GRADE

	Ag	
Calculated Total:	321.3	g/t
Measured Total:	267.5	g/t

LEACH TEST DATA

Time	NaCN		Lime	pH		dO_2	Slurry Weight	Solution			
								Vol.	Assay Vol.	Ag	
(hours)	(g/L)	(g)	(g)	before	after	(mg/L)	(g)	(mL)	(mL)	(mg/L)	(mg)
0	1.00	3.0	10.50	5.8	10.9		4,951	3,000			
2	0.71	0.9	2.00	9.2	10.6	3.8			5		
4	0.90	0.3	1.50	9.5	10.7		5,033	3,082	30	80.7	249.2
7	0.98	0.1	1.00	9.8	10.7		5,023	3,072	30	92.4	286.7
24	0.91	0.3	1.50	9.4	10.7	6.8	4,981	3,030	30	113.7	350.2
30	1.00		0.80	9.9	10.7				5		
48	0.96	0.1	1.50	9.5	10.7	7.9	4,997	3,046	30	132.5	413.3
54	1.00		1.00	10.0	10.9				5		
72	0.94				9.8	9.0	4,969	3,018		135.6	423.6
Total		4.6	19.80								

SOLIDS

Time	Residue		
	Weight	Ag	
(hours)	(g)	(g/t)	(mg)
72	1,951	104.1	203.1

CYANIDATION RESULTS

Time	Distribution	Reagent Consumption		Reducing Power
	Ag	NaCN	$Ca(OH)_2$	0.1 N $KMnO_4$/L
(hours)	(%)	(kg/t)	(kg/t)	(mL)
4	39.8	0.56		
7	45.8	0.59		
24	55.9	0.75		
48	66.0	0.81		
72	67.6	**0.91**	**10.15**	**165**
Residue	32.4			
Total	100.0			

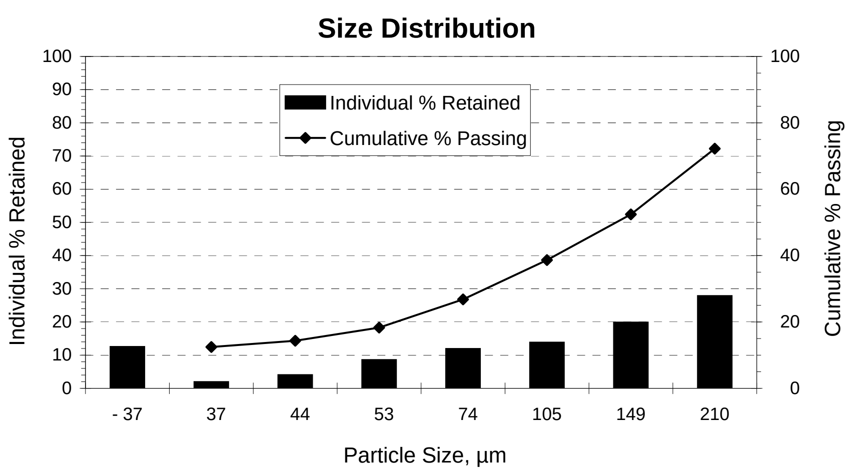
SIZE ANALYSIS REPORT

Client: Alexco
Test: C1
Sample: Cyanidation Residue
Grind: N/A

Date: 25-Feb-09
Project: 0901302

Sieve Size		Individual	Cumulative
Tyler Mesh	**Micrometers**	**% Retained**	**% Passing**
65	210	27.8	72.2
100	149	19.8	52.4
150	105	13.8	38.7
200	74	11.9	26.8
270	53	8.5	18.3
325	44	4.0	14.3
400	37	1.9	12.5
Undersize	- 37	12.5	-
TOTAL:		100.0	

80 % Passing Size (µm) = 234



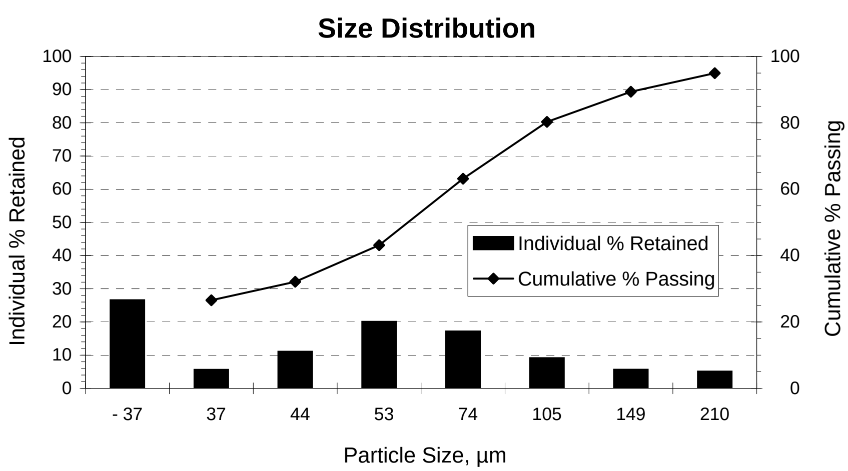
SIZE ANALYSIS REPORT

Client: Alexco
Test: C2
Sample: Cyanidation Residue
Grind: N/A

Date: 25-Feb-09
Project: 0901302

Sieve Size		Individual	Cumulative
Tyler Mesh	**Micrometers**	**% Retained**	**% Passing**
65	210	5.0	95.0
100	149	5.6	89.4
150	105	9.1	80.3
200	74	17.1	63.2
270	53	20.0	43.1
325	44	11.0	32.1
400	37	5.6	26.5
Undersize	- 37	26.5	-
TOTAL:		100.0	

80 % Passing Size (µm) = 104



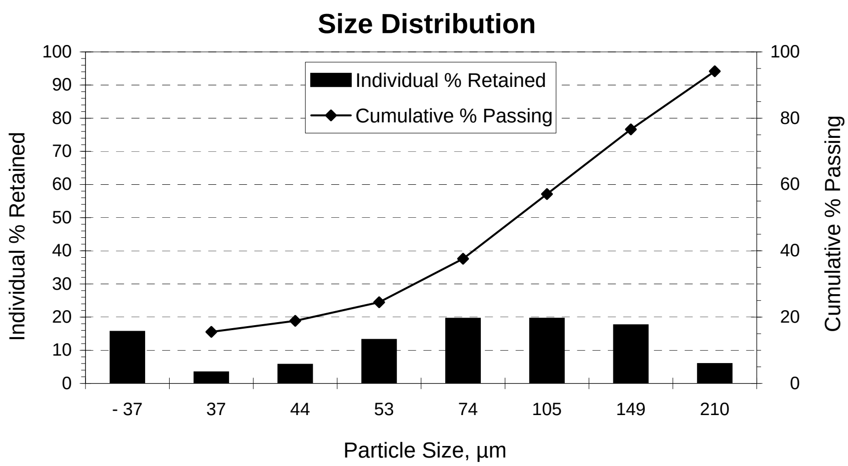
SIZE ANALYSIS REPORT

Client: Alexco
Test: C3
Sample: Cyanidation Residue
Grind: N/A

Date: 25-Feb-09
Project: 0901302

Sieve Size		Individual	Cumulative
Tyler Mesh	**Micrometers**	**% Retained**	**% Passing**
65	210	5.8	94.2
100	149	17.5	76.6
150	105	19.5	57.1
200	74	19.5	37.6
270	53	13.1	24.5
325	44	5.6	18.9
400	37	3.3	15.5
Undersize	- 37	15.5	-
TOTAL:		100.0	

80 % Passing Size (µm) = 160



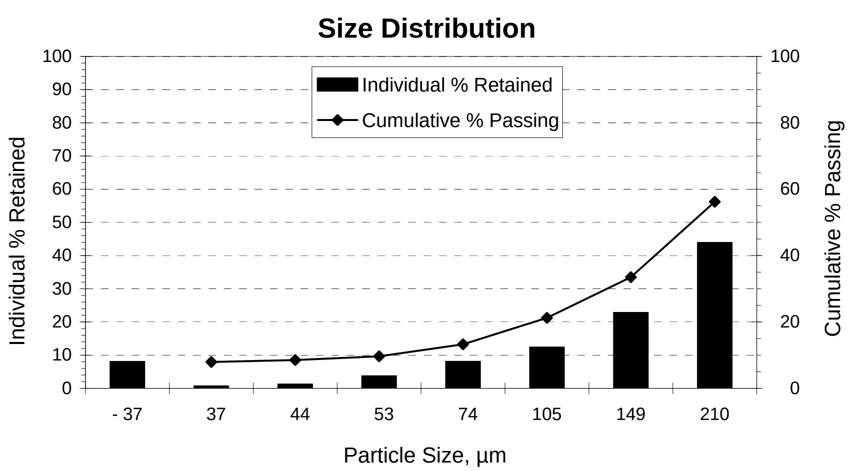
SIZE ANALYSIS REPORT

Client: Alexco
Test: C4
Sample: Cyanidation Residue
Grind: N/A

Date: 25-Feb-09
Project: 0901302

Sieve Size		Individual	Cumulative
Tyler Mesh	**Micrometers**	**% Retained**	**% Passing**
65	210	43.8	56.2
100	149	22.7	33.5
150	105	12.3	21.2
200	74	8.0	13.3
270	53	3.6	9.6
325	44	1.1	8.5
400	37	0.6	7.9
Undersize	- 37	7.9	-
TOTAL:		100.0	

80 % Passing Size (µm) = 260



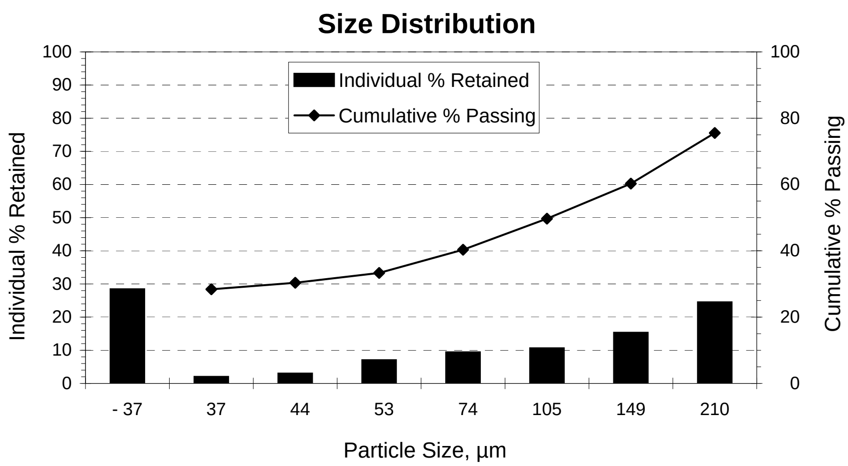
SIZE ANALYSIS REPORT

Client: Alexco
Test: C5
Sample: Cyanidation Residue
Grind: N/A

Date: 25-Feb-09
Project: 0901302

Sieve Size		Individual	Cumulative
Tyler Mesh	Micrometers	% Retained	% Passing
65	210	24.5	75.5
100	149	15.3	60.3
150	105	10.6	49.7
200	74	9.4	40.3
270	53	7.0	33.3
325	44	3.0	30.4
400	37	2.0	28.4
Undersize	- 37	28.4	-
TOTAL:		100.0	

80 % Passing Size (µm) = 225



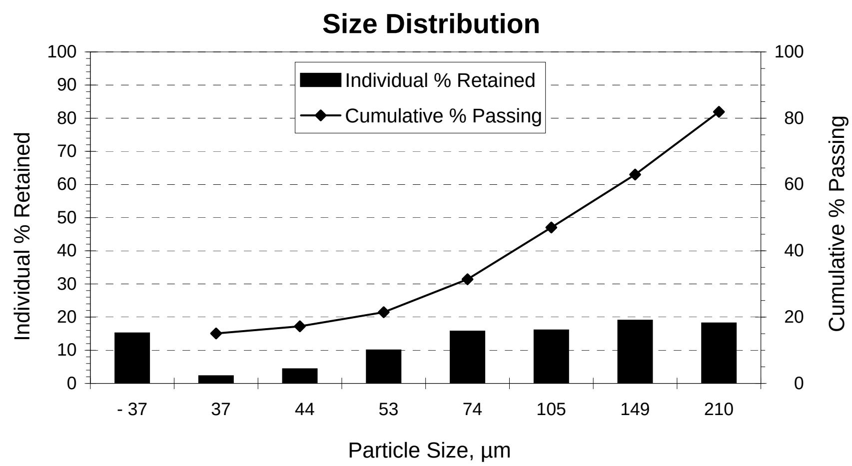
SIZE ANALYSIS REPORT

Client: Alexco
Test: C6
Sample: Cyanidation Residue
Grind: N/A

Date: 25-Feb-09
Project: 0901302

Sieve Size		Individual	Cumulative
Tyler Mesh	**Micrometers**	**% Retained**	**% Passing**
65	210	18.1	81.9
100	149	18.9	63.0
150	105	16.0	47.0
200	74	15.6	31.4
270	53	9.9	21.5
325	44	4.2	17.2
400	37	2.2	15.1
Undersize	- 37	15.1	-
TOTAL:		100.0	

80 % Passing Size (μm) = 204



Appendix B

Capital Cost Estimate

CAPITAL COST DETAIL ESTIMATE
Elsa Tailings Reprocessing - Heap Leach Case

Area	Account	Description	Quantity	Unit Cost	Installation Factor	Total Material	Total $
10		**MINING EQUIPMENT**					
	10-01	Production Drills			1.00	$ -	$ -
	10-02	Excavator 335	2	$325,000	1.00	$ 650,000	$ 650,000
	10-03	Front End Loader 960	1	$175,000	1.00	$ 175,000	$ 175,000
	10-04	Trucks	3	$135,000	1.00	$ 405,000	$ 405,000
	10-05	Grader			1.00	$ -	$ -
	10-06	Track Dozer	1	$325,000	1.00	$ 325,000	$ 325,000
	10-07	Rubber Tire Dozer			1.00	$ -	$ -
	10-08	Backhoe			1.00	$ -	$ -
	10-09	Water Truck	1	$65,000	1.00	$ 65,000	$ 65,000
	10-10	Fuel Truck	1	$75,000	1.00	$ 75,000	$ 75,000
	10-11	Service Truck	1	$75,000	1.00	$ 75,000	$ 75,000
	10-12	Crane			1.00	$ -	$ -
	10-13	Low Boy			1.00	$ -	$ -
	10-14	Blasting Truck/Silos			1.00	$ -	$ -
		Area 10 Subtotal				$ 1,770,000	$ 1,770,000
20		**MOBILE SUPPORT EQUIPMENT**					
	20-01	Pickups	4	$30,000	1.00	$ 120,000	$ 120,000
	20-02	Forklifts	1	$65,000	1.00	$ 65,000	$ 65,000
	20-03	Boom Truck	1	$75,000	1.00	$ 75,000	$ 75,000
	20-04	Ambulance Rescue Equipment			1.00	$ -	$ -
	20-05	Crew Van	1	$35,000	1.00	$ 35,000	$ 35,000
	20-06	Front End Loader 938			1.00	$ -	$ -
		Area 20 Subtotal				$ -	$ 295,000
30		**SITE PREPARATION & SERVICES**					
	30-01	Site Preparation	1	$650,000	1.00	$ 650,000	$ 650,000
	30-02	Water Supply & Distribution	1	$115,000	1.50	$ 115,000	$ 172,500
	30-03	Drainage Control	1	$125,000	1.00	$ 125,000	$ 125,000
	30-04	Fencing	1	$25,000	1.00	$ 25,000	$ 25,000

CAPITAL COST DETAIL ESTIMATE
Elsa Tailings Reprocessing - Heap Leach Case

Area	Account	Description	Quantity	Unit Cost	Installation Factor	Total Material	Total $
	30-05	Fuel Supply & Storage	1	$45,000	1.00	$ 45,000	$ 45,000
	30-06	Communications	1	$25,000	1.00	$ 25,000	$ 25,000
	30-07	Computer Services	0		1.00	$ -	$ -
	30-08	Monitoring Wells	3	$25,000	1.00	$ 75,000	$ 75,000
		Area 30 Subtotal				$ 1,060,000	$ 1,117,500
40		**PRE-PRODUCTION MINING**					
	40-01	Haul Road Construction	1	$225,000	1.00	$ 225,000	$ 225,000
	40-02	Pre-Production Mining					
		Area 40 Subtotal				$ 225,000	$ 225,000
	80-32	Concrete	0		1.00	$ -	$ -
	80-33	Bid Quotation	0		1.25	$ -	$ -
		Area 80 Subtotal				$ -	$ -
90		**MERRILL CROWE RECOVERY**					
	90-01	Carbon Adsorption Columns	0		1.00	$ -	$ -
	90-02	Preg Solution Tank	0		1.00	$ -	$ -
	90-03	Preg Feed Pump	0		1.00	$ -	$ -
	90-04	Barren Solution Tank	0		1.00	$ -	$ -
	90-05	Barren Solution Pump	0		1.00	$ -	$ -
	90-06	Strip Vessels	0		1.00	$ -	$ -
	90-07	Carbon Transfer Pump	0		1.00	$ -	$ -
	90-08	Carbon Sizing Screen	0		1.00	$ -	$ -
	90-09	Acid Wash Vessel	0		1.00	$ -	$ -
	90-10	Acid Tank	0		1.00	$ -	$ -
	90-11	Acid Wash Pump	0		1.00	$ -	$ -
	90-12	Fine Carbon Tank, Filter Press	0		1.00	$ -	$ -
	90-13	Carbon Transfer Tank	0		1.00	$ -	$ -
	90-14	Strip Feed Tank	0		1.00	$ -	$ -
	90-15	Strip Feed Pump	0		1.00	$ -	$ -
	90-16	Strip Heat Exchangers	0		1.00	$ -	$ -

CAPITAL COST DETAIL ESTIMATE
Elsa Tailings Reprocessing - Heap Leach Case

Area	Account	Description	Quantity	Unit Cost	Installation Factor	Total Material	Total $
	90-17	Strip Solution Heater	0		1.00	$ -	$ -
	90-18	Kiln Feed Tank	0		1.00	$ -	$ -
	90-19	Reactivation Kiln	0		1.00	$ -	$ -
	90-20	Reactivation Quench Tank	0		1.00	$ -	$ -
	90-21	Electrowinning Cells	0		1.00	$ -	$ -
	90-22	Electrowinning Cells Return Pump	0		1.00	$ -	$ -
	90-23	Cathode Washer	0		1.00	$ -	$ -
	90-24	Sludge Filter Press	0		1.00	$ -	$ -
	90-25	Rectifier	0		1.00	$ -	$ -
	90-26	Area Sump Pump	0		1.00	$ -	$ -
	90-27	Crane	0		1.00	$ -	$ -
	90-28	Process Piping	0		1.00	$ -	$ -
	90-29	Structural Steel & Building	0		1.00	$ -	$ -
	90-30	Electrical & Instrumentation	1	$225,000	1.25	$ 225,000	$ 281,250
	90-31	Concrete	300	$1,500	1.25	$ 450,000	$ 562,500
	90-32	Merrill Crowe plant equipment package	1	$3,750,000	1.50	$ 3,750,000	$ 5,625,000
		Area 90 Subtotal				$ 4,425,000	$ 6,468,750
100		**<u>REFINING</u>**					
	100-01	Furnace	0		1.00	$ -	$ -
	100-02	Furnace Exhaust System	0		1.00	$ -	$ -
	100-03	Refinery Platework	0		1.00	$ -	$ -
	100-04	Flux Storage Hopper	0		1.00	$ -	$ -
	100-05	Mercury Retort	0		1.00	$ -	$ -
	100-06	Crane	0		1.00	$ -	$ -
	100-07	Area Sump Pump	0		1.00	$ -	$ -
	100-08	Refinery Vault	1	$65,000	1.25	$ 65,000	$ 81,250
	100-09	Refinery Security System	1	$50,000	1.25	$ 50,000	$ 62,500
	100-10	Process Piping	0		1.00	$ -	$ -
	100-11	Structural Steel & Building	0		1	$ -	$ -
	100-12	Electrical & Instrumentation	1	$25,000	1.25	$ 25,000	$ 31,250
	100-13	Concrete	0	$250	1.25	$ -	$ -
	100-14	Bid Quotation			1.25	$ -	$ -

CAPITAL COST DETAIL ESTIMATE
Elsa Tailings Reprocessing - Heap Leach Case

Area	Account	Description	Quantity	Unit Cost	Installation Factor	Total Material	Total $
		Area 100 Subtotal				$ 140,000	$ 175,000
110		**PADS & PONDS**					
	110-01	Leach Pad Containment Dike	1	$250,000	1.00	$ 250,000	$ 250,000
	110-02	Leach Pad Liner System	105,000	$45	1.00	$ 4,725,000	$ 4,725,000
	110-03	Barren Solution Pond	1	$350,000	1.00	$ 350,000	$ 350,000
	110-04	Preg Solution Pond	1	$350,000	1.00	$ 350,000	$ 350,000
	110-05	Overflow Pond	1	$225,000	1.00	$ 225,000	$ 225,000
	110-06	Preg Solution Collection Piping	1	$75,000	1.25	$ 75,000	$ 93,750
	110-07	Barren Solution Piping	1	$75,000	1.25	$ 75,000	$ 93,750
	110-08	Barren Pond Pump	1	$7,500	1.25	$ 7,500	$ 9,375
	110-09	Preg Pond Pump	1	$7,500	1.25	$ 7,500	$ 9,375
	110-10	Overflow Pond Pump	2	$7,500	1.25	$ 15,000	$ 18,750
	110-11	Wildlife Netting	0		1.00	$ -	$ -
	110-12	Leach Pad Piping System	1	$125,000	1.00	$ 125,000	$ 125,000
		Area 110 Subtotal				$ 6,205,000	$ 6,250,000
120		**LEACH PAD STACKING**					
	120-01	Grasshopper Conveyors	1	$300,000	1.10	$ 300,000	$ 330,000
	120-02	Radial Stacker	0		1.00	$ -	$ -
		Area 120 Subtotal				$ 300,000	$ 330,000
130		**REAGENTS**					
	130-01	Cyanide Mix Tank	1	$12,500	1.00	$ 12,500	$ 12,500
	130-02	Cyanide Mix Pump	1	$3,500	1.00	$ 3,500	$ 3,500
	130-03	Cyanide Feed Pump	1	$3,500	1.00	$ 3,500	$ 3,500
	130-04	Caustic Mix Tank	1	$12,500	1.00	$ 12,500	$ 12,500
	130-05	Caustic Mix Pump	1	$3,500	1.00	$ 3,500	$ 3,500
	130-06	Caustic Feed Pump	1	$3,500	1.00	$ 3,500	$ 3,500
	130-07	Flocculant Tank	0		1.00	$ -	$ -
	130-08	Antiscalant Tank	1		1.00	$ -	$ -
	130-09	Antiscalant Pump	1	$1,500	1.00	$ 1,500	$ 1,500
	130-10	Lime Addition Silo	1	$50,000	1.25	$ 50,000	$ 62,500

CAPITAL COST DETAIL ESTIMATE
Elsa Tailings Reprocessing - Heap Leach Case

Area	Account	Description	Quantity	Unit Cost	Installation Factor	Total Material	Total $
	130-11	Propane Storage Tank	1	$35,000	1.00	$ 35,000	$ 35,000
	130-12	Process Air Compressor Package	1	$7,500	1.25	$ 7,500	$ 9,375
	130-13	Process Piping	1	$125,000	1.25	$ 125,000	$ 156,250
	130-14	Structural Steel & Building	1	$350,000	1.25	$ 350,000	$ 437,500
	130-15	Electrical & Instrumentation	1	$15,000	1.25	$ 15,000	$ 18,750
	130-16	Concrete	0		1.00	$ -	$ -
	130-17	Bid Quotation	0	$0	1.25	$ -	$ -
		Area 130 Subtotal				$ 623,000	$ 759,875
150		**WATER TREATMENT**					
	150-01	Cyanide Destruction	1	$125,000	1.25	$ 125,000	$ 156,250
	150-02	Metals Removal	0		1.00	$ -	$ -
		Area 150 Subtotal				$ 125,000	$ 156,250
160		**POWER SUPPLY & DISTRIBUTION**					
	160-01	Diesel Generators	0	$0	1.20	$ -	$ -
	160-02	Electrical Distribution	1	$550,000	1.20	$ 550,000	$ 660,000
	160-03	Electrical Substation	0		1.00	$ -	$ -
	160-04	Lighting / Cabling	1	$15,000	1.20	$ 15,000	$ 18,000
	160-05	Power Line Extension	0	$0	1.00	$ -	$ -
		Area 160 Subtotal				$ 565,000	$ 678,000
170		**ASSAY LABORATORY**					
	170-01	Building	1000	$125	1.00	$ 125,000	$ 125,000
	170-02	Equipment	1	$125,000	1.00	$ 125,000	$ 125,000
	170-03	Concrete	75	$1,250	1.25	$ 93,750	$ 117,188
		Area 170 Subtotal				$ 343,750	$ 367,188
		TOTAL DIRECT COSTS				$ 16,281,750	$ 19,092,563

CAPITAL COST DETAIL ESTIMATE
Elsa Tailings Reprocessing - Heap Leach Case

Area	Account	Description	Quantity	Unit Cost	Installation Factor	Total Material		Total $
195		**FREIGHT**	3%	Equipment Costs			$	488,453
200		**EPCM**	7%	Installed Equipment Cost			$	1,336,479
210		**START UP INVENTORY**						
	210-01	Cyanide	37500	$2.10			$	78,750
	210-02	Lime	208000	$0.54			$	112,320
	210-10	Warehouse Inventory	1.5%	Equipment Cost			$	244,226
220		**WORKING CAPITAL**	1	$0			$	-
230		**CONSTRUCTION INDIRECTS**	5%	Installed Equipment Costs			$	954,628
240	**240-01**	**CONTINGENCY**	30%	Installed Equipment Cost			$	5,727,769
		TOTAL INDIRECT COSTS					$	8,942,625
		TOTAL PROJECT COSTS					$	28,035,188

Appendix C

Operating Cost Estimate

DETAILED OPERATING COSTS
VALLEY TAILINGS REPROCESSING

Annual Ore Tonnage Case 550,000 mtpy

Cost Detail	Total		$/tonne
Process/Manpower			
Total Manpower	**$**	**2,872,090**	$5.22
Maintenance Labor			$0.00
Reagents	**$**	**2,873,149**	$5.22
Water Supply	$	55,000	$0.10
Light Vehicles	$	72,000	$0.13
ADR Maintenance Supplies	$	293,612	$0.53
Crusher Op/Maint Supplies			$0.00
Misc. Operating Supplies	$	175,000	$0.32
Metallurgical Testing	$	25,000	$0.05
Assay Laboratory	$	244,230	$0.44
Subtotal	$	6,610,081	$12.02
Mining			
Contract Mining Direct	**$**	**-**	$0.00
Technical Services Salaries			$0.00
Operating Labor			$0.00
Maintenance Labor			$0.00
Drilling	$	-	$0.10
Blasting	$	-	$0.08
Load, Haul	$	1,809,725	$3.29
Light Vehicles			$0.00
Miscellaneous Maintenance Supplies	$	45,000	$0.08
Miscellaneous Operating Supplies	$	45,000	$0.08
Subtotal	$	1,899,725	$3.45
G&A			
Administration Salaries			$0.00
Camp Accomodations ($50/manday)	**$**	**202,500**	$0.37
Roads and Yards	$	35,000	$0.06
Building Maintenance	$	10,000	$0.02

DETAILED OPERATING COSTS
VALLEY TAILINGS REPROCESSING

Annual Ore Tonnage Case 550,000 mtpy

Cost Detail	Annual Total	/tonne
Light Vehicles	$ 36,000	$0.07
Insurance	$ 35,000	$0.06
Safety Supplies (3% Labor Costs)	**$ 86,163**	$0.16
Employee Transportation	$ 24,000	$0.04
Office Supplies	$ 5,000	$0.01
Consulting		$0.00
Corporate Overhead	$ 125,000	$0.23
Miscellaneous	$ 100,000	$0.18
Subtotal	$ 658,663	$1.20
Environmental		
Environmental Salaries		**$0.00**
Reclamation Accrual	$ 1,375,000	$2.50
Light Vehicles		$0.000
Sampling/Compliance	$ 75,000	$0.14
Subtotal Environmental	$ 1,450,000	$2.636
Power Supply		
Diesel	**$ -**	$0.00
Maintenance	**$ -**	$0.00
Subtotal Power Supply	$ 290,045	$0.53
Total Annual Operating Costs	$ 10,908,513	$19.83

MANPOWER SUMMARY

POSITION	Number Required	Seasonal Base Cdn $	% Burden	Annual Total Cdn $
ADMINISTRATION				
General Manager		$ -	30%	$ -
Executive Assistant		$ -	30%	$ -
Administrative Manager		$ -	30%	$ -
Personnel Coordinator		$ -	30%	$ -
Senior Accountant		$ -	30%	$ -
Accountant	1	$ 39,000	30%	$ 50,700
Payroll Clerk	1	$ 27,000	30%	$ 35,100
Purchasing/Logistics Coordinator		$ -	30%	$ -
Buyer/Warehouseman		$ -	30%	$ -
Health & Safety Coordinator		$ -	30%	$ -
First Aid Attendant		$ -	30%	$ -
Security Attendant		$ -	30%	$ -
Total Administration	**2**			**$ 85,800**
MINING				
Mine Manager	1	$ 125,000	30%	$ 162,500
Operations General Foreman		$ -	30%	$ -
Operations Foreman		$ -	30%	$ -
Equipment Trainer		$ -	30%	$ -
Drillers		$ -	30%	$ -
Blaster		$ -	30%	$ -
Shovel/Loader Operator		$ -	30%	$ -
Truck Drivers		$ -	30%	$ -
Equipment Operators	17	$ 39,000	30%	$ 861,900
Subtotal Mining Operations	**18**			**$ 1,024,400**
Maintenance General Foreman		$ -	30%	$ -
Maintenance Foreman	1	$ 85,000	30%	$ 110,500
Maintenance Planner		$ -	30%	$ -
Master Mechanics/Foreman		$ -	30%	$ -
Mechanics	2	$ 50,700	30%	$ 131,820
Lube Serviceman		$ -	30%	$ -
Subtotal Mining Maintenance	3			$ 242,320
Total Mining	**21**			**$ 1,266,720**
TECHNICAL SERVICES				
Technical Services Manager		$ -	30%	$ -
Senior Mining Engineer		$ -	30%	$ -
Senior Geologist		$ -	30%	$ -
Mine Engineer		$ -	30%	$ -
Geologist		$ -	30%	$ -
Pit Technician		$ -	30%	$ -
Survey/Mine Technician	1	$ 39,000	30%	$ 50,700
Subtotal Mine Technical Services	1			**$ 50,700**
Environmental Coordinator		$ -	30%	$ -
Environmental Technician	1	$ 52,000	30%	$ 67,600
Subtotal Environmental	1			$ 67,600
Total Technical Services	**2**	$ -		**$ 118,300**
PROCESS				
Process Manager		$ -	30%	$ -
Chief Metallurgist		$ -	30%	$ -

MANPOWER SUMMARY

POSITION	Number Required	Seasonal Base Cdn $	% Burden	Annual Total Cdn $
Metallurgist	1	$ 105,000	30%	$ 136,500
Clerk		$ -	30%	$ -
Chief Assayer		$ -	30%	$ -
Assayer	2	$ 39,000	30%	$ 101,400
Sample Prep	1	$ 36,000	30%	$ 46,800
Met Technician		$ -	30%	$ -
Subtotal Process Technical	4			**$ 284,700**
Process General Foreman		$ -	30%	$ -
Foreman	1	$ 75,000	30%	$ 97,500
Crusher Operator		$ -	30%	$ -
Rover		$ -	30%	$ -
Mill Control Operator		$ -	30%	$ -
Mill Helper	2	$ 39,000	30%	$ 101,400
ADR Operator	6	$ 46,800	30%	$ 365,040
ADR Assistant		$ -	30%	$ -
Leach Pad Operator	3	$ 39,000	30%	$ 152,100
Refiner		$ -	30%	$ -
Subtotal Process Operations	12			**$ 716,040**
General Foreman		$ -	30%	$ -
Foreman		$ -	30%	$ -
Maintenance Planner		$ -	30%	$ -
Surface Support		$ -	30%	$ -
Crusher Maintenance Personnel		$ -	30%	$ -
Mill Maintenance Personnel	3	$ 50,700	30%	$ 197,730
ADR Maintenance Personnel		$ -	30%	$ -
Electrical Maintenance Personnel	1	$ 65,000	30%	$ 84,500
Equipment Operator		$ -	30%	$ -
Subtotal Process Maintenance	4			**$ 282,230**
Total Process	**20**			**$ 1,282,970**
TOTAL MANPOWER	45			**$ 2,872,090**
Total Tonnes Ore per Year	550,000			
Total Manpower $/tonne	$ 5.01			

PROCESS CONSUMABLES

Annual Tonnage Case	550,000	mtpy
Annual Production	1,511,888	Ounces
Flow Rate	250	m3/hr

Category	Consumption Rate	Unit	Annual Consumption		Unit $	Annual $Cdn	$/tonne
Reagents							
Cyanide	0.75	kg/tonne	412,500	kg	$ 2.00	$ 825,000	$ 1.50
Lime	4.00	kg/tonne	2,200,000	kg	$ 0.54	$ 1,188,000	$ 2.16
Filter Aid	0.10	kg/tonne	55,000	kg	$ 0.80	$ 44,000	$ 0.08
Pre Coat	0.10	kg/tonne	55,000	kg	$ 0.80	$ 44,000	$ 0.08
Propane	0.25	lit/ounce	377,972	kg	$ 0.35	$ 132,290	$ 0.24
Refinery Fluxes	3.00	kg/ounce	1,650,000	kg	$ 0.05	$ 82,500	$ 0.15
Zinc Dust	0.50	kg/tonne	275,000	kg	$ 2.00	$ 550,000	$ 1.00
Antiscalant	7.00	ppm/m3 sol	15,330	kg	$ 0.48	$ 7,358	$ 0.01
Total						$ 2,873,149	$ 5.22

MINE EQUIPMENT PRODUCTIVITY & SIZING CALCULATIONS

Category	Unit	
Production Statistics		
Annual Tailings Processed	tonnes	550000
Loose Density	t/m3	1.6
Operating Days per Year	days	150
Crew Rotation	on/off	7/7
Operating Hours per Year/Employee	hours	2184
Loading - Excavator		
Excavator Availability	%	85%
Excavator Utilization	%	90%
Struck Bucket Capacity	m3	0.75
Struck Bucket Capacity	tonnes	1
Passes per Truck Calculated		25.00
Swing Cycle Time	sec	15
Truck Spotting Time	sec	15
Excavator Capacity	m3/h	90
Excavator Capacity	tph	144
Loading Time Minutes per Truck		12.5
Individual Loading Capacity	tpd	2644
Total Annual Loader Hours	hours	3819
Annual Loading Capacity - Single Unit	tonnes	396576
Total Loaders Required Calculated		1.39
Total Loaders Required Actual		2.0
# Loader Operators Required		4.0
Haulage		
Haul Truck Capacity	tonnes	30
Truck Availability	%	85%
Truck Utilization	%	90%
Haul Distance Roundtrip	km	3
Pad Load Haul Distance Roundtrip	km	0.25
Turn, Spot, Dump Time	min	1
Average Truck Speed	km/h	20
Ore Haulage Time	min	9.0
Pad Load Haulage Time	min	0.75
Total Ore Cycle Time per Truck	min	22.5
Total Pad Load Cycle Time	min	13.3
Haul Truck Capacity Ore	tph	61
Haul Truck Capacity Pad Load	tph	104
Haul Truck Capacity Ore	tpd	1469
Haul Truck Capacity Pad Load	tpd	2494
Annual Tonnes	tpd	3667
Annual Tonnes	tph	153
# Haul Trucks Required Ore		2.50
Total Haul Trucks Required		3.00
Total Truck Operating Hours per Year	hours	8987
Total Haul Truck Drivers Required		8
Dozing		
Dozing Availability	%	80%
Dozing Utilization	%	70%
# of Dozers		1
Annual Dozer Hours	hours	2016
Total Dozer Operators Required		4
Graders		
Grader Availability	%	80%
Grader Utilization	%	50%
# of Graders		1
Annual Grader Hours	hours	1440
Total Grader Operators Required		1

MINE EQUIPMENT PRODUCTIVITY & SIZING CALCULATIONS

Operating Cost Summary

Diesel Cost $/liter $ 0.95

Unit Tire Cost $ 3,500

Load Haul Hours	Fuel/Supplies $/hr	Annual Op. Hours	Fuel liter/h	Annual Fuel Costs	Tire Life Hours	Annual Tire Cost	Undercarr. $/hr	Annual Undercarr.	Repair Reserve $/hr	Annual Repair Reserve
Annual Truck Hours	87.00	8987	65	$ 554,943	4000	$ 47,181			$ 20.00	$ 179,739
Annual Loader Hours	103.55	3819	85	$ 308,420	5000	$ 10,694			$ 20.00	$ 76,389
Annual Dozer Hours	81.75	2016	65	$ 124,488			$ 14.06	$ 28,350	$ 20.00	$ 40,320
Annual Grader Hours	53.65	1440	31	$ 42,408	5000	$ 6,048			$ 20.00	$ 28,800
Subtotals				$ 1,030,259		$ 63,924		$ 28,350		$ 325,247
Total Fuel, Maintenance Supplies	$ 1,447,780									
Contingency Factor 25%										
Total Load/Haul/Place	1,809,725									

Appendix D-1

Historic Tailings Reprocessing Assessment Reports

For Inter-office Correspondence Only

UNITED KENO HILL MINES LIMITED

TO T. P. Riordon

ADDRESS

IN REPLY TO YOURS OF

FROM C. Thomas

ADDRESS

DATE January 8, 1987

SUBJECT METAL CONTENT OF TAILINGS

Last summer I got the two summer students to go through the annual reports and calculate the metal content of the tailings deposited in the tailings disposal area below Elsa.

They used the mill head tons and grade for each individual year together with the reported recoveries and or the concentrate tons and grade to estimate the tons and grades that went into the tailings. Their results were tabulated using the computer and are attached to this report.

Also taken into consideration were the years that the cyanide plant operated, the metals leached out of the tailings by runoff water by using metal levels and flows in the final effluent, estimated tonnages used for hydraulic fill at Husky Mine and estimated tonnages of slimes which were considered untreatable.

Taking all the above into account, the metal contents in the tailings considered by me to be treatable are as follows:

3.2 million tons @ 3.92 oz./ton Ag, 1.17% Pb, 0.79% Zn

Metal Content: 12,544,000 oz. Ag

74,880,000 lbs. Pb

50,560,000 lbs. Zn

Value of metals using: \$6.43 oz. Ag (CDN), \$ 0.36 Lb. Pb, \$ 0.69 Lb. Zn

Silver	\$80,657,920
Lead	\$26,956,800
Zinc	\$34,886,400
TOTAL VALUE ALL METALS	\$142,501,120

INITIALS OF ADDRESSEE DATE INITIALS OF SENDER

The tailings disposal area covers an area of approximately 170 acres with depth of tailings ranging from 2 ft. to 15 ft. Being spread out over such a large area and having been deposited for a long time has created an advantage to reprocessing the tailings as it appears that most of the slimes have been washed out and have been redeposited behind the No. 1 Tailings dam.

In an effort to produce a plausible method to reprocess the tailings and to find a site to deposit the reprocessed tailings the following is the route I would follow if a feasibility plan was to be initiated.

With reference to the attached sketch, just to the north of the present tailings disposal area is a swamp bounded by small hills. To make sufficient room to hold the reprocessed tailings dykes would have to be built up at the swamp outlets, although initially there is sufficient room to dump, and the dykes could be built up over the life of the project.

Using a fleet of three (3) large wheel tractor-scrapers on the dry tailings and a suction dredge on the saturated tailings to feed a reprocessing plant of 5,000 tons per day the tailings could be processed in approximately 7 years (the plant would operate only for the 3 summer months).

The reprocessing plant could be an open air plant situated on the tailings area consisting of a screening plant to remove logs and other debris, to de-sliming cyclones then to a gravity pre-concentrating plant using tables and spirals etc. If the pre-concentrating plant concentrated the tails 10 times, the 500 tons/day product could go to cyanide leaching tanks and thickeners with the pregnant solution being pumped up to the Elsa Mill cyanide plant's crowe towers and precipiating equipment and presses. The precipitate could be reduced to dore bars in the Elsa refinery to produce a small product and therefore reduce shipping charges. The barren solution could be treated using a sulphur burning $SO_2$2 plant to kill the cyanide.

Another advantage that should be considered is that the tailings would be redeposited into a much smaller isolated area which might have to be done anyway should the mine eventually be abandoned. At present with the tailings being spread over a large area, any abandonment plan would involve excavating long diversion ditches to lead run-off water away from the tailings to reduce leaching. Such a plan might not be acceptable to the Water Board, but if the tailings could be contained in the small area as described, the tailings would be completely isolated with little possibility of leaching. The cleaned off present tailings area could be easily re-vegetated if the original surface was exposed leaving it much more aesthetically acceptable.

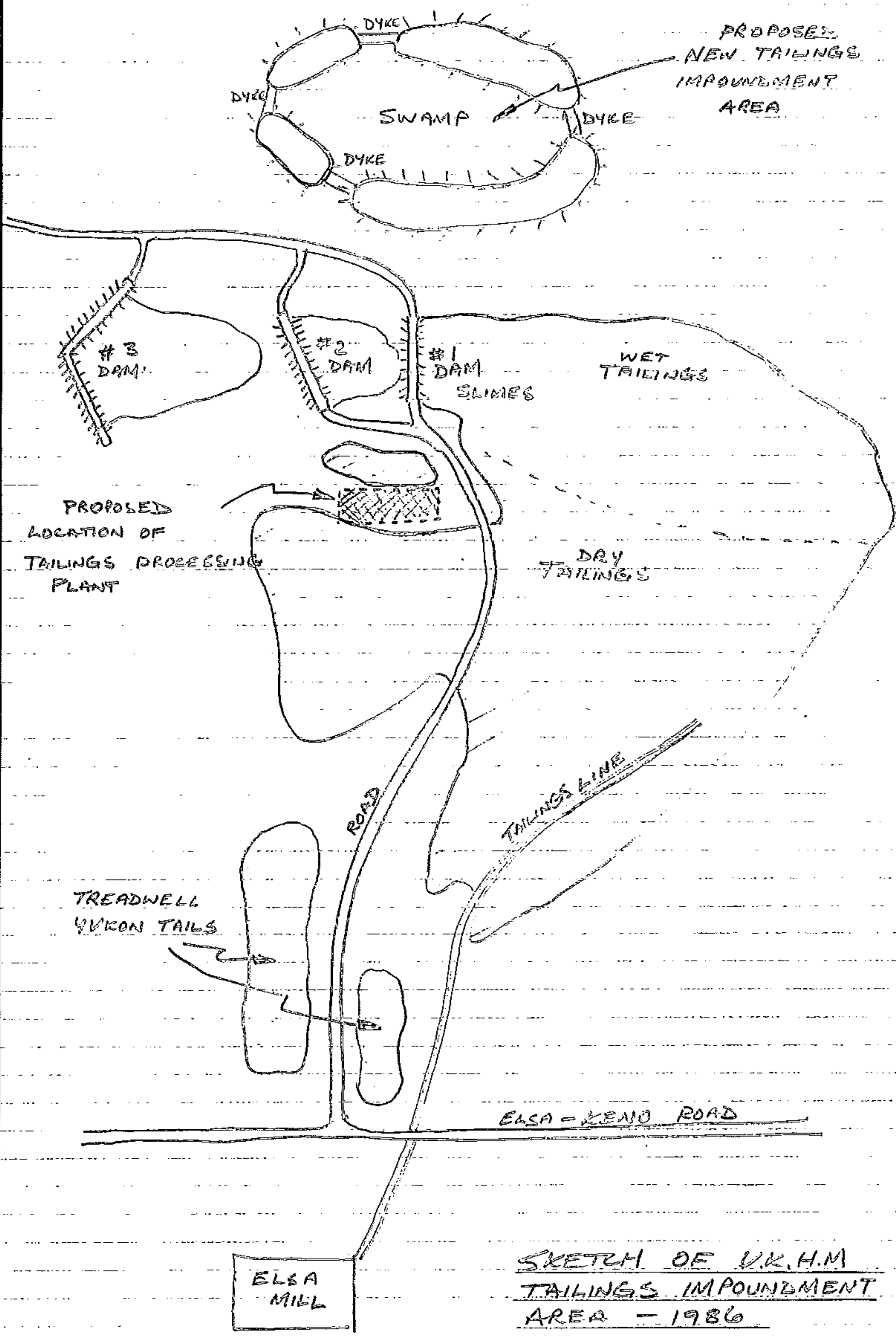
Another possibility is that some of the coarse sand waste could be trucked on the back haul to the outlying mines where a mobile re-pulper could re-pulp the sand and pump it into the stopes for fill, reducing the present practice of mining waste for fill.

Conclusions and Recommendations

Due to the large value of the metals contained in our tailings a serious attempt (costing $50,000 to 75,000) should be initiated in 1987 to confirm the tons & grades, perform metallurgical tests and engineering studies leading to a feasibility evaluation later in the year.

Initially, we could hire two summer students to drill test holes using a hand held gas powered auger to sample, provide tailings depths and to make up a good average bulk sample. Also, we could immediately order a new orthographic 1" = 400 ft. Topographic map with 5 ft. contour intervals based on the 1984 aerial photography to cover the tailings area and the proposed new tailings area which is presently on the border of our topographic map coverage. This map has to be ordered anyway and is budgeted for 1987 at $7,000.

With a good average bulk sample we should hire an independent engineering consulting firm who specialize in such things to perform the metallurgical tests and write a feasibility evaluation.



SKETCH OF U.K.H.M TAILINGS IMPOUNDMENT AREA - 1986

N.T.S. E.R. Thomas

YEAR	TONS MILLED	MILL HEAD GRADE Ag oz	Pb %	Zn %	MILL HEAD METALS Ag oz	Pb lbs	Zn lbs	% recovery CONCENTRATE METALS Ag oz	Pb lbs	Zn lbs	TO TAILINGS - METALS Ag oz	Pb lbs	Zn lbs	TAILS TONS
1936 to 1941	253,764	62.39	6.90	2.80	15,832,336	35,019,432	14,210,787	12,509,163	22,486,844	8,419,552	3,323,173	12,532,588	5,791,232	326,220
1947	20,880	43.40	7.18	—	906,192	2,998,368	—	75.10% 680,461	61.20% 1,835,942	—	225,731	1,162,426	—	18,572
1948	37,593	57.32	8.06	—	2,154,831	6,059,991	—	84.25% 1,816,437	76.59% 4,643,909	—	338,394	1,416,082	—	31,954
1949	29,494	51.18	9.87	4.29	1,509,503	5,822,116	2,503,585	85.39% 1,288,951	78.66% 4,579,902	89.44% 2,263,392	220,552	1,242,214	267,193	23,438
1950	77,465	51.36	10.99	5.48	3,978,602	17,026,807	8,490,164	82.89% 3,298,054	79.87% 13,598,760	82.72% 7,022,820	680,545	3,428,047	1,467,344	63,248
1951	88,026	49.22	9.81	4.70	4,332,640	17,270,701	8,274,414	79.97% 3,464,646	75.64% 13,063,588	86.98% 7,196,832	867,993	4,207,113	1,077,611	73,794
change of fiscal year (9 months) 1952	102,269	35.70	8.77	6.46	3,651,003	17,937,982	13,213,154	93.11% 3,399,630	80.48% 14,436,073	77.80% 10,280,241	251,373	3,501,909	2,932,913	84,247
1953	156,684	40.92	9.45	7.47	6,411,509	29,613,276	23,408,589	96.43% 6,182,442	92.23% 27,313,584	90.76% 21,245,493	229,067	2,299,692	2,163,096	112,829
1954	180,249	34.67	9.01	7.98	6,249,232	32,480,870	28,767,740	94.08% 5,878,791	94.36% 30,663,549	90.81% 26,134,700	370,441	1,817,320	2,633,040	137,849
1955	162,307	37.08	9.51	8.21	6,018,343	27,624,651	26,650,909	95.14% 5,725,852	95.95% 26,505,853	91.71% 24,441,457	292,491	1,118,798	2,209,352	123,978
1956	155,702	38.17	8.44	8.34	5,943,145	26,407,059	25,971,093	93.93% 5,582,979	94.92% 25,085,145	92.77% 24,107,851	360,166	821,080	1,863,242	117,885
1957	159,835	38.65	7.56	6.39	6,179,555	24,174,617	20,433,[illegible]	92.16% 5,694,850	93.40% 22,569,908	88.63% 18,119,454	484,705	1,604,704	2,313,849.	128,974
1958	175,058	36.83	7.18	6.10	6,447,386	25,138,328	21,357,076	92.82% 5,984,373	88.52% 22,255,501	87.08% 18,610,970	463,013	2,882,827	2,746,106	144,700
1959	173,477	44.11	7.42	5.88	7,652,070	25,743,987	20,4[illegible]5	95.50% 7,307,815	88.74% 22,864,276	86.78% 17,717,019	344,255	2,878,710	2,683,876	142,000
1960	176,745	43.35	7.25	4.80	7,661,895	25,628,025	16,967,520	94.62% 7,249,160	85.75% 21,986,887	85.75% 14,440,744	412,795	3,641,138	2,526,746	149,000
1961	186,116	41.16	5.83	4.84	7,660,534	21,701,126	18,016,028	94.40% 7,231,980	82.54% 17,911,672	86.07% 15,512,624	428,554	3,789,453	2,503,404	159,000
1962	184,123	40.55	5.84	4.42	7,466,187	21,505,566	16,276,473	93.77% 7,000,837	81.77% 17,587,767	85.33% 13,885,884	465,350	3,917,799	2,390,589.	15,900
1963	186,721	34.03	5.44	4.69	6,354,115	20,315,244	17,514,429	94.08% 5,978,075	82.46% 16,751,012	84.27% 14,759,821	376,040	3,564,232	2,754,608	158,397
15 months 1964	227,845	33.37	6.38	4.92	7,603,187	29,073,022	22,419,948	95.63% 7,270,911	90.48% 26,304,902	89.05% 19,965,295	332,276	2,768,120	2,454,653	190,083
					114,012,265			103545076	352444029					

YEAR	TONS MILLED	MILL HEAD GRADE Ag oz	Pb %	Zn %	MILL HEAD METALS Ag oz	Pb lbs	Zn lbs	CONCENTRATE METALS Ag oz	Pb lbs	Zn lbs	TO TAILINGS - METALS Ag oz	Pb lbs	Zn lbs	TAILS TONS
1965	146,850	33.25	7.06	6.22	4,882,762	20,735,220	18,268,140	96.29% 4,701,820	90.44% 18,753,650	91.42% 16,700,565	180,942	1,981,570	1,567,545	119,585
1966	120,374	36.56	7.60	5.61	4,400,873	18,296,848	13,505,962	96.25% 4,235,678	90.99% 16,647,849	88.85% 11,999,953	165,195	1,648,999	1,506,009	73,266
1967	106,189	37.71	7.97	5.89	4,004,387	16,926,526	12,509,064	95.01% 3,804,644	91.39% 15,469,569	86.91% 10,872,074	199,743	1,456,957	1,636,990	51,619
1968	60,800	33.93	6.53	5.55	2,062,944	7,940,480	6,748,800	94.70% 1,981,777	94.20% 7,418,645	88.6% 6,212,589	81,167	521,835	536,211	50,400
1969	87,483	27.98	4.56	4.67	2,447,774	7,978,449	8,170,912	95.70% 2,405,615	96.40% 7,719,096	90.8% 7,845,682	42,159	259,353	325,230	74,376
1970	93,215	27.3	4.07	5.28	2,544,769	7,587,701	9,843,504	95.50% 2,430,254	96.70% 6,946,332	92.0% 8,155,894	114,515	641,369	1,687,610	80,111
1971	94,754	3057	5.17	5.19	2,896,630	9,797,564	9,835,465	96.89% 2,806,567	90.17% 8,834,831	76.45% 7,518,885	90063	962,733	2,316,580	81,480
1972	80,646	34.23	4.61	3.19	2,760,513	7,435,561	5,145,215	95.42% 2,634,176	95.57% 7,106,437	77.11% 3,967,611	126,337	329,124	1,177,604	72,209
1973	94,819	37.91	4.38	1.73	3594588	8,306,144	3,280,737	94.87% 3,410,045	94.87% 7,879,855	71.68% 2,351,650	184,543	426,289	929,087	86,633
1974	93,232	37.73	4.22	1.15	3,517,643	7,868,781	2,144,336	94.88% 3,337,383	91.76% 7,220,476	65.84% 1,411,750	180,260	648,305	732586	85,866
1975	90,860	34.96	4.03	1.15	3,176,466	7,323,316	2,089,780	94.21% 2,992,420	89.89% 6,582,728	72.38% 1,512,675	184,046	740,588	577,105	84,024
1976	75,515	35.49	4.02	1.17	2680027	6071406	1767051	93.31% 2,500,752	86.50% 5,251,997	74.66% 1,319,243	179,275	819,409	447,808	70,451
1977	91,486	35.49	4.57	1.12	3246838	8,361,820	2,049,286	89.76% 2914,406	75.04% 6,274,710	72.76% 1491,150	332,432	2087,228	558136	84,874
1978	90,082	35.72	5.51	0.79	3,217,729	9927036	1,423,296	90.67% 2,917,456	76.59% 7603472	75.42% 1073449	300273	2323564	349,846	82,559
1979	124,322	23.87	3.67	0.60	2967566	9,125,235	1,491,864	85.50% 2536975	65.88% 6,011,617	56.03% 835,871	430591	3113618	655993	117,140
1980	87,784	22.96	3.39	0.79	2,015,521	5,951,755	1386987	85.94% 1732086	57.57% 3426648	36.51% 506,441	283435	2525107	880546	82,589
1981	66,924	21.88	3.59	0.64	1,464,297	4,805,143	856,627	80.36% 1,103,752	47.22% 2,265,950	32.18% 275,731	360,545	2,539,193	580,890	63,033
* 1982	55,491	24.63	2.99	0.67	1,366,743	3,318,361	743,579	85.99% 1,175,327	59.00% 1,957,832	35.30% 262,483	191,416	1,360,534	481,096	51,586
1983	32,628	22.79	2.56	0.49	743,592	1,670,553	319,754	86.09% 640,155	63.90% 1,067,478	43.81% 139,245	103,437	603,055	180,509	30,232

YEAR	TONS MILLED	MILL HEAD GRADE Ag oz	Pb %	Zn %	MILL HEAD METALS Ag oz	Pb lbs	Zn lbs	CONCENTRATE METALS Ag oz	Pb lbs	Zn lbs	TO TAILINGS . METALS Ag oz	Pb lbs	Zn lbs	
1984	72,409	19.74	2.38	0.32	1,429,353	3,446,668	463,417	87.32% 1,247,937	67.25% 2,313,350	54.41% 250,881	181,416	1,133,318	212,536	67,287
1985	74,609	20.91	2.36	0.25	1,560,074	3,521,544	373,045	83.02% 1,295,288	60.15% 2,115,713	37.37% 139,687	264,786	1,405,831	233,358	68,813
TOTAL	4,574,875	36.74	6.43	4.26	170,993,354	587,937,274	392,204,539	156,349,590	501,812,234 502,665,136	264,124,149	14,643,590	86,671,134	58,501,685	3,680,183
											14,643,764	86,624,990	- %	
											-174			

YEAR	TONS MILLED	MILL HEAD GRADE Ag oz	Pb %	Zn %	MILL HEAD METALS Ag oz	Pb lbs	Zn lbs	CONCENTRATE METALS Ag oz	Pb lbs	Zn lbs	TO TAILINGS METALS Ag oz	Pb lbs	Zn lbs
1936 to 1941	253764 (TAILS 226,220)	62.39	6.90	2.80	15832335.96	35019,432.00	14210,784.00	12,509,163	22,486,844	841,9552	3323172	12532588	5791,232
from 48' report 1947	20,880 tons	43.40	7.18	N/A	906,192.00	2,998,368.00	N/A	75.10 680,461.28	61.20 1,835,942	N/A	225,730.72	1,162,426.00	N/A
1948	37,593 Tons	57.32	8.06	N/A	2,154830.75	6,059991.60	N/A	84.25 1,816,436.52	76.59 4,643,909.0	N/A	338,394.23	1,416082.60	N/A
1949													
1950	see back												
1951													
1952													
~~1953~~	~~156684~~	~~40.92~~	~~9.45~~	~~7.47~~				c 405,695 ~~5,846,787~~	— ~~27313584~~	~~21,245,493~~			
1954	180,249	34.67	9.01	7.98	6,249,232.83	32,480,869.80	28767740.40	94.08 5,878,791	94.36 30,663,549	90.81 26,134,700	370,441.83	1,817320.80	2,633,040.40
1955	162,307	37.08	8.51	8.21	6,018,343.56	27,624,651.40	26,650,809.40	95.14 5,725,852.06	95.73 26,505,853.02	91.71 24,441,457.30	292,491.50	1,118,778.38	2,209,352.10
1956													
1957													
1958													
1959													

closed down excess due to surge of past years

YEAR	TONS MILLED	MILL HEAD GRADE Ag oz	Pb %	Zn %	MILL HEAD METALS Ag oz	Pb lbs	Zn lbs	CONCENTRATE METALS Ag oz	Pb lbs	Zn lbs	TO TAILINGS METALS Ag oz	Pb lbs	Zn lbs	
1982	55,491	24.63	2.99	0.67	1,366,743	N.T. 3318361	743,579	85.99 1,175,327	59.00 3,319,639	[illegible].30 747,479	775,596	1,958,587	483,618	51,586
1983	32,628	22.79	2.56	0.49	743,592	1,670,553	319,754	86.09 640,155	63.90 1,067,498	43.81 139,245	103,437	603,055	180,509	30,232
1984	72,409	19.74	2.38	0.32	1,429,353	3,446,668	463,417	87.32 1,247,937	67.25 2,313,350	54.41 250,881	181,416	1,133,318	212,536	67,287
1985	74,609	20.91	2.36	0.25	1,560,074	3,521,544	373,045	83.02 1,295,288	60.15 2,115,713	37.37 139,687	264,786	1,405,831	233,358	68,813
1981	66,924	21.88	3.59	0.64	1,464,297	4,805,143	856,627	80.36 1,103,752	47.22 2,265,950	32.18 275,731	360,545	2,539,193	580,896	63,023

1965 → 1985

YEAR	TONS MILLED	MILL HEAD GRADE Ag oz	Pb %	Zn %	MILL HEAD METALS Ag oz	Pb lbs	Zn lbs	CONCENTRATE METALS Ag oz	Pb lbs	Zn lbs	TO TAILINGS METALS Ag oz	Pb lbs	Zn lbs
Sub Total 19	1,840,492	—	—	—	54,059,089	176,396,111	102,416,821	52,804,513	150,230,062	85,328,505	~~[illegible]~~	28,126,032	17,575,797
1936 →	1964							103,545,346	339,525,252	257,452,470	4,760,774 10,467,483	55,337,607	39,384,877
Sub Total	2,734,463	—	—	—	114,012,265	411,541,163	304,876,059				~~10,440,483~~	~~2,768,120~~	~~2,454,653~~
Grand Total				77	+2,922,000						15,183,257	30,894,152	20,030,450
													1,578,115

mill
7.2
26,000
2.00

UNITED KENO HILL MINES LIMITED

To: A. Hayward

From: K. Watson

Date: December 8, 1987

cc:

Re: Update to Sept. 7/85 Memo on Tailings/Spirals

Gravity concentration test work has not as yet been done on any tailings.

A drill program to test U.K.H.M. and Treadwell-Yukon tailings has been initiated and is expected to be completed in March, 1988.

The "lost" 75,000 tons of Galkeno tailings has not yet been located.

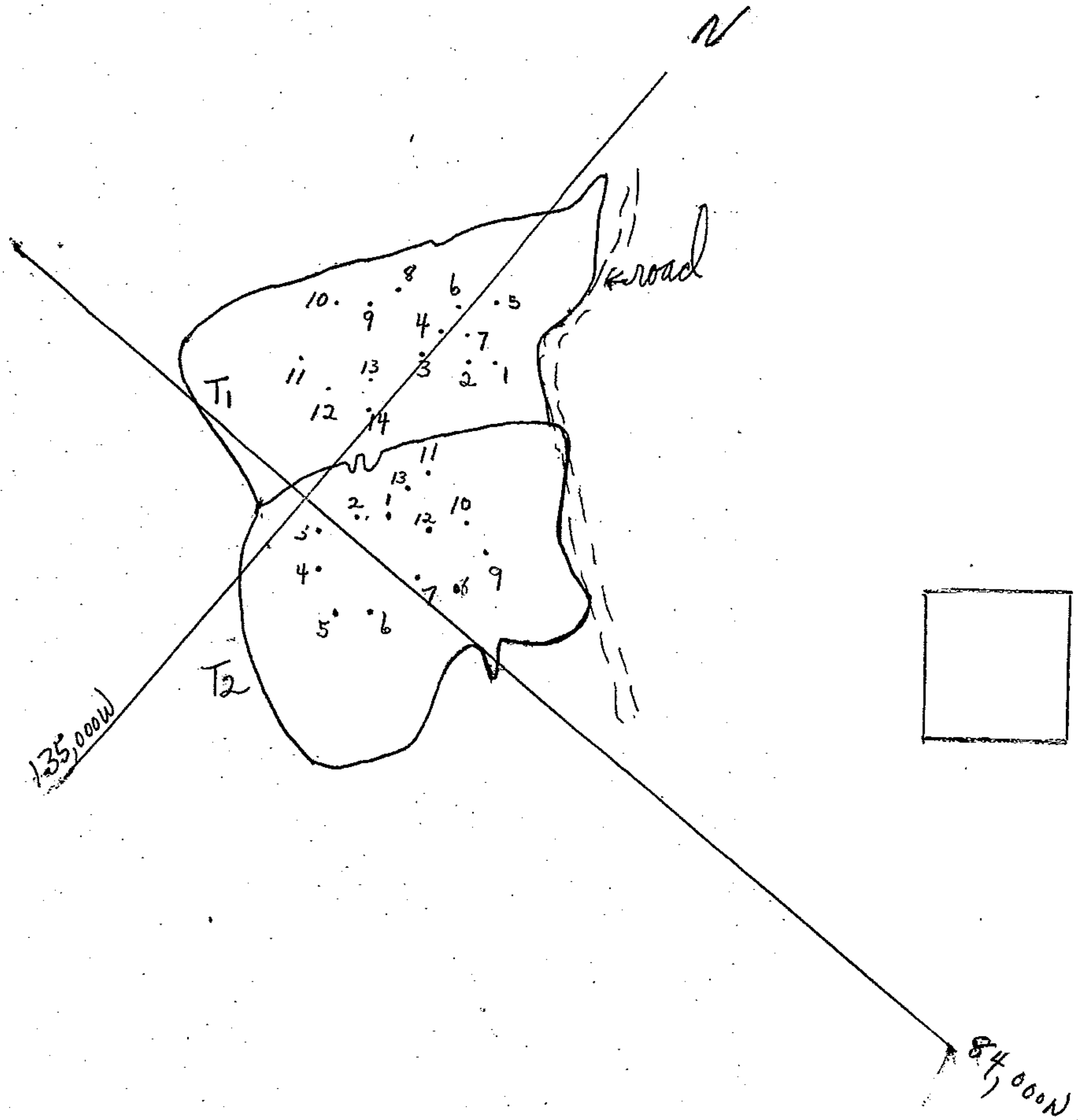
Sadie-Ladue tailings were briefly investigated in 1987. They were dumped down the hillside below the Wernecke mill and now cover an area several thousand feet long, by 50 to 300 wide, by 1 to 2 feet deep. Reported tailings were 316,577 tons at 3.65 oz Ag/ton, 0.74% Pb, and 3.22% Zn.

The terraced tailings directly below Elsa (15,500 tons at 10.7 oz Ag/t) have been identified as U.K.H.M. tails, dating about 1950.

860602

Year	Tons	Mill Head Grade			Mill Head MetalS			Concentrate Metals			Recovery			Tailings Metals			Tailings
	Milled	Ag oz	Pb %	Zn %	Ag oz.	Pb lbs.	Zn lbs.	Ag oz.	Pb lbs.	Zn lbs.	%Ag	%Pb	%Zn	Ag oz.	Pb lbs.	Zn lbs.	Tons
36-41	253,764	62.39	6.90	2.80	15,832,336	35,019,432	14,210,784	12,509,163	22,486,844	8,419,552	79.01	64.21	59.25	3,323,173	12,532,588	5,791,232	226,220
1947	20,880	43.40	7.18	0.00	906,192	2,998,368	0	680,461	1,835,942	0	75.09	61.23	0.00	225,731	1,162,426	0	18,572
1948	37,593	57.32	8.06	0.00	2,154,831	6,059,992	0	1,816,437	4,643,909	0	84.30	76.63	0.00	338,394	1,416,083	0	31,954
1949	29,494	51.18	9.87	4.29	1,509,503	5,822,116	2,530,585	1,288,951	4,579,902	2,263,392	85.39	78.66	89.44	220,552	1,242,214	267,193	23,438
1950	77,465	51.36	10.99	5.48	3,978,602	17,026,807	8,490,164	3,298,054	13,598,760	7,022,820	82.89	79.87	82.72	680,548	3,428,047	1,467,344	63,248
1951	88,026	49.22	9.81	4.70	4,332,640	17,270,701	8,274,444	3,464,646	13,063,588	7,196,832	79.97	75.64	86.98	867,994	4,207,113	1,077,612	73,794
1952	102,269	35.70	8.77	6.46	3,651,003	17,937,983	13,213,155	3,399,630	14,436,073	10,280,241	93.11	80.48	77.80	251,373	3,501,910	2,932,914	84,247
1953	156,684	40.92	9.45	7.47	6,411,509	29,613,276	23,408,590	6,182,442	27,313,584	21,245,493	96.43	92.23	90.76	229,067	2,299,692	2,163,097	112,829
1954	180,249	34.67	9.01	7.98	6,249,233	32,480,870	28,767,740	5,878,791	30,663,549	26,134,700	94.07	94.40	90.85	370,442	1,817,321	2,633,040	137,849
1955	162,307	37.08	8.51	8.21	6,018,344	27,624,651	26,650,809	5,725,852	26,505,853	24,441,457	95.14	95.95	91.71	292,492	1,118,798	2,209,352	123,978
1956	155,702	38.17	8.48	8.34	5,943,145	26,407,059	25,971,094	5,582,979	25,083,145	24,107,851	93.94	94.99	92.83	360,166	1,323,914	1,863,243	117,885
1957	159,885	38.65	7.56	6.39	6,179,555	24,174,612	20,433,303	5,694,850	22,569,908	18,119,454	92.16	93.36	88.68	484,705	1,604,704	2,313,849	128,974
C1958	175,058	36.83	7.18	6.10	6,447,386	25,138,329	21,357,076	5,984,373	22,255,501	18,610,970	92.82	88.53	87.14	463,013	2,882,828	2,746,106	144,700
C1959	173,477	44.11	7.42	5.88	7,652,070	25,743,987	20,400,895	7,307,815	22,865,276	17,717,019	95.50	88.82	86.84	344,255	2,878,711	2,683,876	142,000
C1960	176,745	43.35	7.25	4.80	7,661,896	25,628,025	16,967,520	7,249,100	21,986,887	14,440,744	94.61	85.79	85.11	412,796	3,641,138	2,526,776	149,000
C1961	186,116	41.16	5.83	4.84	7,660,535	21,701,126	18,016,029	7,231,980	17,911,672	15,512,624	94.41	82.54	86.10	428,555	3,789,454	2,503,405	159,000
C1962	184,123	40.55	5.84	4.42	7,466,188	21,505,566	16,276,473	7,000,837	17,587,767	13,885,884	93.77	81.78	85.31	465,351	3,917,799	2,390,589	15,900
C1963	186,721	34.03	5.44	4.69	6,354,116	20,315,245	17,514,430	5,978,075	16,751,012	14,759,821	94.08	82.46	84.27	376,041	3,564,233	2,754,609	158,397
C1964	227,845	33.37	6.38	4.92	7,603,188	29,073,022	22,419,948	7,270,911	26,304,902	19,965,295	95.63	90.48	89.05	332,277	2,768,120	2,454,653	190,083
C1965	146,850	33.25	7.06	6.22	4,882,763	20,735,220	18,268,140	4,701,820	18,753,650	16,700,565	96.29	90.44	91.42	180,943	1,981,570	1,567,575	119,585
C1966	120,374	36.56	7.60	5.61	4,400,873	18,296,848	13,505,963	4,235,678	16,647,849	11,999,953	96.25	90.99	88.85	165,195	1,648,999	1,506,010	73,266
1967	106,189	37.71	7.97	5.89	4,004,387	16,926,527	12,509,064	3,804,644	15,469,569	10,872,074	95.01	91.39	86.91	199,743	1,456,958	1,636,990	51,619
C1968	60,800	33.93	6.53	5.55	2,062,944	7,940,480	6,748,800	1,981,777	7,418,645	6,212,589	96.07	93.43	92.05	81,167	521,835	536,211	50,400
1969	87,483	27.98	4.56	4.67	2,447,774	7,978,450	8,170,912	2,405,615	7,719,096	7,845,682	98.28	96.75	96.02	42,159	259,354	325,230	74,378
1970	93,215	27.30	4.07	5.28	2,544,770	7,587,701	9,843,504	2,430,254	6,946,332	8,155,894	95.50	91.55	82.86	114,516	641,369	1,687,610	80,111
1971	94,754	30.57	5.17	5.19	2,896,630	9,797,564	9,835,465	2,806,567	8,834,831	7,518,885	96.89	90.17	76.45	90,063	962,733	2,316,580	81,480
1972	80,646	34.23	4.61	3.19	2,760,513	7,435,561	5,145,215	2,634,176	7,106,437	3,967,611	95.42	95.57	77.11	126,337	329,124	1,177,604	72,209
1973	94,819	37.91	4.38	1.73	3,594,588	8,306,144	3,280,737	3,410,045	7,879,855	2,351,650	94.87	94.87	71.68	184,543	426,289	929,087	86,633
1974	93,232	37.73	4.22	1.15	3,517,643	7,868,781	2,144,336	3,337,383	7,220,476	1,411,750	94.88	91.76	65.84	180,260	648,305	732,586	85,866
1975	90,860	34.96	4.03	1.15	3,176,466	7,323,316	2,089,780	2,992,420	6,582,728	1,512,675	94.21	89.89	72.38	184,046	740,588	577,105	84,024
1976	75,515	35.49	4.02	1.17	2,680,027	6,071,406	1,767,051	2,500,752	5,251,997	1,319,243	93.31	86.50	74.66	179,275	819,409	447,808	70,451
1977	91,486	35.49	4.57	1.12	3,246,838	8,361,820	2,049,286	2,914,406	6,274,710	1,491,150	89.76	75.04	72.76	332,432	2,087,110	558,136	84,874
1978	90,082	35.72	5.51	0.79	3,217,729	9,927,036	1,423,296	2,917,456	7,603,472	1,073,449	90.67	76.59	75.42	300,273	2,323,564	349,847	82,539
C 1979	124,322	23.87	3.67	0.60	2,967,566	9,125,235	1,491,864	2,536,975	6,011,617	835,871	85.49	65.88	56.03	430,591	3,113,618	655,993	117,140
C 1980	87,784	22.96	3.39	0.79	2,015,521	5,951,755	1,386,987	1,732,086	3,426,648	506,441	85.94	57.57	36.51	283,435	2,525,107	880,546	82,589
C 1981	66,924	21.88	3.59	0.64	1,464,297	4,805,143	856,627	1,103,752	2,265,950	275,713	75.38	47.16	32.19	360,545	2,539,193	580,914	63,033
1982	55,491	24.63	2.99	0.67	1,366,743	3,318,362	743,579	1,175,327	1,957,832	262,483	85.99	59.00	35.30	191,416	1,360,530	481,096	51,586
1983	32,628	22.79	2.56	0.49	743,592	1,670,554	319,754	640,155	1,067,498	139,245	86.09	63.90	43.55	103,437	603,056	180,509	30,232
1984	72,409	19.74	2.38	0.32	1,429,354	3,446,668	463,418	1,247,937	2,313,350	250,681	87.31	67.12	54.14	181,417	1,133,318	212,537	67,287
1985	74,609	20.91	2.36	0.25	1,560,074	3,521,545	373,045	1,295,288	2,115,713	139,687	83.03	60.08	37.45	264,786	1,405,832	233,358	68,813
1986	34,140	23.62	2.97	0.40	806,387	2,027,916	273,120	692,887	1,450,377	146,493	85.92	71.52	53.64	113,500	577,539	126,627	32,400
SUBTOT	4,609,015	37.27	6.40	4.42	171,799,740	589,965,185	407,592,975	157,042,747	502,762,706	349,114,133	91.41	85.22	85.65	14,757,003	87,202,492	58,478,850	3,712,583
LEACHED														202,168	31,729	56,135	
TOTAL														14,554,835	87,170,763	58,422,715	3,712,583
GRADE														3.92	1.17%	0.79%	

C INDICATES YEARS WHEN CYANIDE PLANT WAS USED.



TAIlings terrace.

1" = 100'

DAILY CHIP ASSAY REPORT

United Keno Hill Mines Ltd.

DATE Sept 1 1981.

MINE Tailings. Terrace

WORKING PLACE	LOCATION	WIDTH	SAMPLE NO.	AG. OZ.	% Pb	% Xn	REMARKS
Tailings. (T2)	T2-1		2715	8.9	3.30	.64	Tailings.
"	T2-2		2716	8.5	3.20	.65	"
"	T2-3		2717	7.4	2.37	.44	"
"	T2-4		2718	7.3	2.43	.46	"
"	T2-5		2719	7.3	2.41	.42	"
"	T2-6		2720	8.7	2.88	.46	"
"	T2-7		2721	8.6	2.92	.47	"
"	T2-8		2722	15.0	4.16	.96	"
"	T2-9		2723	16.2	4.40	1.02	"
"	T2-10		2724	6.8	2.28	.44	"
"	T2-11		2725	6.9	2.29	.45	"
"	T2-12		2726	7.6	3.03	.51	"
"	T2-13		2727	7.6	2.92	.52	"
			average	(8.98)	(2.96)		(1.74 NS Pb)

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DAILY CHIP ASSAY REPORT

United Keno Hill Mines Ltd.

DATE Sept 1 19 81

MINE Tailings. Terrace

WORKING PLACE	LOCATION	WIDTH	SAMPLE NO.	AG. OZ.	% Pb	% Xn	REMARKS
Tailings. (T2)	T2-1		2715	8.9	3.30	.64	Tailings.
"	T2-2		2716	8.5	3.20	.65	"
"	T2-3		2717	7.4	2.37	.44	"
"	T2-4		2718	7.3	2.43	.46	"
"	T2-5		2719	7.3	2.41	.42	"
"	T2-6		2720	8.7	2.88	.46	"
"	T2-7		2721	8.6	2.92	.47	"
"	T2-8		2722	15.0	4.16	.96	"
"	T2-9		2723	16.2	4.40	1.02	"
"	T2-10		2724	6.8	2.28	.44	"
"	T2-11		2725	6.9	2.29	.45	"
"	T2-12		2726	7.6	3.03	.51	"
"	T2-13		2727	7.6	2.92	.52	"
			average	(8.98)	(2.96)		(1.74 NS Pb)

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DAILY CHIP ASSAY REPORT

United Keno Hill Mines Ltd.

DATE Sept 1 1981

MINE Tailings S. Terrace.

WORKING PLACE	LOCATION	WIDTH	SAMPLE NO.	AG. OZ.	% Pb	% Xn	REMARKS
Tailings (T1)	1		2701	7.3	2.04	.71	Tailings.
"	2		2702	6.5	1.49	.65	"
"	3		2703	9.0	2.48	.70	"
"	4		2704	9.9	2.55	.61	"
"	5		2705	8.0	2.10	.80	"
"	6		2706	8.0	2.02	.75	"
"	7		2707	6.8	2.24	.75	"
"	8		2708	6.8	2.23	.71	"
"	9		2709	7.3	2.53	.41	"
"	10		2710	7.2	2.57	.43	"
"	11		2711	4.6	1.38	.52	"
"	12		2712	4.4	1.37	.54	"
"	13		2713	8.2	2.14	.73	"
"	14		2714	8.1	2.70	.74	"
			average.	(7.29)	(2.17)		1.32 NS Pb.

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DAILY CHIP ASSAY REPORT

United Keno Hill Mines Ltd.

DATE Sept 1 19 81

MINE TAILINGS. TERRACE.

WORKING PLACE	LOCATION	WIDTH	SAMPLE NO.	AG. OZ.	% Pb	% Xn	REMARKS
Tailings (T1)	1		2701	3	2.04	.71	Tailings.
"	2		2702	6.5	1.49	.65	"
"	3		2703	9.0	2.48	.70	"
"	4		2704	9.9	2.55	.61	"
"	5		2705	8.0	2.10	.80	"
"	6		2706	8.0	2.02	.75	"
"	7		2707	6.8	2.24	.75	"
"	8		2708	6.8	2.23	.71	"
"	9		2709	7.3	2.53	.41	"
"	10		2710	7.2	2.57	.43	"
"	11		2711	4.6	1.38	.52	"
"	12		2712	4.4	1.37	.54	"
"	13		2713	8.2	2.14	.73	"
"	14		2714	8.1	2.70	.74	"
			average	(7.29)	(2.17)		1.32 NS Pb.

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Appendix D-2

Historic Tailings Reprocessing Assessment Reports

UNITED KENO HILL MINES LIMITED

1987 - 1988

TAILINGS DRILLING PROGRAM

by

Ken W. Watson
Chief Geologist

and

Richard Houncaren
Project Geologist

April 30, 1988

INTRODUCTION

A program to drill the higher grade, non cyanided portion of the Elsa mill tailings was initiated in 1987. The lower grade, east end portion of the tailings had been previously drilled in 1970. Airphoto investigation revealed that the Treadwell-Yukon tailings and the early, higher grade U.K.H.M. tailings were all deposited on the west side of the tailings pond. A program to test these tailings areas was proposed in early 1987.

The drill program was conducted in two stages. In August and September, 1987, the tailings were fully delineated, a 100 foot by 100 foot grid established and the dry portion of the tailings was drilled. The second stage, involved drilling of the wet tailings areas in February, after they had been frozen. A total of 5,396 feet of drilling in 379 holes was completed. The drilling was done under contract, rotary drilling with a Schramm drill and a duocone bit.

SUMMARY and CONCLUSIONS

Appendix A contains tables showing overall results of the 1987/88 drill program. In addition to these, are tables showing the U.K.H.M. and Treadwell-Yukon mill discharge figures and a table showing the total estimate of all drilled tailings areas. The grade and depth distribution of the tailings are shown in Figures 1 and 2.

The 1987/88 program delineated 1.7 million tons of tails at a grade of 4.45 oz Ag/ton. Table 1 shows the drill results along with estimated of the total tailings area.

Table 1

Source	Tons	Grade (oz Ag/t)	Ounces Ag
1987/88 Drilling	1,699,405	4.45	7,562,352
1970 Drilling	2,156,175	1.91	4,118,294
1950 Terraced	15,500	10.70	165,850
New Discharge	70,000	5.50	385,000
Under 2nd Pond	108,590	4.63	502,772
TOTAL TAILINGS	4,049,670	3.14	12,734,268
Mill Discharge	4,049,670	3.98	16,117,687

The drill results indicate a grade 0.84 oz Ag/ton lower than the mill discharge figure. This may be due to inaccuracies in the drill grade calculations or may also represent leaching of the contained silver in the tailings ponds.

An initial estimate of selective mining potential indicates 1.0 million tons at a grade of 5.35 oz Ag/ton.

<u>Calculation Parameters</u>

Appendix B contains tables showing the results from each drill hole. A 5 foot sample interval was used since it was anticipated that an exact tailings-overburden contact would be difficult to determine in the field. In all holes, a field estimate of the contact location was made. Two numbers were generated based on this contact location. The first number is a straight average with the second number being a bias weight average. An example of how the number were calculated is to take a hole with a field estimated tailings depth of 18 feet. The straight average would be the silver assay over 20 feet applied to a tailings depth of 18 feet (i.e. 4 oz Ag/t/20' = 4 oz Ag/t/18'). With the bias weight average, the final assay number is corrected back to the actual tailings depth assuming overburden to have a silver value of 0. In our example, if the last sample interval was 4 oz Ag/t/5' this would be corrected to 6.67 oz Ag/t/3' for a total hole average of 4.45 oz Ag/t/18'. This number was then averaged with the straight average number to produce a composite average (A+B/2 column on the calculation tables) which was used in all final calculations (i.e. (4 + 4.45)/2 = 4.2 oz Ag/t/18'). Values for Pb, Zn, Fe, Cd, & NSPb were all calculated in the same manner as silver.

Since the overall drill grade average is lower than the mill discharge figure it can be assumed that drill average represents a conservative minimum grade.

APPENDIX A

Summary Sheets

Area: 1987/88 Drilling U.K.H.M. TAILINGS SAMPLING PROGRAM Mar/88

Area No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Ag	Pb	Zn	Fe	Cd	NSPb
1	289,250	14.2	4,107,350	205,368	4.63	1.13	0.58	11.71		
2	360,000	13.5	4,860,000	243,000	6.50	1.24	0.53	10.76		
3	90,250	11.0	992,750	49,638	5.69	1.15	0.32	8.11	0.01	0.81
4	246,250	5.6	1,379,000	68,950	3.97	1.05	0.74	11.81	0.03	0.80
5	272,500	14.3	3,896,750	194,838	3.41	1.07	0.87	12.93	0.02	0.60
6	267,500	13.7	3,664,750	183,238	3.87	1.02	0.79	11.50	0.02	0.80
7	67,750	14.9	1,009,475	50,474	5.74	1.24	0.50	10.57		
8	264,000	12.7	3,352,800	167,640	4.89	1.23	0.91	13.30	0.03	0.85
9	144,250	10.3	1,485,775	74,289	3.71	0.69	0.35	7.65	0.01	0.50
10	222,250	11.6	2,578,100	128,905	3.61	0.91	1.00	14.23	0.02	0.59
11	142,250	4.4	625,900	31,295	3.90	0.82	0.67	14.67	0.02	0.48
12	214,750	6.2	1,331,450	66,573	3.57	0.93	0.84	13.60	0.02	0.52
13	392,000	12.0	4,704,000	235,200	3.80	0.96	4.75	12.96	0.02	0.61
			0	0						
	2,973,000	12.3	33,988,100	1,699,405	4.45	1.06	1.27	12.03	0.03	0.84

NOTE: A tonnage factor of 20 cu ft/ton was used.

Area: TOTAL U.K.H.M. TAILINGS SAMPLING PROGRAM Mar/88

Area No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Ag	Pb	Zn	Fe	Cd	NSPb
1	289,250	14.2	4,107,350	205,368	4.63	1.13	0.58	11.71		
2	360,000	13.5	4,860,000	243,000	6.50	1.24	0.53	10.76		
3	90,250	11.0	992,750	49,638	5.69	1.15	0.32	8.11	0.01	0.81
4	246,250	5.6	1,379,000	68,950	3.97	1.05	0.74	11.81	0.03	0.80
5	272,500	14.3	3,896,750	194,838	3.41	1.07	0.87	12.93	0.02	0.60
6	267,500	13.7	3,664,750	183,238	3.87	1.02	0.79	11.50	0.02	0.80
7	67,750	14.9	1,009,475	50,474	5.74	1.24	0.50	10.57		
8	264,000	12.7	3,352,800	167,640	4.89	1.23	0.91	13.30	0.03	0.85
9	144,250	10.3	1,485,775	74,289	3.71	0.69	0.35	7.65	0.01	0.50
10	222,250	11.6	2,578,100	128,905	3.61	0.91	1.00	14.23	0.02	0.59
11	142,250	4.4	625,900	31,295	3.90	0.82	0.67	14.67	0.02	0.48
12	214,750	6.2	1,331,450	66,573	3.57	0.93	0.84	13.60	0.02	0.52
13	392,000	12.0	4,704,000	235,200	3.80	0.96	4.75	12.96	0.02	0.61
1970	3,885,000	11	43,123,500	2,156,175	1.91	0.50	0.55	5.20	0.01	0.39
	6,858,000	11.6	77,111,600	3,855,580	3.03	0.75	0.87	8.21	0.01	0.49

NOTE: A tonnage factor of 20 cu ft/ton was used.

T.Y. & U.K.H.M. - Mill Tailings Discharge

SOURCE	YEAR	TONS	GRADE (oz Ag/t)	OUNCES AG
T.Y	1936-41	226,220	14.69	3,323,172
U.K.H.M.	1947	18,572	12.16	225,836
U.K.H.M.	1948	31,954	10.62	339,351
U.K.H.M.	1949	23,438	9.41	220,552
U.K.H.M.	1950	62,964	10.76	677,493
U.K.H.M.	1951	73,794	11.76	867,817
U.K.H.M.	1952	84,247	8.17	688,298
U.K.H.M.	1953	120,826	4.67	564,257
U.K.H.M.	1954	137,849	2.68	369,435
U.K.H.M.	1955	123,978	2.35	291,348
U.K.H.M.	1956	117,885	3.06	360,728
U.K.H.M.	1957	128,974	3.75	483,652
U.K.H.M.	1958	63,900	5.18	331,002
U.K.H.M.	1958	80,000 C	2.65	212,000
U.K.H.M.	1959	142,000 C	2.37	336,540
U.K.H.M.	1960	149,000 C	2.74	408,260
U.K.H.M.	1961	159,000 C	2.63	418,170
U.K.H.M.	1962	159,000 C	2.89	459,510
U.K.H.M.	1963	158,000 C	1.86	293,880
U.K.H.M.	1964	193,000 C	1.68	324,240
U.K.H.M.	1965	120,000 C	1.51	181,200
U.K.H.M.	1966	96,700 C	1.40	135,380
U.K.H.M.	1967	35,008	3.66	128,129
U.K.H.M.	1967	51,619 C	1.31	67,621
U.K.H.M.	1968	50,400	2.86	144,144
U.K.H.M.	1969	74,378	1.42	105,617
U.K.H.M.	1970	80,111	1.43	114,559
U.K.H.M.	1971	81,480	1.11	90,443
U.K.H.M.	1972	72,209	1.75	126,366
U.K.H.M.	1973	86,633	2.13	184,528
U.K.H.M.	1974	85,866	2.10	180,319
U.K.H.M.	1975	84,024	2.19	184,013
U.K.H.M.	1976	70,675	2.54	179,515
U.K.H.M.	1977	84,677	3.93	332,781
U.K.H.M.	1978	82,407	4.03	332,100
U.K.H.M.	1979	117,200	3.46 *	405,512
U.K.H.M.	1980	82,500	3.23 *	266,475
U.K.H.M.	1981	63,033	4.38 *	276,085
U.K.H.M.	1982	51,586	3.45 *	177,972
U.K.H.M.	1983	30,232	3.42	103,393
U.K.H.M.	1984	67,286	2.69	180,999
U.K.H.M.	1985	68,812	3.85	264,926
U.K.H.M.	1986	75,769	4.32	327,322
U.K.H.M.	1987	82,464	5.24	432,111
TOTAL TAILINGS:		4,049,670	3.98	16,117,051
Treadwell-Yukon =		226,220	14.69	3,323,172
U.K.H.M. Tails =		3,823,450	3.35	12,793,879

C = Cyanided tailings
* = Poorly documented

COMPARISON OF DRILLED TAILINGS TO MILL DISCHARGE FIGURES

SOURCE	TONS	GRADE (oz Ag/t)	OUNCES AG	
1970 Drilling	2,156,175	1.91	4,118,294	
1987/88 Drilling	1,699,405	4.45	7,562,352	
1950 Terraced	15,500	10.70	165,850	
New Discharge	70,000	5.50	385,000	estimated
Under 2nd Pond	108,590	4.63	502,772	estimated
			0	
TOTAL TAILINGS:	4,049,670	3.14	12,734,268	
Mill Discharge	4,049,670	3.98	16,117,687	
Difference	0	-0.84	(3,383,418)	

APPENDIX B

Calculation Sheets

AREA 1 U.K.H.M. TAILINGS SAMPLING PROGRAM Mar/88

Hole No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Avg	Bias	A+B/2	Pb	Zn	Fe	Cd	NSPb
E40	2,500	10	25,000	1,250	2.21	2.21	2.21	0.78	0.48	11.09	0.00	0.00
E39	5,000	10	50,000	2,500	6.17	6.17	6.17	1.58	0.47	11.59	0.00	0.00
E38	3,750	10	37,500	1,875	2.67	2.67	2.67	0.83	0.45	9.65	0.00	0.00
F40	5,000	10	50,000	2,500	3.11	3.11	3.11	0.98	0.42	10.28	0.00	0.00
F39	10,000	13	130,000	6,500	5.29	6.05	5.67	1.19	0.35	9.15	0.00	0.00
F38	10,000	15	150,000	7,500	1.78	1.78	1.78	0.57	0.48	10.25	0.00	0.00
F37	6,750	9	60,750	3,038	0.95	1.04	1.00	0.18	0.14	4.75	0.00	0.00
F36	5,000	13	65,000	3,250	0.76	0.90	0.83	0.26	0.16	5.37	0.00	0.00
G40	5,000	14	70,000	3,500	3.11	3.39	3.25	1.14	0.51	11.44	0.00	0.00
G39	10,000	12	120,000	6,000	3.44	3.77	3.60	0.85	0.76	14.21	0.00	0.00
G38	10,000	12	120,000	6,000	4.88	5.95	5.42	1.33	0.56	11.69	0.00	0.00
G37	10,000	12	120,000	6,000	3.16	3.76	3.46	0.76	0.32	10.43	0.00	0.00
G36	10,000	15	150,000	7,500	4.72	4.72	4.72	1.17	0.58	8.27	0.00	0.00
H40	5,000	15	75,000	3,750	6.61	6.61	6.61	1.95	0.61	14.48	0.00	0.00
H39	10,000	14	140,000	7,000	3.48	3.70	3.59	1.11	0.67	12.87	0.00	0.00
H38	10,000	17	170,000	8,500	7.59	8.89	8.24	1.90	0.58	12.09	0.00	0.00
H37	10,000	18	180,000	9,000	4.91	5.39	5.15	1.15	0.52	11.42	0.00	0.00
H36	10,000	20	200,000	10,000	3.83	3.83	3.83	0.97	1.08	16.82	0.00	0.00
I40	5,000	18	90,000	4,500	4.45	4.94	4.70	1.29	0.62	12.74	0.00	0.00
I39	10,000	17	170,000	8,500	2.88	3.26	3.07	0.89	0.74	13.87	0.00	0.00
I38	10,000	17	170,000	8,500	8.73	10.42	9.58	2.33	0.66	13.64	0.00	0.00
I37	10,000	17	170,000	8,500	5.53	6.59	6.06	1.11	0.41	9.09	0.00	0.00
I36	10,000	19	190,000	9,500	8.29	8.76	8.52	1.50	0.58	10.66	0.00	0.00
J40	5,000	19	95,000	4,750	6.19	6.46	6.32	1.49	0.41	10.95	0.00	0.00
J39	10,000	15	150,000	7,500	3.85	3.85	3.85	1.02	0.72	13.33	0.00	0.00
J38	10,000	17	170,000	8,500	4.97	6.08	5.52	1.41	0.71	14.59	0.00	0.00
J37	10,000	19	190,000	9,500	4.37	4.57	4.47	0.97	0.63	12.61	0.00	0.00
J36	10,000	18	180,000	9,000	6.07	6.91	6.49	1.26	0.56	12.68	0.00	0.00
K40	5,000	5	25,000	1,250	1.53	1.53	1.53	0.73	0.62	11.20	0.00	0.00
K39	10,000	8	80,000	4,000	1.78	2.00	1.89	0.50	0.52	11.09	0.00	0.00
K38	10,000	12	120,000	6,000	1.43	1.75	1.59	0.76	0.54	11.06	0.00	0.00
K37	10,000	8	80,000	4,000	2.08	2.19	2.13	0.67	0.49	11.20	0.00	0.00
K36	10,000	15	150,000	7,500	2.88	4.30	3.59	1.22	0.76	15.17	0.00	0.00
L40	5,000	12	60,000	3,000	1.85	2.03	1.94	0.72	0.38	6.76	0.00	0.00
L39	6,250	10	62,500	3,125	1.20	1.90	1.55	0.50	0.21	4.67	0.00	0.00
L38	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
L37	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
L36	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
M40	2,500	12	30,000	1,500	2.52	2.84	2.68	1.02	0.47	9.22	0.00	0.00
M39	2,500	8	20,000	1,000	2.40	2.76	2.58	0.98	0.27	5.20	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	289,250	14.2	4,115,750	205,788	4.37	4.89	4.63	1.13	0.58	11.71	0.00	0.00

NOTE: A tonnage factor of 20 cu ft/ton was used.
Final grade is the A+B/2 column - see report for details

Hole No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Avg	Bias	A+B/2	Pb	Zn	Fe	Cd	NSPb
F29	3,750	11	41,250	2,063	2.84	3.61	3.23	0.60	0.35	8.47	0.00	0.00
F30	5,000	10	50,000	2,500	4.45	4.45	4.45	1.04	0.53	11.86	0.00	0.00
F31	5,000	8	40,000	2,000	1.91	2.17	2.04	0.38	0.20	6.30	0.00	0.00
F32	5,000	4	20,000	1,000	1.59	3.09	2.34	0.40	0.32	9.58	0.00	0.00
F33	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
F34	5,000	8	40,000	2,000	0.82	1.01	0.92	0.12	0.09	3.87	0.00	0.00
F35	5,000	12	60,000	3,000	1.80	2.24	2.02	0.61	0.46	9.58	0.00	0.00
G29	10,000	12	120,000	6,000	5.50	8.29	6.89	1.38	0.64	14.86	0.00	0.00
G30	10,000	12	120,000	6,000	1.33	1.52	1.42	0.39	0.12	5.28	0.00	0.00
G31	10,000	13	130,000	6,500	1.81	1.97	1.89	0.55	0.26	8.44	0.00	0.00
G32	10,000	9	90,000	4,500	1.54	2.31	1.92	0.52	0.19	6.29	0.00	0.00
G33	10,000	18	180,000	9,000	8.89	10.14	9.51	1.85	0.53	13.03	0.00	0.00
G34	10,000	18	180,000	9,000	11.41	12.83	12.12	2.30	0.70	12.33	0.00	0.00
G35	10,000	19	190,000	9,500	10.97	11.54	11.26	2.17	0.82	15.12	0.00	0.00
H29	10,000	13	130,000	6,500	4.17	4.54	4.35	0.83	0.29	7.48	0.00	0.00
H30	10,000	15	150,000	7,500	5.86	5.86	5.86	0.93	0.33	9.21	0.00	0.00
H31	10,000	13	130,000	6,500	8.86	10.15	9.50	1.97	0.65	10.92	0.00	0.00
H32	10,000	12	120,000	6,000	9.55	12.24	10.90	2.69	0.71	15.41	0.00	0.00
H33	10,000	20	200,000	10,000	8.99	8.99	8.99	1.25	0.33	8.68	0.00	0.00
H34	10,000	20	200,000	10,000	10.19	10.19	10.19	1.50	0.59	10.52	0.00	0.00
H35	10,000	15	150,000	7,500	6.31	6.31	6.31	1.25	0.50	9.89	0.00	0.00
I29	10,000	12	120,000	6,000	3.46	3.80	3.63	0.65	0.36	8.73	0.00	0.00
I30	10,000	14	140,000	7,000	7.57	8.11	7.84	1.65	0.50	11.37	0.00	0.00
I31	10,000	13	130,000	6,500	7.49	8.30	7.90	1.42	0.51	10.01	0.00	0.00
I32	10,000	14	140,000	7,000	5.13	5.52	5.32	0.73	0.41	7.38	0.00	0.00
I33	10,000	19	190,000	9,500	9.29	9.87	9.58	1.31	0.56	10.32	0.00	0.00
I34	10,000	19	190,000	9,500	7.11	7.51	7.31	1.24	0.62	10.97	0.00	0.00
I35	10,000	18	180,000	9,000	9.38	10.54	9.96	1.64	0.63	11.16	0.00	0.00
J29	10,000	8	80,000	4,000	3.03	4.91	3.97	0.81	0.84	12.85	0.00	0.00
J30	10,000	11	110,000	5,500	2.28	2.28	2.28	0.57	0.43	7.55	0.00	0.00
J31	10,000	8	80,000	4,000	5.29	10.72	8.00	1.81	0.90	16.39	0.00	0.00
J32	10,000	12	120,000	6,000	4.83	5.94	5.38	1.27	0.70	13.61	0.00	0.00
J33	10,000	18	180,000	9,000	3.11	3.44	3.27	0.84	0.77	15.33	0.00	0.00
J34	10,000	18	180,000	9,000	5.08	5.84	5.46	1.12	0.58	12.46	0.00	0.00
J35	10,000	19	190,000	9,500	3.13	3.13	3.13	0.63	0.28	6.55	0.00	0.00
K29	5,000	4	20,000	1,000	0.80	1.00	0.90	0.11	0.07	3.42	0.00	0.00
K30	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
K31	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
K32	5,000	4	20,000	1,000	1.12	1.40	1.26	0.63	0.23	5.58	0.00	0.00
K33	10,000	10	100,000	5,000	6.27	6.27	6.27	1.30	0.62	12.70	0.00	0.00
K34	10,000	15	150,000	7,500	7.33	7.33	7.33	1.56	0.53	11.85	0.00	0.00
K35	10,000	12	120,000	6,000	2.49	3.17	2.83	0.88	0.78	14.49	0.00	0.00
L29	3,750	6	22,500	1,125	2.89	4.23	3.56	0.83	0.98	14.88	0.00	0.00
L33	3,750	3	11,250	563	3.20	5.33	4.27	0.85	0.55	11.73	0.00	0.00
L34	3,750	8	30,000	1,500	2.14	2.24	2.19	0.65	0.42	7.82	0.00	0.00
L35	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	360,000	13.5	4,845,000	242,250	6.15	6.85	6.50	1.24	0.53	10.76	0.00	0.00

NOTE: A tonnage factor of 20 cu ft/ton was used
Final grade is the A+B/2 column - see report for details

AREA 3 U.K.H.M. TAILINGS SAMPLING PROGRAM Mar/88

Hole No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Avg	Bias	A+B/2	Pb	Zn	Fe	Cd	NSPb
G28	5,000	11	55,000	2,750	1.76	3.65	2.71	0.67	0.32	9.86	0.00	0.00
H25	4,000	5	20,000	1,000	0.60	0.60	0.60	0.10	0.02	2.86	0.00	0.00
H26	6,000	15	90,000	4,500	18.01	18.01	18.01	4.65	0.73	13.97	0.00	0.00
H27	6,750	14	94,500	4,725	6.87	7.36	7.11	1.06	0.55	10.53	0.00	0.00
H28	10,000	13	130,000	6,500	6.18	7.08	6.63	1.19	0.24	7.40	0.00	0.00
I24	6,500	6	39,000	1,950	7.23	10.07	8.65	1.80	0.30	9.14	0.00	0.00
I25	6,000	14	84,000	4,200	3.66	3.35	3.75	0.59	0.28	5.76	0.00	0.00
I26	5,000	12	60,000	3,000	2.39	4.50	3.44	0.56	0.14	6.33	0.00	0.00
I27	6,250	10	62,500	3,125	3.59	3.59	3.59	0.78	0.30	7.38	0.00	0.00
I28	10,000	9	90,000	4,500	2.71	2.86	2.78	0.50	0.26	8.45	0.00	0.00
J23	4,000	13	52,000	2,600	4.18	5.25	4.72	0.92	0.45	8.96	0.01	0.68
J24	5,000	13	65,000	3,250	2.83	3.09	2.96	0.56	0.19	5.62	0.00	0.00
J28	6,250	12	75,000	3,750	0.49	0.59	0.54	0.04	0.03	3.23	0.00	0.00
K23	4,500	8	36,000	1,800	6.21	7.01	6.61	1.22	0.44	9.44	0.01	0.85
L23	5,000	8	40,000	2,000	6.68	7.70	7.19	1.27	0.36	9.66	0.01	0.94
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	90,250	11.0	993,000	49,650	5.35	6.04	5.69	1.15	0.32	8.11	0.01	0.81

NOTE: A tonnage factor of 20 cu ft/ton was used.
Final grade is the A+B/2 column - see report for details

AREA 4 U.K.H.M. TAILINGS SAMPLING PROGRAM Mar/88

Hole No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Avg	Bias	A+B/2	Pb	Zn	Fe	Cd	NSPb
K28	5,000	7	35,000	1,750	5.11	6.02	5.57	1.22	0.42	10.99	0.00	0.00
L27	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
L28	6,250	6	37,500	1,875	4.38	5.49	4.94	0.95	0.27	7.66	0.00	0.00
M23	10,000	6	60,000	3,000	4.64	7.42	6.03	1.24	0.93	17.00	0.03	0.76
M24	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
M26	3,750	2	7,500	375	17.74	44.35	31.05	3.52	0.96	22.47	0.00	0.00
M27	10,000	6	60,000	3,000	3.89	4.32	4.10	0.92	0.27	7.78	0.00	0.00
M28	6,250	2	12,500	625	2.26	5.65	3.96	0.94	0.21	8.02	0.00	0.00
M29	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
N23	10,000	4	40,000	2,000	5.10	6.38	5.74	1.05	0.61	9.82	0.02	0.61
N24	5,000	3	15,000	750	3.44	5.73	4.59	1.35	0.87	15.24	0.03	0.87
N26	6,250	2	12,500	625	0.44	1.10	0.77	0.18	0.05	6.89	0.00	0.00
N27	10,000	12	120,000	6,000	2.38	2.92	2.65	0.46	0.16	5.33	0.00	0.00
N28	4,000	4	16,000	800	3.18	3.98	3.58	0.70	0.18	6.79	0.00	0.00
O23	10,000	4	40,000	2,000	2.78	3.48	3.13	0.86	0.68	14.90	0.02	0.50
O24	10,000	5	50,000	2,500	6.74	6.74	6.74	1.40	0.71	14.24	0.02	0.60
O25	10,000	2	20,000	1,000	2.32	5.80	4.06	1.56	1.47	17.64	0.05	0.58
O26	6,250	2	12,500	625	5.61	14.03	9.82	1.75	0.56	14.88	0.02	0.91
O27	3,750	2	7,500	375	0.46	1.15	0.80	0.04	0.02	4.97	0.00	0.00
P23	10,000	12	120,000	6,000	0.43	0.63	0.53	0.05	0.04	2.98	0.00	0.00
P24	10,000	3	30,000	1,500	2.66	4.43	3.55	1.29	1.27	14.05	0.03	0.68
P25	6,250	1	6,250	313	2.01	10.05	6.03	2.31	1.68	27.34	0.06	0.90
P26	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Q23	10,000	12	120,000	6,000	3.44	4.25	3.85	1.37	0.92	14.05	0.03	0.94
Q24	10,000	3	30,000	1,500	4.28	7.13	5.71	1.55	2.03	24.11	0.03	0.76
Q25	3,750	1	3,750	188	1.47	7.35	4.41	5.33	2.22	28.08	0.06	2.46
R23	10,000	10	100,000	5,000	2.98	2.98	2.98	0.89	0.74	8.33	0.00	0.00
R24	8,000	4	32,000	1,600	3.66	4.58	4.12	1.33	1.24	15.05	0.03	0.87
S23	10,000	7	70,000	3,500	3.31	4.32	3.81	1.39	1.09	14.13	0.03	0.96
S24	8,000	6	48,000	2,400	2.98	4.62	3.80	1.37	1.00	15.51	0.04	0.99
T23	10,000	7	70,000	3,500	3.81	5.03	4.42	1.39	1.30	15.12	0.04	0.94
T24	10,000	6	60,000	3,000	3.81	5.58	4.70	1.25	1.14	16.27	0.04	0.70
U23	10,000	7	70,000	3,500	3.46	4.46	3.96	1.07	0.95	15.06	0.04	0.78
U24	10,000	7	70,000	3,500	3.13	4.09	3.61	1.13	0.86	13.23	0.03	0.73
U25	3,750	3	11,250	563	1.44	2.40	1.92	0.51	0.44	7.44	0.01	0.27
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	246,250	5.6	1,387,250	69,362	3.34	4.59	3.97	1.05	0.74	11.81	0.03	0.80

NOTE: A tonnage factor of 20 cu ft/ton was used
Final grade is the A+B/2 column - see report for details

AREA 5 U.X.H.M. TAILINGS SAMPLING PROGRAM Mar/88

Hole No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Avg	Bias	A+B/2	Pb	Zn	Fe	Cd	NSPb
V20	10,000	10	100,000	5,000	2.66	2.66	2.66	0.96	0.72	13.48	0.02	0.70
V21	10,000	10	100,000	5,000	3.04	3.04	3.04	0.97	0.98	15.27	0.02	0.65
V22	10,000	17	170,000	8,500	2.80	3.31	3.06	0.89	1.00	15.00	0.02	0.60
V23	10,000	10	100,000	5,000	2.81	2.81	2.81	0.86	0.92	13.52	0.02	0.57
V24	10,000	5	50,000	2,500	3.90	3.90	3.90	0.92	0.88	14.86	0.02	0.44
V25	3,750	4	15,000	750	2.90	3.63	3.26	0.68	0.30	9.83	0.01	0.37
W20	10,000	10	100,000	5,000	4.42	4.81	4.61	1.28	0.64	12.68	0.00	0.00
W21	10,000	15	150,000	7,500	1.86	1.86	1.86	0.67	0.46	8.77	0.00	0.00
W22	10,000	15	150,000	7,500	3.44	3.44	3.44	0.79	0.69	13.72	0.00	0.00
W23	10,000	11	110,000	5,500	3.21	4.41	3.81	1.37	1.08	13.66	0.00	0.00
W24	6,250	10	62,500	3,125	3.30	3.30	3.30	0.91	0.53	12.30	0.00	0.00
X20	10,000	20	200,000	10,000	4.94	4.94	4.94	1.33	1.30	15.93	0.00	0.00
X21	10,000	20	200,000	10,000	3.00	3.00	3.00	1.33	0.92	12.05	0.00	0.00
X22	10,000	15	150,000	7,500	2.50	2.50	2.50	1.13	0.45	9.16	0.00	0.00
X23	10,000	13	130,000	6,500	2.42	2.76	2.59	1.18	0.54	9.37	0.00	0.00
X24	3,750	3	11,250	563	2.15	3.58	2.87	1.19	0.52	9.60	0.00	0.00
Y20	10,000	15	150,000	7,500	6.18	6.18	6.18	1.57	1.04	15.64	0.00	0.00
Y21	10,000	17	170,000	8,500	3.91	4.69	4.30	1.30	1.17	17.43	0.00	0.00
Y22	10,000	18	180,000	9,000	3.41	3.97	3.69	1.08	0.71	11.56	0.00	0.00
Y23	6,250	13	81,250	4,063	2.26	2.47	2.37	0.90	0.58	9.62	0.00	0.00
Y24	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Z20	10,000	15	150,000	7,500	3.68	3.68	3.68	1.10	1.02	13.71	0.00	0.00
Z21	10,000	18	180,000	9,000	3.02	3.48	3.25	0.93	1.08	13.18	0.00	0.00
Z22	10,000	18	180,000	9,000	3.73	3.89	3.81	1.04	0.89	11.86	0.00	0.00
Z23	5,000	20	100,000	5,000	2.68	2.68	2.68	0.86	0.59	9.15	0.00	0.00
Z24	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
AA20	10,000	14	140,000	7,000	5.19	5.58	5.38	1.48	1.21	15.37	0.00	0.00
AA21	10,000	19	190,000	9,500	1.92	2.05	1.98	0.86	0.75	10.88	0.00	0.00
AA22	10,000	14	140,000	7,000	2.27	2.44	2.36	1.02	0.89	12.55	0.00	0.00
AA23	6,250	13	81,250	4,063	2.18	2.51	2.35	0.83	0.98	12.19	0.00	0.00
AA24	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
AB20	5,000	18	90,000	4,500	3.66	4.04	3.85	0.85	0.89	14.22	0.00	0.00
AB21	5,000	17	85,000	4,250	3.41	4.20	3.80	1.04	1.01	16.18	0.00	0.00
AB22	5,000	18	90,000	4,500	2.31	2.63	2.47	0.98	0.80	13.12	0.00	0.00
AB23	5,000	14	70,000	3,500	3.32	3.41	3.37	0.93	0.69	10.43	0.00	0.00
AB24	1,250	8	10,000	500	5.26	6.53	5.89	1.53	0.99	12.42	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	272,500	14.3	3,886,250	194,313	3.29	3.54	3.41	1.07	0.87	12.93	0.02	0.60

NOTE: A tonnage factor of 20 cu ft/ton was used.
Final grade is the A+B/2 column - see report for details

AREA 6 U.X.H.M. TAILINGS SAMPLING PROGRAM Mar/88

Hole No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Avg	Bias	A+B/2	Pb	Zn	Fe	Cd	NSPb
V15	6,500	10	65,000	3,250	1.29	1.29	1.29	0.26	0.27	5.70	0.00	0.00
V16	6,000	19	114,000	5,700	1.72	1.87	1.79	0.35	0.29	6.75	0.00	0.00
V17	10,000	10	100,000	5,000	3.20	3.20	3.20	1.00	0.78	10.61	0.02	0.72
V18	10,000	10	100,000	5,000	2.80	2.80	2.80	1.14	0.82	11.34	0.02	0.79
V19	10,000	5	50,000	2,500	3.54	3.54	3.54	1.34	0.94	14.08	0.02	0.96
W15	10,000	10	100,000	5,000	0.83	0.83	0.83	0.21	0.18	4.20	0.00	0.00
W16	10,000	15	150,000	7,500	4.16	4.16	4.16	0.97	0.68	13.82	0.00	0.00
W17	10,000	15	150,000	7,500	3.23	3.23	3.23	0.66	0.44	7.65	0.00	0.00
W18	10,000	15	150,000	7,500	5.19	5.19	5.19	1.36	1.11	13.18	0.00	0.00
W19	10,000	19	190,000	9,500	4.81	5.11	4.96	1.24	0.92	13.48	0.00	0.00
X15	10,000	13	130,000	6,500	3.71	4.13	3.92	1.10	0.95	13.95	0.00	0.00
X16	10,000	13	130,000	6,500	2.09	2.20	2.15	0.47	0.26	5.55	0.00	0.00
X17	10,000	15	150,000	7,500	5.76	5.76	5.76	1.24	0.32	11.85	0.00	0.00
X18	10,000	15	150,000	7,500	4.90	4.90	4.90	1.31	1.06	14.62	0.00	0.00
X19	10,000	20	200,000	10,000	7.14	7.14	7.14	1.66	1.01	13.66	0.00	0.00
Y16	10,000	15	150,000	7,500	2.62	2.62	2.62	0.95	0.58	9.43	0.00	0.00
Y17	10,000	11	110,000	5,500	4.20	4.34	4.27	0.89	0.72	8.19	0.00	0.00
Y18	10,000	15	150,000	7,500	3.19	3.19	3.19	1.19	0.77	12.84	0.00	0.00
Y19	10,000	15	150,000	7,500	5.83	5.83	5.83	1.29	1.06	13.68	0.00	0.00
Z16	10,000	13	130,000	6,500	3.17	3.76	3.46	1.05	0.89	11.16	0.00	0.00
Z17	10,000	13	130,000	6,500	2.79	3.15	2.97	0.94	0.76	11.42	0.00	0.00
Z18	10,000	13	130,000	6,500	2.69	3.16	2.93	1.07	0.97	13.72	0.00	0.00
Z19	10,000	15	150,000	7,500	1.92	1.92	1.92	0.73	0.56	8.84	0.00	0.00
AA17	10,000	12	120,000	6,000	2.27	2.51	2.39	0.62	0.89	12.05	0.00	0.00
AA18	10,000	15	150,000	7,500	2.76	2.76	2.76	1.03	0.77	11.07	0.00	0.00
AA19	10,000	15	150,000	7,500	5.41	5.41	5.41	1.38	1.31	16.03	0.00	0.00
AB17	5,000	8	40,000	2,000	1.64	1.71	1.67	0.38	0.24	6.26	0.00	0.00
AB18	5,000	16	80,000	4,000	5.66	7.60	6.63	1.39	0.94	13.82	0.00	0.00
AB19	5,000	17	85,000	4,250	5.27	6.53	5.90	1.25	1.06	15.02	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	267,500	13.7	3,654,000	182,700	3.78	3.95	3.87	1.02	0.79	11.50	0.02	0.80

NOTE: A tonnage factor of 20 cu ft/ton was used
Final grade is the A+B/2 column - see report for details

AREA 7 U.K.H.M. TAILINGS SAMPLING PROGRAM Mar/88

Hole No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Avg	Bias	A+B/2	Pb	Zn	Fe	Cd	NSPb
I23	2,000	15	30,000	1,500	2.75	2.75	2.75	0.57	0.25	6.15	0.00	0.00
J22	4,000	10	40,000	2,000	1.74	1.74	1.74	0.26	0.09	4.21	0.00	0.00
K20	3,750	15	56,250	2,813	4.17	4.17	4.17	0.71	0.13	6.73	0.00	0.00
K21	5,750	15	86,250	4,313	5.36	5.36	5.36	1.40	0.51	11.97	0.00	0.00
K22	4,500	18	81,000	4,050	3.28	3.56	3.42	0.77	0.32	8.03	0.00	0.00
L19	6,250	15	93,750	4,688	5.02	5.02	5.02	0.98	0.33	8.15	0.00	0.00
L20	10,000	15	150,000	7,500	6.42	6.42	6.42	1.15	0.34	11.69	0.00	0.00
L21	4,000	15	60,000	3,000	8.82	8.82	8.82	1.73	0.39	9.60	0.00	0.00
L22	5,000	15	75,000	3,750	6.66	6.66	6.66	1.66	0.41	10.02	0.00	0.00
M19	12,500	15	187,500	9,375	5.29	5.29	5.29	1.24	0.71	12.93	0.00	0.00
M20	10,000	15	150,000	7,500	8.10	8.10	8.10	1.31	0.95	13.29	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	67,750	14.9	1,009,750	50,488	5.73	5.75	5.74	1.24	0.50	10.57	0.00	0.00

NOTE: A tonnage factor of 20 cu ft/ton was used.
Final grade is the A+B/2 column - see report for details

AREA 8			U.K.H.M. TAILINGS SAMPLING PROGRAM										Mar/88

Hole No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Avg	Bias	A+B/2	Pb	Zn	Fe	Cd	NSPb
M21	10,000	7	70,000	3,500	10.59	14.74	12.67	2.79	0.85	17.86	0.02	1.36
M22	10,000	7	70,000	3,500	9.85	16.00	12.92	2.32	0.82	15.44	0.02	1.44
N20	12,500	18	225,000	11,250	3.67	4.08	3.88	0.95	0.59	14.22	0.00	0.00
N21	10,000	12	120,000	6,000	3.46	4.58	4.02	1.05	0.85	15.42	0.02	0.56
N22	10,000	7	70,000	3,500	4.82	7.19	6.01	1.21	1.01	15.92	0.02	0.74
O20	10,000	10	100,000	5,000	2.00	3.97	2.98	0.51	0.37	8.64	0.00	0.00
O21	10,000	12	120,000	6,000	5.35	6.95	6.15	1.23	0.99	15.74	0.02	0.73
O22	10,000	18	180,000	9,000	3.57	1.98	2.77	0.68	0.62	10.23	0.02	0.46
P20	10,000	15	150,000	7,500	5.47	5.47	5.47	1.21	0.52	9.82	0.02	0.85
P21	10,000	18	180,000	9,000	6.72	7.59	7.15	1.45	0.67	10.70	0.03	0.90
P22	10,000	19	190,000	9,500	8.69	9.28	8.98	1.77	0.75	12.10	0.02	1.28
Q20	10,000	15	150,000	7,500	3.97	3.97	3.97	1.28	0.87	13.83	0.02	1.03
Q21	10,000	19	190,000	9,500	6.90	7.37	7.13	1.46	1.00	13.21	0.03	0.93
Q22	10,000	17	170,000	8,500	3.61	4.11	3.86	1.21	1.01	13.76	0.03	0.82
R20	10,000	13	130,000	6,500	2.75	3.22	2.98	1.04	1.19	16.43	0.03	0.73
R21	10,000	15	150,000	7,500	2.95	2.95	2.95	0.87	1.05	13.87	0.03	0.49
R22	10,000	13	130,000	6,500	2.35	2.62	2.48	1.15	0.95	13.53	0.03	0.86
S20	12,500	15	187,500	9,375	3.46	3.46	3.46	1.04	1.14	13.74	0.03	0.69
S21	10,000	18	180,000	9,000	3.27	3.67	3.47	1.04	1.13	12.31	0.03	0.65
S22	10,000	6	60,000	3,000	3.18	5.37	4.27	1.37	1.12	14.66	0.04	1.04
T21	15,000	7	105,000	5,250	3.86	4.85	4.36	1.34	1.26	14.52	0.04	0.96
T22	10,000	7	70,000	3,500	3.39	4.51	3.95	1.21	1.15	14.84	0.03	0.38
U19	5,500	18	99,000	4,950	3.14	3.66	3.40	1.13	0.80	11.13	0.02	0.84
U20	8,500	12	102,000	5,100	4.78	5.74	5.26	1.53	1.46	14.64	0.04	1.17
U21	10,000	8	80,000	4,000	3.08	3.81	3.45	1.31	0.80	12.60	0.03	0.89
U22	10,000	8	80,000	4,000	3.72	4.63	4.17	1.07	1.38	14.15	0.05	0.80
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	264,000	12.7	3,358,500	167,925	4.53	5.25	4.89	1.23	0.91	13.30	0.03	0.85

NOTE: A tonnage factor of 20 cu ft/ton was used.
Final grade is the A+B/2 column - see report for details

AREA 9 U.K.H.M. TAILINGS SAMPLING PROGRAM Mar/88

Hole No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Avg	Bias	A+B/2	Pb	Zn	Fe	Cd	NSPb
K16	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
K17	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
K18	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
L16	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
L17	2,500	8	20,000	1,000	7.30	9.15	8.23	1.81	0.78	16.56	0.02	0.95
L18	5,000	10	50,000	2,500	3.84	3.84	3.84	1.03	0.79	12.04	0.02	0.78
M16	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
M17	6,000	7	42,000	2,100	6.63	9.10	7.87	1.55	0.51	13.10	0.02	0.93
M18	10,000	10	100,000	5,000	4.79	4.79	4.79	0.62	0.10	5.17	0.00	0.00
N15	2,500	2	5,000	250	1.38	3.45	2.42	0.56	0.16	7.35	0.00	0.00
N16	7,000	8	56,000	2,800	6.68	7.56	7.12	1.39	0.68	14.08	0.02	0.52
N17	10,000	10	100,000	5,000	1.77	1.77	1.77	0.19	0.06	3.36	0.01	0.13
N18	12,500	10	125,000	6,250	3.23	3.23	3.23	0.43	0.17	4.35	0.01	0.33
O15	5,000	5	25,000	1,250	1.99	1.99	1.99	0.28	0.03	2.91	0.00	0.00
O16	10,000	12	120,000	6,000	8.31	9.39	8.85	1.51	0.61	11.59	0.02	1.13
O17	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
O18	10,000	5	50,000	2,500	1.14	1.14	1.14	0.13	0.13	3.46	0.01	0.08
O19	12,500	20	250,000	12,500	3.59	3.59	3.59	0.72	0.32	7.14	0.00	0.00
P15	5,000	9	45,000	2,250	4.75	5.19	4.97	1.27	1.06	13.72	0.02	0.96
P16	10,000	8	80,000	4,000	2.76	3.63	3.20	0.76	0.59	9.37	0.02	0.43
P17	10,000	6	60,000	3,000	0.98	1.65	1.32	0.15	0.12	4.02	0.01	0.05
P18	10,000	13	130,000	6,500	2.46	2.79	2.62	0.49	0.45	8.89	0.01	0.29
P19	10,000	15	150,000	7,500	1.07	1.07	1.07	0.23	0.15	4.71	0.00	0.00
Q15	6,250	13	81,250	4,063	2.89	3.12	3.00	0.78	0.40	9.73	0.01	0.49
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	144,250	10.3	1,489,250	74,463	3.54	3.89	3.71	0.69	0.35	7.65	0.01	0.50

NOTE: A tonnage factor of 20 cu ft/ton was used.
Final grade is the A+B/2 column - see report for details

AREA 10 U.X.H.M. TAILINGS SAMPLING PROGRAM Mar/88

Hole No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Avg	Bias	A+B/2	Pb	Zn	Fe	Cd	NSPb
Q16	10,000	9	90,000	4,500	4.48	5.19	4.84	1.12	0.68	13.61	0.02	0.71
Q17	10,000	10	100,000	5,000	4.42	4.42	4.42	0.99	0.59	12.97	0.02	0.73
Q18	10,000	12	120,000	6,000	1.94	2.47	2.20	0.55	0.80	13.06	0.02	0.40
Q19	10,000	15	150,000	7,500	3.72	3.72	3.72	0.83	0.82	15.41	0.02	0.60
R15	6,250	7	43,750	2,188	2.17	3.17	2.67	0.58	0.98	16.69	0.02	0.46
R16	10,000	9	90,000	4,500	3.36	3.69	3.53	0.73	0.77	13.95	0.02	0.66
R17	10,000	13	130,000	6,500	3.22	3.74	3.48	0.72	0.91	15.91	0.03	0.38
R18	10,000	16	160,000	8,000	2.81	3.56	3.18	0.86	1.26	17.64	0.03	0.52
R19	10,000	19	190,000	9,500	3.42	3.56	3.49	0.98	1.07	14.61	0.02	0.47
S15	10,000	9	90,000	4,500	1.06	1.11	1.08	0.34	0.53	8.33	0.01	0.29
S16	10,000	5	50,000	2,500	1.76	1.76	1.76	0.70	0.63	11.21	0.02	0.50
S17	10,000	13	130,000	6,500	3.78	4.25	4.02	1.08	1.31	15.65	0.04	0.61
S18	10,000	13	130,000	6,500	3.72	4.35	4.04	1.26	1.30	14.11	0.03	0.78
S19	12,500	17	212,500	10,625	3.60	4.21	3.90	1.13	1.39	14.92	0.03	0.63
T15	10,000	7	70,000	3,500	4.09	4.63	4.36	0.85	1.04	17.69	0.02	0.54
T16	10,000	7	70,000	3,500	3.25	3.89	3.57	0.78	1.49	17.11	0.03	0.46
T17	10,000	7	70,000	3,500	3.20	4.90	4.05	0.97	1.41	19.97	0.04	0.62
T18	15,000	13	195,000	9,750	3.46	3.92	3.69	0.81	1.19	14.30	0.03	0.59
U15	10,000	7	70,000	3,500	3.45	4.20	3.82	0.97	0.93	17.21	0.02	0.59
U16	10,000	10	100,000	5,000	2.66	2.66	2.66	0.54	0.49	7.03	0.01	0.41
U17	10,000	15	150,000	7,500	5.03	5.03	5.03	1.12	0.87	11.45	0.02	0.78
U18	8,500	19	161,500	8,075	3.76	3.93	3.84	1.26	0.86	12.14	0.02	0.95
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	222,250	11.6	2,572,750	128,638	3.40	3.82	3.61	0.91	1.00	14.23	0.02	0.59

NOTE: A tonnage factor of 20 cu ft/ton was used.
Final grade is the A+B/2 column - see report for details

Hole No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Avg	Bias	A+B/2	Pb	Zn	Fe	Cd	NSPb
S14	5,000	10	50,000	2,500	1.82	1.82	1.82	0.39	0.36	8.60	0.01	0.47
S10	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
T9	0	0	0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
T10	3,750	3	11,250	563	1.04	1.73	1.39	0.27	0.15	6.48	0.01	0.11
T11	5,000	3	15,000	750	1.54	2.57	2.05	0.47	0.32	7.97	0.01	0.27
T12	5,500	5	27,500	1,375	2.03	2.03	2.03	0.50	0.45	8.48	0.00	0.28
T13	7,000	7	49,000	2,450	4.20	4.80	4.50	0.89	1.12	17.59	0.02	0.57
T14	10,000	7	70,000	3,500	3.21	3.64	3.43	0.69	0.95	20.70	0.03	0.31
U9	3,750	3	11,250	563	1.68	2.80	2.24	0.53	0.27	9.09	0.01	0.20
U10	10,000	7	70,000	3,500	2.98	4.17	3.58	0.93	0.50	13.40	0.01	0.34
U11	10,000	6	60,000	3,000	1.08	1.72	1.40	0.16	0.25	7.95	0.00	0.00
U12	9,000	5	45,000	2,250	1.33	1.33	1.33	0.25	0.41	8.84	0.00	0.00
U13	7,000	7	49,000	2,450	7.61	8.57	8.09	1.53	1.31	19.11	0.00	0.00
U14	10,000	6	60,000	3,000	6.52	7.08	6.79	1.27	0.58	12.21	0.00	0.94
V9	6,250	4	25,000	1,250	3.52	4.40	3.96	1.05	0.62	15.71	0.02	0.43
V10	10,000	3	30,000	1,500	3.57	5.95	4.76	0.91	0.63	17.04	0.01	0.45
V11	10,000	1	10,000	500	2.13	10.65	6.39	1.56	1.14	28.20	0.06	0.42
V12	10,000	2	20,000	1,000	2.50	6.25	4.38	1.40	0.86	22.51	0.02	0.61
V13	10,000	1	10,000	500	3.10	15.50	9.30	2.31	1.62	49.86	0.06	1.02
V14	10,000	1	10,000	500	2.26	11.30	6.78	1.20	0.84	25.32	0.06	0.60
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	142,250	4.4	623,000	31,150	3.29	4.52	3.90	0.82	0.67	14.67	0.02	0.48

NOTE: A tonnage factor of 20 cu ft/ton was used
Final grade is the A+B/2 column - see report for details

AREA 12 U.K.H.M. TAILINGS SAMPLING PROGRAM Mar/88

Hole No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Avg	Bias	A+B/2	Pb	Zn	Fe	Cd	NSPb
W9	6,250	2	12,500	625	3.00	7.50	5.25	1.44	0.95	18.55	0.02	0.45
W10	10,000	6	60,000	3,000	3.24	4.95	4.10	1.27	0.73	12.63	0.01	0.53
W11	10,000	6	60,000	3,000	2.76	4.25	3.50	1.00	0.67	13.17	0.01	0.42
W12	10,000	2	20,000	1,000	1.90	4.75	3.33	1.23	1.47	16.80	0.04	0.26
W13	10,000	2	20,000	1,000	1.48	3.70	2.59	0.84	0.77	14.40	0.04	0.23
W14	10,000	1	10,000	500	2.61	13.05	7.83	1.89	1.59	35.19	0.06	0.90
X6	3,000	3	9,000	450	4.42	7.37	5.89	1.39	0.71	18.05	0.03	0.53
X7	5,000	3	15,000	750	1.89	3.15	2.52	0.47	0.76	13.01	0.03	0.19
X8	6,250	8	50,000	2,500	2.74	3.28	3.01	0.72	0.76	11.67	0.02	0.42
X9	10,000	9	90,000	4,500	2.36	2.62	2.49	0.70	0.89	13.07	0.02	0.52
X10	10,000	10	100,000	5,000	2.22	2.22	2.22	0.81	0.67	10.45	0.02	0.55
X11	10,000	10	100,000	5,000	3.46	3.46	3.46	0.83	0.83	13.45	0.02	0.60
X12	10,000	9	90,000	4,500	2.07	2.31	2.19	0.79	0.96	11.46	0.02	0.32
X13	10,000	5	50,000	2,500	1.62	1.62	1.62	0.52	0.37	7.33	0.02	0.08
X14	10,000	2	20,000	1,000	2.76	6.90	4.83	1.45	1.07	17.22	0.04	0.77
Y9	10,000	7	70,000	3,500	5.79	8.22	7.01	1.54	1.34	24.57	0.03	0.93
Y10	10,000	11	110,000	5,500	6.04	7.73	6.88	1.10	1.73	22.01	0.03	0.64
Y11	8,000	6	48,000	2,400	2.79	4.95	3.87	1.30	1.02	14.44	0.01	0.67
Y12	6,500	6	39,000	1,950	2.52	3.74	3.13	1.12	0.76	10.79	0.03	0.58
Y13	6,000	4	24,000	1,200	2.38	2.98	2.68	0.70	0.41	8.53	0.01	0.26
Y14	6,250	1	6,250	313	2.94	14.70	8.82	3.09	1.53	32.67	0.06	1.38
Y15	10,000	11	110,000	5,500	1.97	2.32	2.14	0.75	0.45	11.68	0.00	0.00
Z15	5,000	6	30,000	1,500	0.81	0.87	0.84	0.25	0.14	5.42	0.00	0.00
AA15	5,000	6	30,000	1,500	1.63	2.83	2.23	1.20	0.81	11.92	0.00	0.00
AA16	10,000	11	110,000	5,500	4.10	5.42	4.76	0.84	0.55	10.48	0.00	0.00
AB15	2,500	1	2,500	125	2.20	11.00	6.60	1.68	1.08	23.52	0.00	0.00
AB16	5,000	8	40,000	2,000	1.45	1.63	1.54	0.56	0.42	8.69	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	214,750	6.2	1,326,250	66,312	3.00	4.13	3.57	0.93	0.84	13.60	0.02	0.52

NOTE: A tonnage factor of 20 cu ft/ton was used
Final grade is the A+B/2 column - see report for details

AREA 13 U.K.H.M. TAILINGS SAMPLING PROGRAM Mar/88

Hole No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Avg	Bias	A+B/2	Pb	Zn	Fe	Cd	NSPb
Y3	5,000	2	10,000	500	6.56	16.40	11.48	2.07	1.26	30.52	0.04	1.45
Y4	5,000	1	5,000	250	0.41	2.05	1.23	0.15	0.18	9.78	0.03	0.12
Y5	7,000	6	42,000	2,100	3.08	4.62	3.85	0.98	1.53	16.93	0.03	0.52
Y6	10,000	8	80,000	4,000	2.96	3.90	3.43	0.69	0.64	13.77	0.02	0.51
Y7	10,000	8	80,000	4,000	4.44	5.53	4.99	0.92	1.05	17.62	0.02	0.59
Y8	10,000	10	100,000	5,000	3.87	3.87	3.87	0.74	0.84	13.96	0.02	0.55
Z1	3,000	6	18,000	900	3.07	5.84	4.46	0.86	0.94	13.83	0.02	0.37
Z2	7,000	12	84,000	4,200	3.67	4.68	4.18	0.78	1.25	12.25	0.02	0.41
Z3	10,000	17	170,000	8,500	4.19	5.05	4.62	1.12	1.66	15.11	0.03	0.69
Z4	10,000	17	170,000	8,500	5.04	5.72	5.38	0.98	1.27	15.83	0.02	0.61
Z5	10,000	16	160,000	8,000	3.27	4.37	3.82	0.89	1.67	14.07	0.03	0.49
Z6	10,000	15	150,000	7,500	2.84	2.84	2.84	0.82	1.12	11.26	0.03	0.51
Z7	10,000	12	120,000	6,000	2.47	3.12	2.79	0.91	1.08	11.17	0.03	0.61
Z8	10,000	15	150,000	7,500	2.93	2.93	2.93	0.88	0.80	10.98	0.02	0.50
Z9	10,000	13	130,000	6,500	2.87	3.32	3.10	0.94	0.88	11.31	0.02	0.64
Z10	10,000	9	90,000	4,500	2.60	2.90	2.75	0.98	0.75	11.07	0.03	0.66
Z11	7,000	8	56,000	2,800	3.12	3.94	3.53	1.02	0.94	13.23	0.01	0.48
AA1	5,000	2	10,000	500	2.23	5.58	3.90	1.12	0.80	13.09	0.04	0.39
AA2	10,000	4	40,000	2,000	3.80	4.75	4.28	0.86	0.83	12.92	0.02	0.38
AA3	10,000	15	150,000	7,500	3.69	3.69	3.69	1.15	0.79	10.38	0.02	0.65
AA4	10,000	18	180,000	9,000	3.70	4.06	3.88	0.97	1.04	15.36	0.02	0.57
AA5	10,000	17	170,000	8,500	3.74	4.37	4.06	0.99	1.15	16.13	0.03	0.61
AA6	10,000	18	180,000	9,000	4.03	4.44	4.23	1.12	1.51	15.36	0.03	0.65
AA7	10,000	14	140,000	7,000	3.19	3.46	3.33	0.79	1.09	12.82	0.02	0.46
AA8	10,000	14	140,000	7,000	3.16	3.45	3.31	0.81	0.97	12.06	0.02	0.45
AA9	10,000	12	120,000	6,000	2.73	3.47	3.10	1.03	0.87	11.57	0.02	0.59
AA10	10,000	12	120,000	6,000	2.82	3.64	3.23	0.95	148.07	11.59	0.02	0.55
AA11	8,000	7	56,000	2,800	2.37	3.17	2.77	1.13	0.66	10.83	0.02	0.56
AB1	5,000	1	5,000	250	1.74	8.70	5.22	0.30	0.60	11.70	0.03	0.15
AB2	10,000	13	130,000	6,500	5.75	6.38	6.07	1.41	1.03	15.35	0.02	0.98
AB3	10,000	10	100,000	5,000	7.31	7.31	7.31	1.40	0.63	10.52	0.02	0.93
AB4	10,000	11	110,000	5,500	8.25	12.22	10.23	2.40	0.86	11.77	0.02	1.58
AB5	10,000	14	140,000	7,000	4.31	4.70	4.50	1.10	1.15	15.50	0.02	0.80
AB6	10,000	13	130,000	6,500	2.96	3.44	3.20	0.70	0.74	11.83	0.01	0.50
AB7	10,000	14	140,000	7,000	3.13	3.42	3.27	1.05	0.83	11.47	0.01	0.78
AB8	10,000	14	140,000	7,000	2.73	2.93	2.83	0.79	0.73	12.00	0.01	0.58
AB9	10,000	13	130,000	6,500	4.09	4.57	4.33	0.88	0.82	12.15	0.01	0.56
AB10	10,000	11	110,000	5,500	2.42	3.35	2.88	0.92	1.02	13.61	0.02	0.55
AB11	10,000	5	50,000	2,500	1.52	1.52	1.52	0.64	0.53	7.32	0.02	0.34
AC6	10,000	15	150,000	7,500	2.49	2.49	2.49	0.83	0.75	10.91	0.02	0.64
AC7	10,000	15	150,000	7,500	2.42	2.42	2.42	0.67	0.79	11.59	0.02	0.49
AD6	10,000	15	150,000	7,500	2.68	2.68	2.68	0.66	0.95	13.35	0.02	0.46
AD7	10,000	15	150,000	7,500	2.14	2.14	2.14	0.55	0.86	11.86	0.02	0.40
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
			0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	392,000	12.0	4,706,000	235,300	3.52	4.08	3.80	0.96	4.75	12.96	0.02	0.61

NOTE: A tonnage factor of 20 cu ft/ton was used
Final grade is the A+B/2 column - see report for details

Area: 1970 DRILLING (1) U.K.H.M. TAILINGS SAMPLING PROGRAM

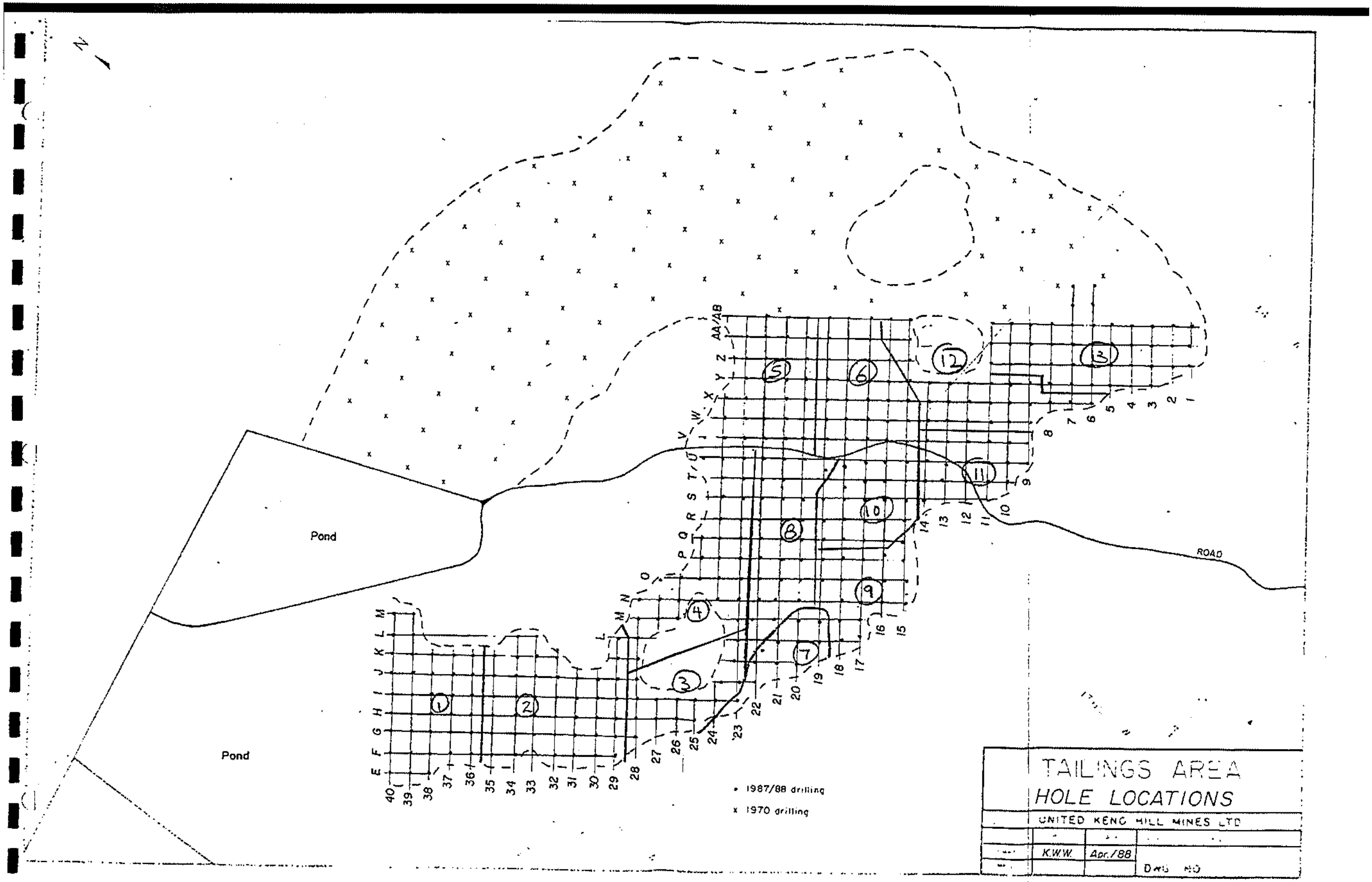
Area No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Ag	Pb	Zn	Fe
BA2	50,000	10.0	500,000	25,000	1.00			
BA3	30,000	15.0	450,000	22,500	1.70			
BA4	30,000	20.0	600,000	30,000	1.70			
BB1	60,000	10.0	600,000	30,000	1.80			
BB2	40,000	10.0	400,000	20,000	1.40			
BB3	40,000	10.0	400,000	20,000	1.60			
BB4	40,000	10.0	400,000	20,000	1.10			
BB5	50,000	10.0	500,000	25,000	1.70			
BC1	40,000	10.0	400,000	20,000	1.90			
BC2	40,000	10.0	400,000	20,000	1.50			
BC3	40,000	10.0	400,000	20,000	1.50			
BC4	40,000	15.0	600,000	30,000	1.80			
BC5	60,000	15.0	900,000	45,000	1.90			
BD1	40,000	5.0	200,000	10,000	1.50			
BD2	40,000	10.0	400,000	20,000	1.20			
BD3	40,000	10.0	400,000	20,000	1.40			
BD4	40,000	10.0	400,000	20,000	1.60			
BD5	40,000	10.0	400,000	20,000	1.50			
BD6	50,000	10.0	500,000	25,000	1.80			
BE1	40,000	10.0	400,000	20,000	1.20			
BE2	40,000	10.0	400,000	20,000	1.20			
BE3	40,000	10.0	400,000	20,000	1.50			
BE4	40,000	10.0	400,000	20,000	1.40			
BE5	40,000	10.0	400,000	20,000	1.70			
BE6	50,000	10.0	500,000	25,000	1.90			
BF1	30,000	5.0	150,000	7,500	1.50			
BF2	40,000	10.0	400,000	20,000	1.20			
BF3	40,000	10.0	400,000	20,000	1.50			
BF4	40,000	10.0	400,000	20,000	1.50			
BF5	40,000	10.0	400,000	20,000	2.30			
BF6	60,000	10.0	600,000	30,000	1.70			
BG2	45,000	5.0	225,000	11,250	1.20			
BG3	40,000	10.0	400,000	20,000	1.40			
BG4	40,000	10.0	400,000	20,000	1.70			
BG5	40,000	10.0	400,000	20,000	1.80			
BG6	40,000	10.0	400,000	20,000	1.80			
BG7	50,000	10.0	500,000	25,000	2.30			
BH2	20,000	5.0	100,000	5,000	1.60			
BH3	40,000	5.0	200,000	10,000	1.60			
BH4	40,000	10.0	400,000	20,000	2.00			
BH5	40,000	10.0	400,000	20,000	2.10			
BH6	40,000	10.0	400,000	20,000	2.60			
BH7	40,000	10.0	400,000	20,000	3.20			
BH8	40,000	10.0	400,000	20,000	3.00			
BH9	30,000	10.0	300,000	15,000	3.70			
	1,855,000	10.6	18,625,000	931,250	1.75	0.00	0.00	0.00

NOTE: A tonnage factor of 20 cu ft/ton was used.

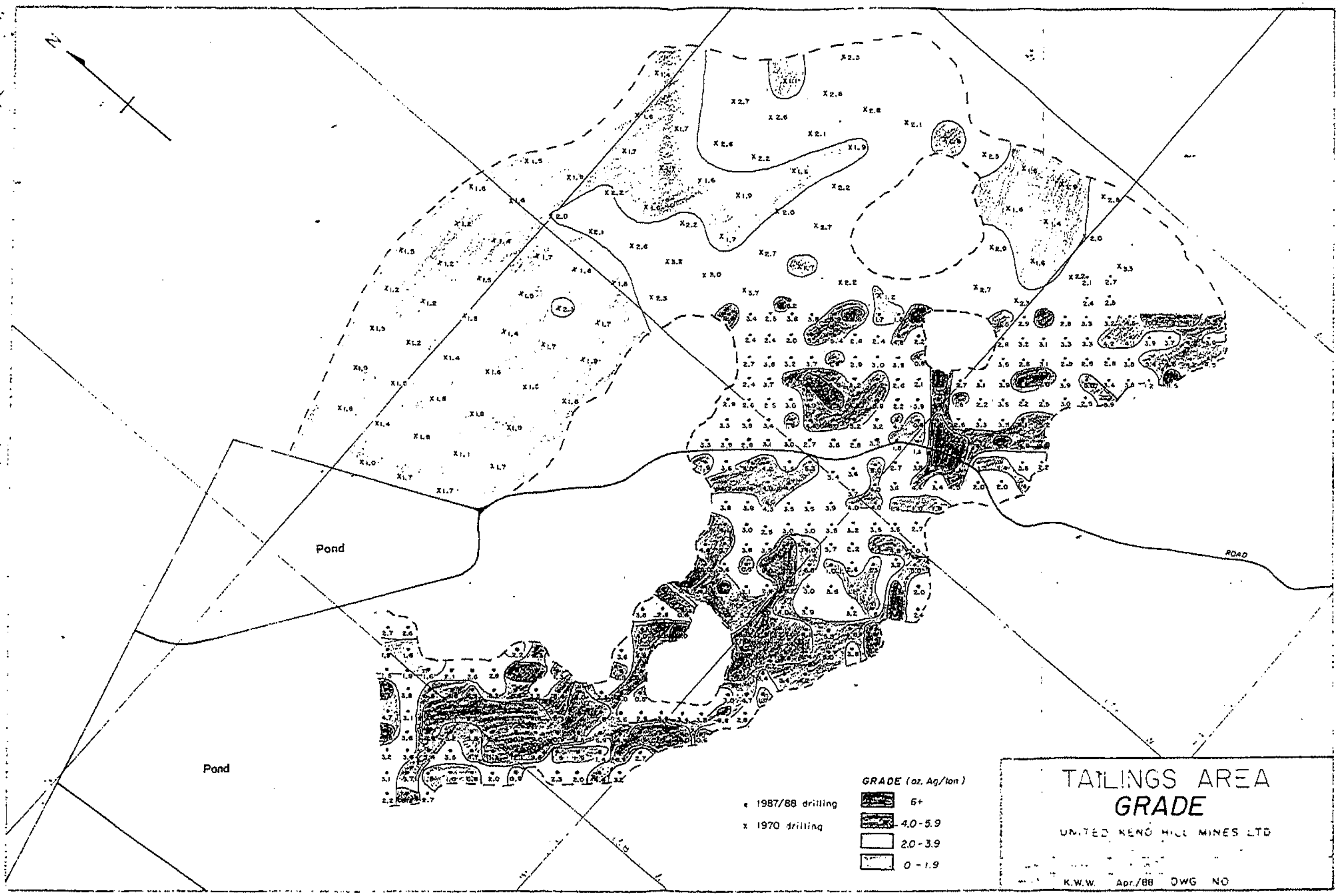
Area: 1970 DRILLING: TOTAL U.K.H.M. TAILINGS SAMPLING PROGRAM

Area No.	Area (sq ft)	Tails Depth (ft)	Volume (cu ft)	Tons	Grade (oz Ag/ton) Ag	Pb	Zn	Fe
1	1,855,000	10.6	19,682,849	984,142	1.75			
BI3	30,000	10.0	300,000	15,000	1.50			
BI4	50,000	10.0	500,000	25,000	1.90			
BI5	50,000	10.0	500,000	25,000	2.20			
BI6	40,000	10.0	400,000	20,000	1.80			
BI7	40,000	15.0	600,000	30,000	2.20			
BI8	40,000	10.0	400,000	20,000	1.70			
BI9	40,000	15.0	600,000	30,000	2.70			
BI10	40,000	10.0	400,000	20,000	1.70			
BI11	45,000	10.0	450,000	22,500	2.20			
BI12	40,000	10.0	400,000	20,000	1.20			
BJ5	50,000	10.0	500,000	25,000	1.70			
BJ6	40,000	10.0	400,000	20,000	1.70			
BJ7	40,000	10.0	400,000	20,000	1.60			
BJ8	40,000	15.0	600,000	30,000	1.90			
BJ9	40,000	15.0	600,000	30,000	2.00			
BJ10	50,000	10.0	500,000	25,000	2.70			
BJ14	40,000	10.0	400,000	20,000	2.70			
BJ15	20,000	10.0	200,000	10,000	2.30			
BK5	40,000	10.0	400,000	20,000	1.60			
BK6	50,000	10.0	500,000	25,000	1.70			
BK7	40,000	10.0	400,000	20,000	2.60			
BK8	40,000	10.0	400,000	20,000	2.20			
BK9	40,000	10.0	400,000	20,000	1.60			
BK10	45,000	10.0	450,000	22,500	2.20			
BK14	40,000	10.0	400,000	20,000	2.90			
BK15	40,000	10.0	400,000	20,000	1.60			
BK16	40,000	15.0	600,000	30,000	2.20			
BL5	50,000	10.0	500,000	25,000	1.40			
BL7	70,000	10.0	700,000	35,000	2.70			
BL8	40,000	10.0	400,000	20,000	2.60			
BL9	40,000	10.0	400,000	20,000	2.10			
BL10	50,000	15.0	750,000	37,500	1.90			
BL14	50,000	15.0	750,000	37,500	1.60			
BL15	40,000	15.0	600,000	30,000	1.40			
BL16	60,000	15.0	900,000	45,000	2.00			
BM8	50,000	5.0	250,000	12,500	1.10			
BM9	40,000	10.0	400,000	20,000	2.80			
BM10	70,000	10.0	700,000	35,000	2.80			
BM11	80,000	15.0	1,200,000	60,000	2.10			
BM12	60,000	10.0	600,000	30,000	1.80			
BM13	40,000	10.0	400,000	20,000	2.30			
BM14	40,000	10.0	400,000	20,000	1.60			
BM15	30,000	10.0	300,000	15,000	1.90			
BM16	60,000	10.0	600,000	30,000	2.50			
BN9	50,000	5.0	250,000	12,500	2.30			
	3,885,000	11.1	41,882,849	2,094,142	1.91	0.00	0.00	0.00

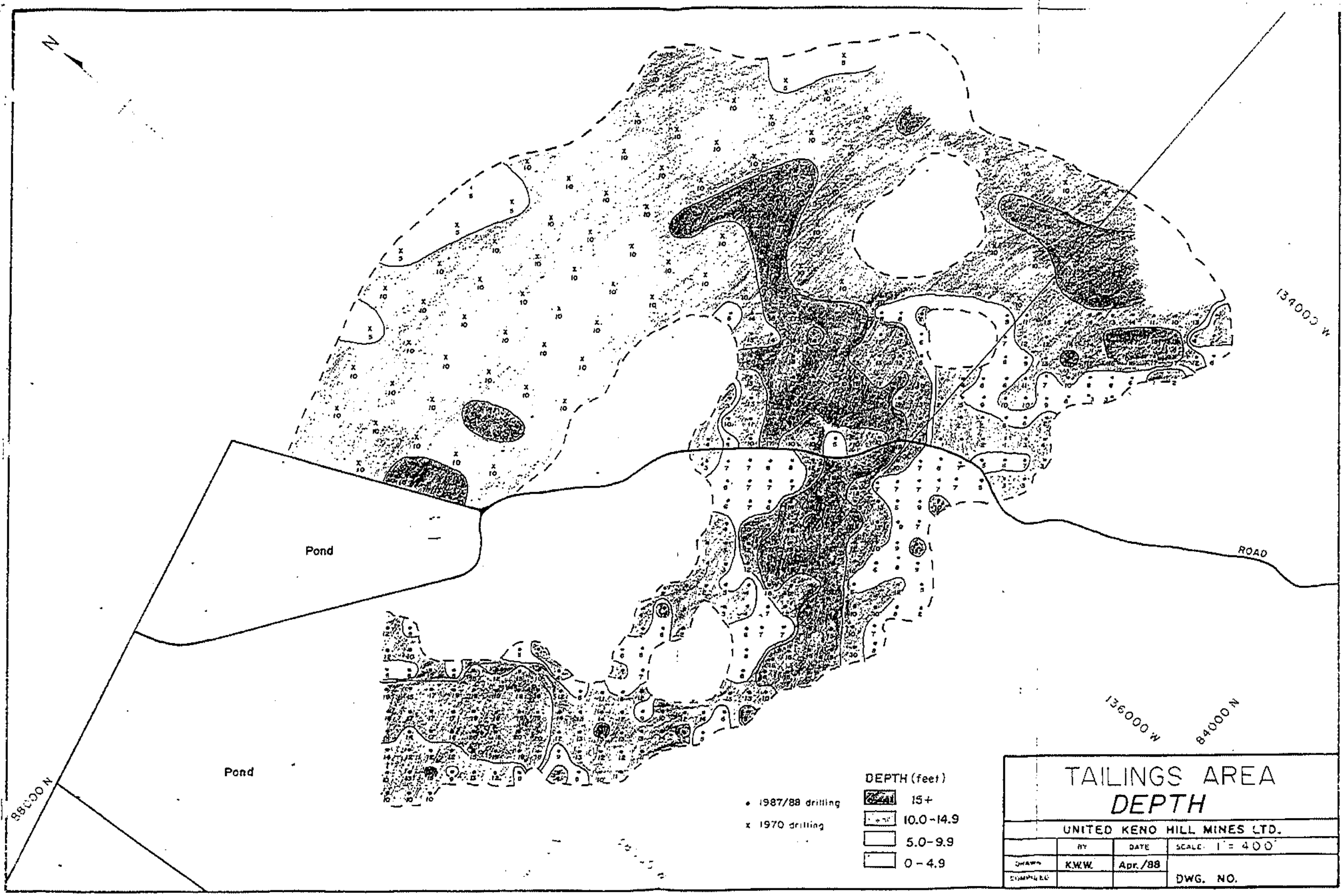
NOTE: A tonnage factor of 20 cu ft/ton was used.



N
Pond
Pond
ROAD
• 1987/88 drilling
x 1970 drilling
TAILINGS AREA
HOLE LOCATIONS
UNITED KENO HILL MINES LTD
K.W.W.
Apr./88
DWG NO



N
Pond
Pond
ROAD
GRADE (oz. Ag/ton)
6+
4.0-5.9
2.0-3.9
0-1.9
• 1987/88 drilling
x 1970 drilling
TAILINGS AREA
GRADE
UNITED KENO HILL MINES LTD
K.W.W.
Apr./88
DWG NO


N
Pond
Pond
ROAD
134000 W
136000 W
84000 N
88000 N
DEPTH (feet)
15+
10.0-14.9
5.0-9.9
0-4.9
• 1987/88 drilling
x 1970 drilling
TAILINGS AREA
DEPTH
UNITED KENO HILL MINES LTD.
BY
DATE
SCALE 1" = 400'
DRAWN
K.W.W.
Apr./88
COMPILED
DWG. NO.

Appendix D-3

Historic Tailings Reprocessing Assessment Reports

Scoping Testwork

on the

RECOVERY OF SILVER, LEAD, AND ZINC

on tailing material

from United Keno Hill Mines Limited

Sandy Sveinson

December 27, 1995

TABLE OF CONTENTS

INTRODUCTION

The following report summarizes the results of scoping testwork conducted on two tailing samples as requested by Linda Broughton, P.Eng., for United Keno Hill Mines Limited. The testwork included gravity separation and flotation tests to investigate the recovery of silver, lead, and zinc.

SUMMARY

1. Head Analysis

Representative head samples prepared from the two different tailing samples returned the following analyses:

TABLE 1 HEAD SAMPLE ANALYSES

	Sample #1a (Tailing Sample # 1)	Sample #1b (Tailing Sample # 1)	Sample # 2 (Tailing Sample # 2)
Silver (oz/ton)	3.03	3.85	2.45
Lead % (total lead)	0.65	0.84	0.48
Lead Oxide %	0.65	0.67	0.48
Zinc % (total zinc)	0.44	0.51	0.59
Zinc Oxide %	0.28	0.25	0.22
Iron %	9.2	9.6	8.2

It is significant to note that the oxide lead ranges from 80 -100% of the total lead, while the zinc oxide ranges from 37 - 64% of the total zinc.

In addition, semi-quantitative ICP scans were conducted on representative samples. Analyses were conducted on whole ore samples and on water from 25% solid solutions.

The results of the ICP analyses are presented in Tables 2 and 3 on the following page. The 25% solids solution represents approximately a 17:1 dilution ratio based on a moisture content of 15% in the tailings material. The concentration of heavy metals such as manganese and zinc, therefore, indicate significant leaching from the tailing.

TABLE 2
ICP ANALYSES FROM WHOLE ORE SAMPLES

Element	Sample #1	Sample #2
	ppm	ppm
Ag	72	55
Al	1,800	2,300
As	1,195	886
B	185	189
Ba	61	356
Be	< 1	< 1
Bi	< 3	< 3
Ca	2,500	3,700
Cd	46	65
Co	2	2
Cr	165	234
Cu	98	130
Fe	79,300	70,000
Hg	< 3	< 3
La	< 2	< 2
Mg	2,300	2,300
Mn	30,968	27,671
Mo	8	8
Na	100	100
Ni	14	16
P	324	312
Pb	5,575	3,959
Sb	74	55
Sr	5	11
Ti	100	100
V	3	4
W	36	15
Zn	3,953	4,941

TABLE 3
ICP ANALYSES FROM 25% SOLIDS SOLUTION

Element	Sample #1	Sample #2
	ppm	ppm
Ag	0.121	0.012
Al	1.300	1.840
As	0.590	0.260
Au	0.870	0.070
B	3.030	1.680
Ba	0.395	0.084
Be	0.008	0.004
Bi	< .05	< .05
Ca	259.800	512.700
Cd	2.910	0.792
Co	0.040	0.005
Cr	0.061	< .005
Cu	0.158	0.013
Fe	2.210	0.130
Hg	< .02	< .02
K	n/a	n/a
La	0.008	0.005
Mg	113.100	39.500
Mn	617.000	70.910
Mo	0.019	0.003
Na	21.000	5.000
Ni	0.047	0.008
P	1.260	0.680
Pb	0.150	0.170
Sb	< .02	< .02
Si	3.470	1.540
Sr	0.033	0.409
Ti	0.016	0.003
V	0.013	0.002
W	< .02	< .02
Zn	129.000	15.440

2. Gravity Separation Testwork

A single test was conducted on a 68.4 kilogram sample of tailing sample #2. A Falcon batch concentrator (6 inch) was used for the primary separation, followed by a Knelson batch concentrator (3 inch) to obtain the final concentrate.

The results are presented in Table 4.

TABLE 4 GRAVITY SEPARATION TEST ONE

FALCON/KNELSON CONCENTRATORS

Product	Weight (g)	Weight %	Assays g/t, %								Distribution %							
			Ag	Fe	Pb (total)	Pb (oxide	Zn (total)	Zn (oxide)	S (total)	Sulfides	Ag	Fe	Pb (total)	Pb (oxide	Zn (total)	Zn (oxide)	S (total)	Sulfides
2nd Knelson Conc	76	0.1	252.0	25.0	1.32	0.85	1.14	0.22	11.40	10.90	0.3	0.3	0.2	0.2	0.2	0.1	0.7	0.8
2nd Knelson Cl. Tails	98	0.1	117.0	16.8	0.86	0.56	0.83	0.23	5.10	4.80	0.2	0.2	0.2	0.2	0.2	0.1	0.4	0.4
1st Knelson Cl. Conc	172	0.2	176.6	20.4	1.06	0.69	0.97	0.23	7.88	7.49	0.5	0.5	0.4	0.4	0.4	0.2	1.0	1.2
1st Knelson Cl. Tails	584	0.8	94.6	15.5	0.90	0.66	0.84	0.25	2.50	2.23	0.9	1.3	1.3	1.3	1.1	0.8	1.1	1.2
Falcon Cl. Conc	736	1.0	113.7	16.5	0.94	0.67	0.87	0.24	3.76	3.46	1.4	1.7	1.6	1.7	1.4	1.0	2.1	2.4
Falcon Cl. Tails	1,986	2.6	75.4	10.7	0.47	0.31	0.63	0.19	2.28	2.14	2.5	3.0	2.2	2.1	2.7	2.1	3.5	4.0
Total Falcon Ro. Conc	2,722	3.6	85.8	12.3	0.60	0.41	0.69	0.20	2.68	2.50	3.8	4.8	3.9	3.8	4.0	3.1	5.6	6.4
Falcon Ro. Tails	65,278	86.8	80.9	9.3	0.56	0.40	0.60	0.24	1.66	1.34	86.9	85.9	87.1	88.2	84.3	87.6	83.7	82.7
Total Falcon Feed	68,000	90.4	81.1	9.4	0.56	0.40	0.60	0.24	1.72	1.39	90.7	90.7	90.9	92.0	88.4	90.7	89.3	89.1
Oversize Solids	7,190	9.6	78.2	9.1	0.53	0.33	0.75	0.23	1.95	1.60	9.3	9.3	9.1	8.0	11.6	9.3	10.7	10.9
Calculated Feed	75,190	100.0	80.8	9.4	0.56	0.39	0.62	0.24	1.74	1.41	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.00
Assay			79.5	10.1	0.55	0.39	0.68	0.22	1.96	1.69								

3. Flotation Testwork

The first two flotation tests were conducted as baseline tests on the two tailing samples, under identical conditions. The pH of the pulp was not adjusted. The tailing was not conditioned prior to flotation. The first rougher concentrate was collected with air flow only to the flotation cell. A second rougher concentrate was collected after the addition of frother. Potassium amyl xanthate (PAX) and frother were added and a third rougher concentrate was collected. The fourth rougher concentrate was collected after the addition of 300 g/t copper sulfate. The fifth and final rougher concentrate was collected after sulfidizing with 1000 g/t NaHS. Additions of frother and collector were added as required in the fourth and fifth rougher stages. Each of the rougher concentrates was then cleaned separately, with no further addition of reagents. A sample of the final rougher tailing was collected and wet screened at 325 mesh. The +325 mesh material was later dry screened.

Results of Tests One and Two are summarized in Tables 5 and 6. The results of the +325 mesh screen fraction analyses are presented in the section entitled Details of Tests.

The assay results indicating 100% lead oxides in both samples and 64% (sample #1) and 37% (sample #2) zinc oxides, help explain the poor recoveries of silver, lead, and zinc.

Flotation Tests Three and Four were conducted on tailing sample #1. In Test Three, the tailing material was first re-ground for 30 seconds. The material was then conditioned for one minute with 250 g/t copper sulfate. Two stage additions of Aerophine 3418A and frother were used to collect a bulk rougher concentrate, which was cleaned. Sulfidizing was carried out with stage additions of NaHS (three stages of 200 g/t, followed by two stages of 1000 g/t) to collect a second rougher concentrate.

In Test Four, the tailing material was reground for 5 minutes. The material was then conditioned for one minute with 250 g/t copper sulfate. Two stage additions of Aerophine 3418A and frother were added to collect a bulk rougher concentrate, which was cleaned. Sulfidizing was carried out with stage additions of NaHS (two stages of 200 g/t, followed by two stages of 1000 g/t) to collect a second rougher concentrate.

The results of Tests Three and Four are summarized in Tables 7 and 8.

Again, the high percent of lead oxides (80%) and zinc oxides (49%) obviously contributed to the poor metal recoveries. The five minute re-grind in Test Four appeared to contribute to improvement in recoveries.

TABLE 5 FLOTATION TEST ONE

SAMPLE # 2

Product	Weight (g)	Wt. %	Assays, oz/t, %				% Distribution			
			Ag	Pb	Zn	Fe	Ag	Pb	Zn	Fe
Ro. Conc-1	6.6	0.8	7.47	1.40	1.70	11.00	2.3	2.0	2.2	1.0
Cl. Conc-2	2.1	0.2	19.50	0.42	17.60	25.10	1.9	0.2	7.4	0.7
Cl. Tail-2	27.0	3.1	11.40	0.87	4.50	21.20	14.5	5.2	24.3	8.0
Ro. Conc-2	29.1	3.4	11.98	0.84	5.45	21.48	16.4	5.4	31.7	8.8
Cl. Conc-3	2.8	0.3	11.70	0.62	8.90	27.30	1.5	0.4	5.0	1.1
Cl. Tail-3	17.6	2.0	6.53	1.11	1.50	11.80	5.4	4.3	5.3	2.9
Ro. Conc-3	20.4	2.4	7.24	1.04	2.52	13.93	7.0	4.7	10.3	4.0
Cl. Conc-4	1.2	0.1	8.63	1.37	1.55	10.60	0.5	0.4	0.4	0.2
Cl. Tail-4	17.3	2.0	5.72	1.21	1.05	10.10	4.7	4.6	3.6	2.4
Ro. Conc-4	18.5	2.2	5.91	1.22	1.08	10.13	5.1	5.0	4.0	2.6
Total Ro. Conc.	74.6	8.7	8.78	1.04	3.23	15.67	30.8	17.0	48.2	16.4
Ro. Tails	785.8	91.3	1.87	0.48	0.33	7.60	69.2	83.0	51.8	83.6
Head (calc)	860.4	100.0	2.47	0.53	0.58	8.30	100.0	100.0	100.0	100.0
Head (assay)			2.45	0.48	0.59	8.20				

Head (assay)	% PbO	0.48
	% ZnO	0.22
% Moisture	12.0%	

TABLE 6 FLOTATION TEST TWO

SAMPLE # 1

Product	Weight (g)	Wt. %	Assays, oz/t, %				% Distribution			
			Ag	Pb	Zn	Fe	Ag	Pb	Zn	Fe
Ro. Conc-1	4.7	0.6	7.12	1.09	0.77	13.20	1.2	0.8	1.0	0.8
Cl. Conc-2	2.0	0.2	7.82	1.38	1.03	17.20	0.5	0.4	0.5	0.4
Cl. Tail-2	21.2	2.5	5.83	1.19	0.84	15.20	4.0	3.9	4.6	3.9
Ro. Conc-2	23.2	2.8	6.00	1.21	0.86	15.37	4.5	4.3	5.1	4.3
Cl. Conc-3	3.8	0.5	20.90	0.62	1.85	32.20	2.6	0.4	1.8	1.5
Cl. Tail-3	34.1	4.1	10.60	1.08	1.91	3.30	11.7	5.7	16.9	1.4
Ro. Conc-3	37.9	4.5	11.63	1.03	1.90	6.20	14.3	6.0	18.7	2.8
Cl. Conc-4	0.9	0.1	11.20	1.14	4.28	16.90	0.3	0.2	1.0	0.2
Cl. Tail-4	19.2	2.3	7.70	1.16	1.65	16.00	4.8	3.4	8.2	3.7
Ro. Conc-4	20.1	2.4	7.86	1.16	1.77	16.04	5.1	3.6	9.2	3.9
Cl. Conc-5	0.8	0.1	5.27	1.10	0.77	13.20	0.1	0.1	0.2	0.1
Cl. Tail-5	14.5	1.7	5.25	1.10	0.78	13.90	2.5	2.5	2.9	2.4
Ro. Conc-5	15.3	1.8	5.25	1.10	0.78	13.86	2.6	2.6	3.1	2.5
Total Ro. Conc.	101.2	12.1	8.44	1.11	1.42	11.78	27.7	17.3	37.1	14.3
Ro. Tails	736.3	87.9	3.03	0.73	0.33	9.70	72.3	82.7	62.9	85.7
Head (calc)	837.5	100.0	3.68	0.78	0.46	9.95	100.0	100.0	100.0	100.0
Head (assay)			3.03	0.65	0.44	9.20				

Head (assay) % PbO 0.65
% ZnO 0.28
% Moisture 14.3%

TABLE 7 FLOTATION TEST THREE

SAMPLE # 1

Product	Weight (g)	Wt. %	Assays, oz/t, %				% Distribution			
			Ag	Pb	Zn	Fe	Ag	Pb	Zn	Fe
Cl. Conc-1	12.8	1.3	17.40	1.29	5.00	24.90	5.2	2.0	13.7	3.2
Cl. Tail-1	41.8	4.4	7.70	1.34	1.50	19.30	7.5	6.7	13.5	8.2
Ro. Conc-1	54.6	5.7	9.97	1.33	2.32	20.61	12.7	8.7	27.2	11.4
Ro. Conc-2	64.4	6.8	6.90	1.52	0.99	16.60	10.4	11.7	13.7	10.8
Total Ro. Conc	119.0	12.5	8.31	1.43	1.60	18.44	23.1	20.3	40.9	22.2
Ro. Tails	834.1	87.5	3.95	0.80	0.33	9.20	76.9	79.7	59.1	77.8
Head (calc)	953.1	100.0	4.49	0.88	0.49	10.35	100.0	100.0	100.0	100.0
Head (assay)			3.85	0.84	0.51	9.60				

Head (assay)	% PbO	0.67
	% ZnO	0.25

TABLE 8 FLOTATION TEST FOUR

SAMPLE # 1

Product	Weight (g)	Wt. %	Assays, oz/t, %				% Distribution			
			Ag	Pb	Zn	Fe	Ag	Pb	Zn	Fe
Cl. Conc-1	10.7	1.1	32.10	1.13	6.00	26.30	8.4	1.4	12.0	2.7
Cl. Tail-1	50.0	5.2	8.85	1.25	1.47	18.90	11.0	7.4	13.9	9.3
Ro. Conc-1	60.7	6.3	12.81	1.22	2.24	20.04	19.5	8.8	25.9	12.0
Ro. Conc-2	80.8	8.5	7.10	1.35	1.00	14.80	14.4	12.9	15.4	11.8
Total Ro. Conc	141.5	14.8	9.55	1.29	1.53	17.05	33.8	21.7	41.2	23.8
Ro. Tails	814.5	85.2	3.25	0.81	0.38	9.50	66.2	78.3	58.8	76.3
Head (calc)	956.0	100.0	4.18	0.88	0.55	10.62	100.0	100.0	100.0	100.0
Head (assay)			4.10	0.89	0.51	9.80				

4. Discussion and Recommendations

Gravity separation using a Falcon concentrator followed by a Knelson concentrator resulted in a recovery of 4% of the silver. Earlier gravity testwork conducted by Lakefield Research (October 1994) indicated higher recoveries of silver using a Knelson concentrator with cleaning on a Mozley separator. It is recommended that further tests be conducted using a Knelson concentrator to determine if silver recovery can be improved.

The dry screen analyses of the +325 mesh wet screen material from the flotation tests indicate significant metal losses in the fine fractions.

Flotation tests resulted in up to 31% silver recovery. However, the assays of the samples returned values of 80% - 100% lead oxides and 37% - 64% zinc oxides, indicating that flotation of this material is unlikely to be a viable option for metal recovery.

SAMPLE PREPARATION

Five pails of tailing material from United Keno Hill were received at the Department of Mining and Mineral Processing lab at the University of British Columbia in October 1995. The sample material consisted of tailing from two different sample sites as shown in Figure 1. The initial total weights of the samples were as follows:

Sample #1	58.4 kilograms
Sample #2	39.6 kilograms

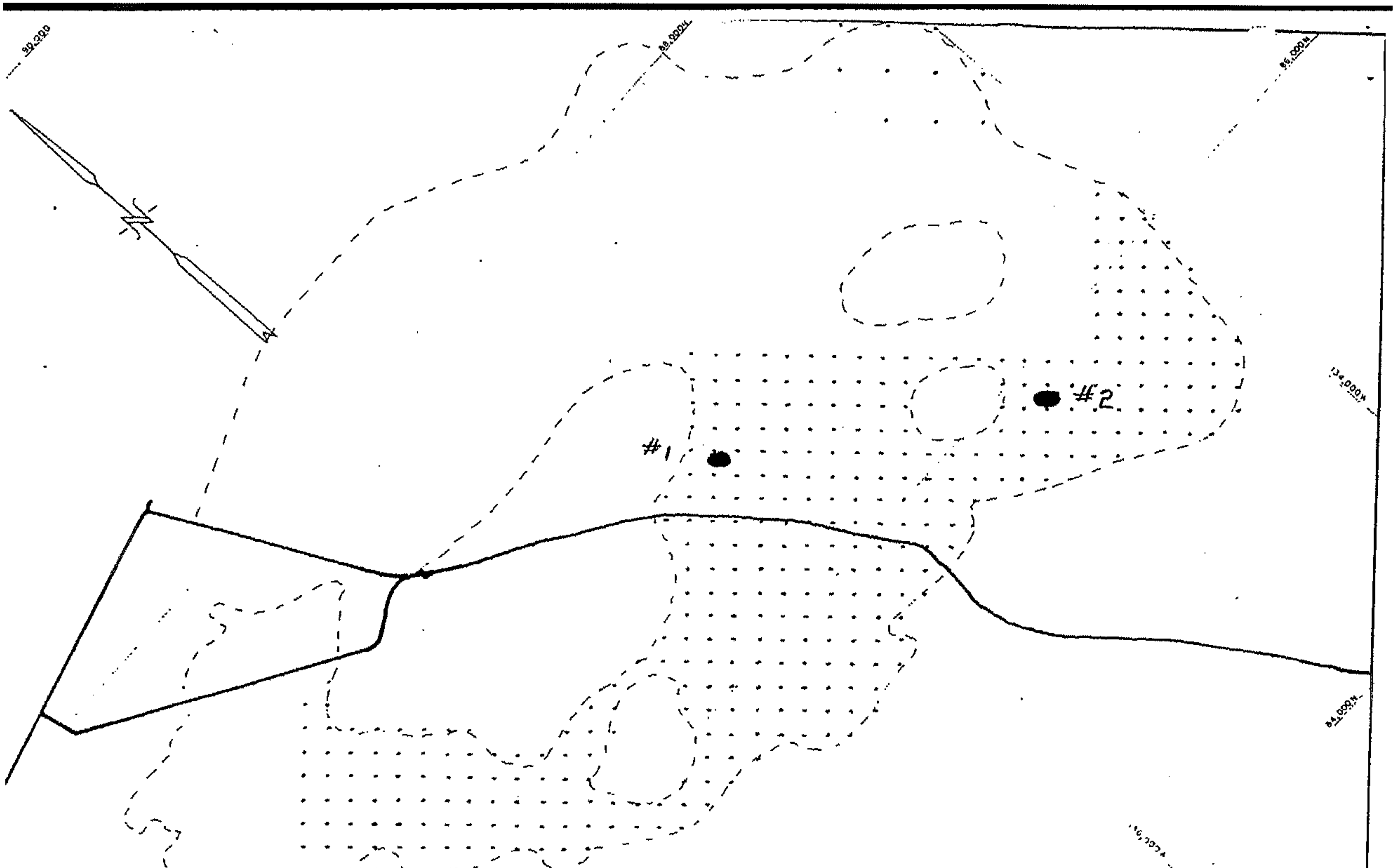
A further four pails of tailing material (sample #2), weighing 68.4 kilograms, were shipped directly to Process Research Associates lab in Vancouver, where the gravity separation test was conducted.

The material shipped to UBC was screened and split using cone and quartering followed by riffling to obtain test charges of approximately 1 and 2.5 kilogram sizes.

Moisture analysis of the two samples determined the moisture content of the samples to be 14.3% and 12% respectively for Sample #1 and #2. Due to the high moisture content, the material was dried prior to screen analysis. The screen analyses of the two samples are presented in the following section.

Screen fractions were conducted on both samples and the results are summarized in the next section.

Representative head samples were removed for ICP analyses of whole ore. As well, a 25% solids solution was made up from a sample of each material. The slurry was shaken for 10 minutes and the material was then allowed to settle. A sample of the water was then decanted off for ICP analysis.



TAILINGS AREA
SAMPLE LOCATIONS

UNITED KENO HILL MINES LTD.

	BY	DATE	SCALE 1" = 400'
DRAWN	K.W.W.	Mar./87	
COMPILED			DWG. NO. FIG 1

DETAILS OF TESTS

Screen Analyses

1. Head Samples

Procedure: The material was screened for 20 minutes using a Ro-Tap machine. Screen fractions were individually weighed.

Feed: 250 grams of tailing material

Results and graphs of the screen analyses are presented in Tables 9 and 10.

2. Dry Screen Analyses from +325 Mesh Wet Screening of Rougher Tailings in Flotation Tests One and Two

Procedure: The +325 mesh material from wet screening of rougher tailings was dried and screened for 25 minutes using a Ro-Tap machine. Screen fractions were weighed.

Results and graphs of the screen analyses for Flotation Tests One and Two are presented in Tables 11 - 14.

TABLE 9

SAMPLE #1 SCREEN ANALYSIS

Size		Weight	%	Cumulative	
Mesh	Microns	(g)	Retained	% Retained	% Passing
65	250	37.0	15.9	15.9	84.1
100	149	37.4	16.1	32.0	68.0
140	105	39.0	16.8	48.8	51.2
200	74	38.7	16.6	65.4	34.6
270	53	26.2	11.3	76.7	23.3
325	44	11.9	5.1	81.8	18.2
400	37	15.6	6.7	88.5	11.5
Pan	-37	26.7	11.5	100.0	
Total		232.5	100.0		

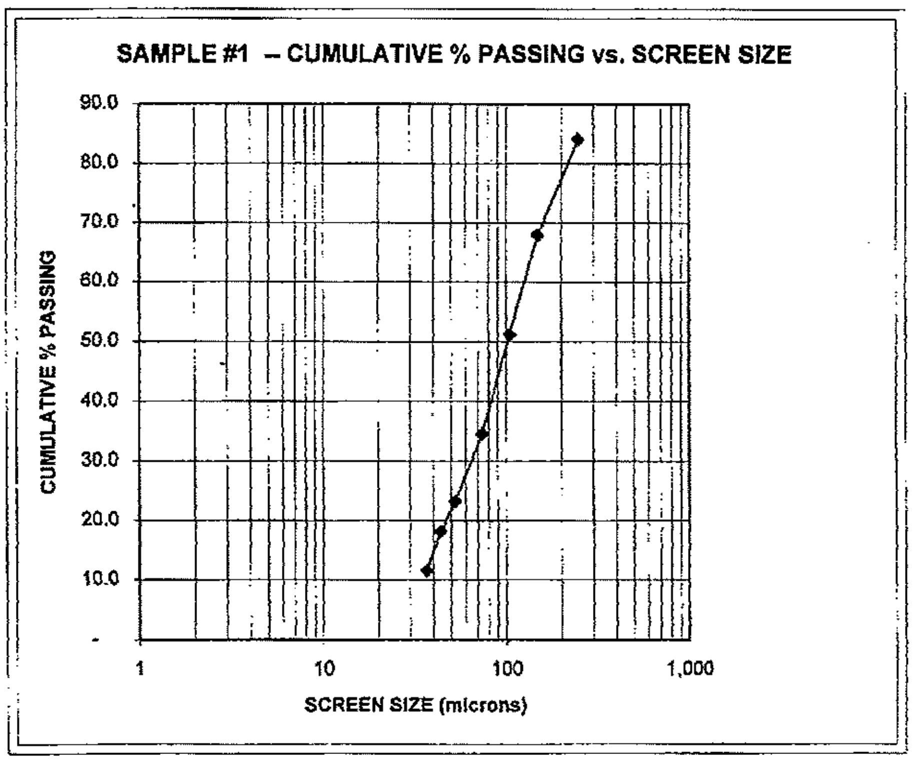


TABLE 10

SAMPLE #2 **SCREEN ANALYSIS**

Size		Weight	%	Cumulative	
Mesh	Microns	(g)	Retained	% Retained	% Passing
65	250	48.4	18.4	18.4	81.6
100	149	35.0	13.3	31.7	68.3
140	105	32.0	12.1	43.8	56.2
200	74	42.7	16.2	60.0	40.0
270	53	46.6	17.7	77.7	22.3
325	44	11.5	4.4	82.0	18.0
400	37	11.6	4.4	86.5	13.5
Pan	-37	35.7	13.5	100.0	
Total		263.5	100.0		

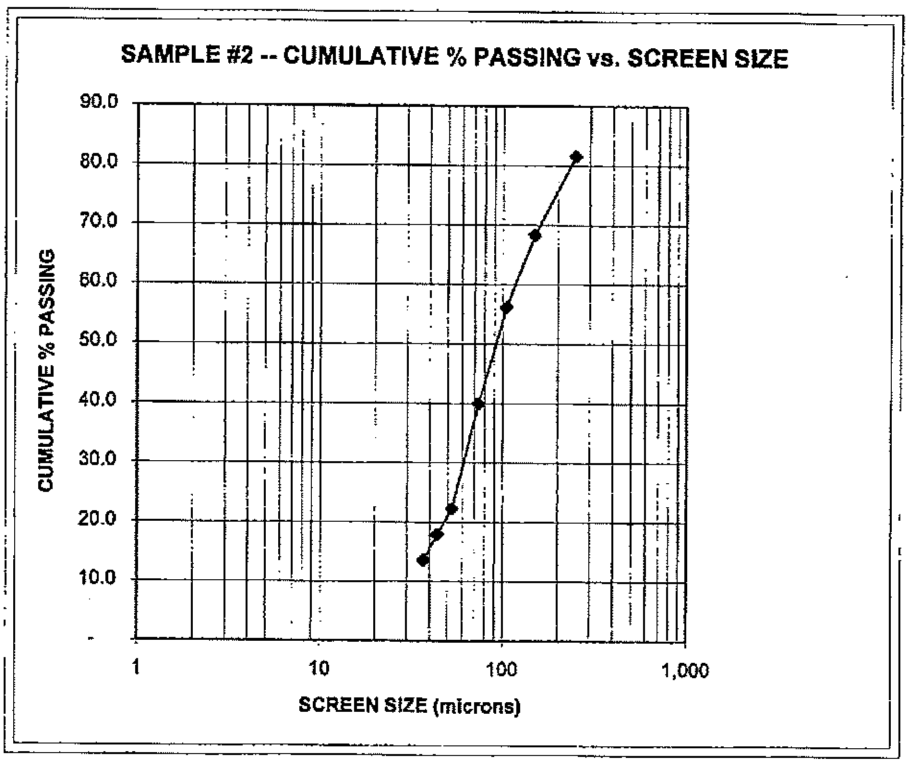


TABLE 11 SCREEN ANALYSIS

TEST ONE SAMPLE #2 ROUGHER TAILING
+ 325 MESH

Size		Weight	%	Cumulative	
Mesh	Microns	(g)	Retained	% Retained	% Passing
65	250	7.4	7.9	7.9	92.1
100	149	12.7	13.6	21.6	78.4
140	105	11.8	12.7	34.2	65.8
200	74	20.0	21.5	55.7	44.3
270	53	11.1	11.9	67.6	32.4
325	44	13.3	14.3	81.9	18.1
Pan	-44	16.9	18.1	100.0	
Total		93.2	100.0		

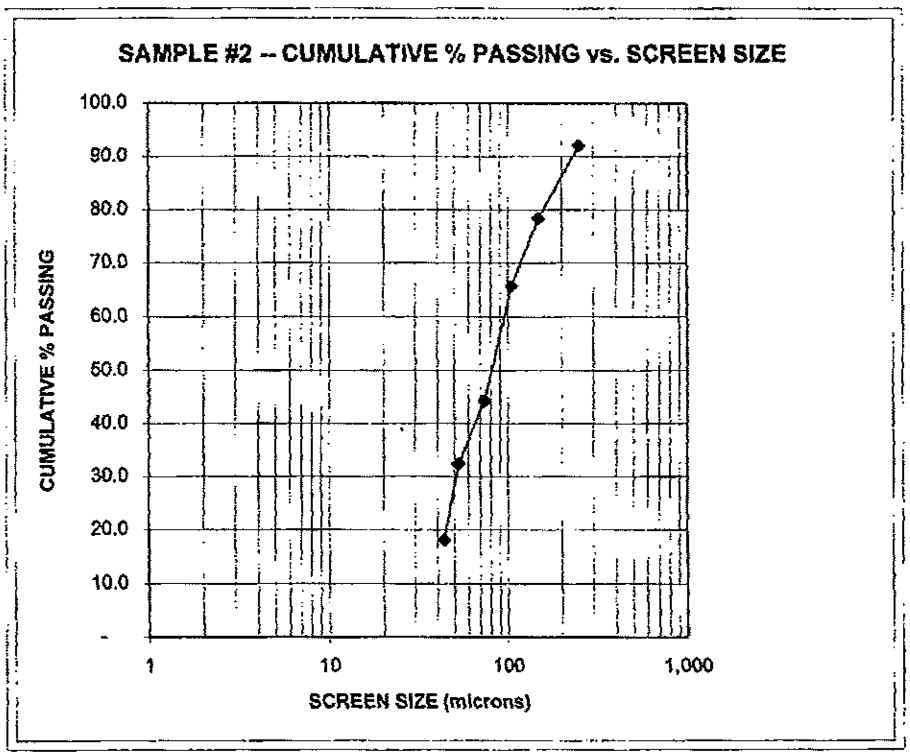


TABLE 12

TEST ONE SAMPLE #2 SIZE FRACTION ANALYSIS ROUGHER TAILING FROM +325 MESH WET SCREEN FRACTION

Product	Weight (g)	Weight %	Assays oz/t, %				% Distribution			
			Ag	Pb	Zn	Fe	Ag	Pb	Zn	Fe
65 Mesh	7.4	7.9	0.70	0.13	0.08	3.40	4.7	3.8	2.9	4.2
100 Mesh	12.7	13.6	0.93	0.16	0.40	4.70	10.7	8.1	25.3	10.0
150 Mesh	11.8	12.7	1.28	0.23	0.32	6.60	13.8	10.8	18.9	13.1
200 Mesh	20.0	21.5	1.28	0.24	0.10	6.60	23.3	19.1	10.0	22.2
270 Mesh	11.1	11.9	0.93	0.26	0.11	6.90	9.4	11.5	6.1	12.8
325 Mesh	13.3	14.3	1.52	0.30	0.16	6.90	18.4	15.9	10.6	15.4
Minus 325	16.9	18.1	1.28	0.46	0.31	7.90	19.6	30.8	26.1	22.3
	93.2	100.0	1.18	0.27	0.21	6.40	100.0	100.0	100.0	100.0

TABLE 13 SCREEN ANALYSIS

TEST TWO | SAMPLE #1 | ROUGHER TAILING + 325 MESH

Size		Weight	%	Cumulative	
Mesh	Microns	(g)	Retained	% Retained	% Passing
60.0	250	0.9	0.9	0.9	99.1
100.0	149	6.2	6.1	7.0	93.0
150.0	105	12.9	12.7	19.7	80.3
200.0	74	19.6	19.3	39.0	61.0
270.0	53	17.3	17.0	56.0	44.0
325.0	44	20.9	20.6	76.6	23.4
Pan	-44	23.8	23.4	100.0	
Total		101.6	100.0		

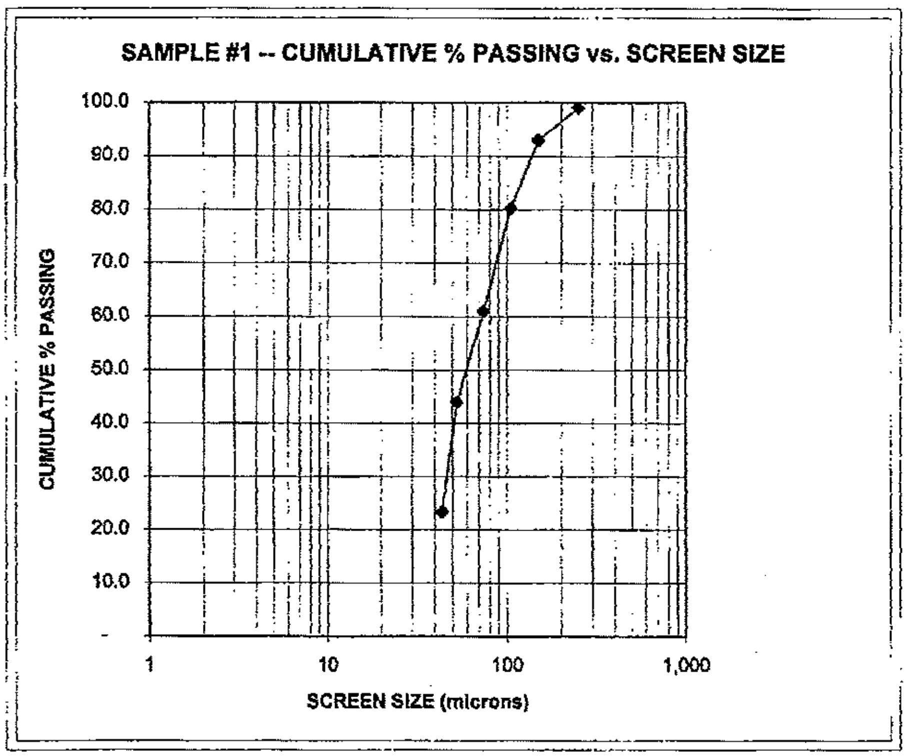


TABLE 14

TEST TWO SAMPLE #1 SIZE FRACTION ANALYSIS ROUGHER TAILING
FROM +325 MESH WET SCREEN FRACTION

Product	Weight (g)	Weight %	Assays oz/t, %				% Distribution			
			Ag	Pb	Zn	Fe	Ag	Pb	Zn	Fe
65 Mesh	0.9	0.9	2.80	0.74	0.33	7.40	1.0	1.1	1.5	0.8
100 Mesh	6.2	6.1	1.40	0.28	0.10	3.30	3.4	2.8	3.1	2.4
140 Mesh	12.9	12.7	1.98	0.28	0.10	5.50	10.0	5.9	6.6	8.4
200 Mesh	19.6	19.3	2.45	0.38	0.14	7.70	18.8	12.1	13.9	17.8
270 Mesh	17.3	17.0	2.92	0.49	0.16	8.40	19.7	13.8	14.0	17.1
325 Mesh	20.9	20.6	2.57	0.68	0.22	8.90	21.0	23.2	23.4	22.0
Minus 325	23.8	23.4	2.80	1.06	0.31	11.20	26.0	41.1	37.4	31.4
	101.6	100.0	2.52	0.60	0.19	8.33	100.0	100.0	100.0	100.0

Gravity Separation Test

November 2, 1995

Purpose: To investigate the recovery of silver and lead by gravity separation.

Procedure: The material was concentrated in three passes using a B6 Falcon concentrator, followed by two passes in a Knelson concentrator (3 inch). On the initial pass through the Falcon concentrator, oversize material was screened off. All products were dried and weighed and submitted for assay.

Feed: 68.4 kilograms of Sample #2 tailing material

Results of the gravity separation materials balance are summarized in Table 15.

TABLE 15 GRAVITY SEPARATION TEST ONE

FALCON/KNELSON CONCENTRATORS

Product	Weight (g)	Weight %	Assays g/t, %							
			Ag	Fe	Pb (total)	Pb (oxide	Zn (total)	Zn (oxide	S (total)	Sulfides
2nd Knelson Conc	76	0.1	252.0	25.0	1.32	0.85	1.14	0.22	11.40	10.90
2nd Knelson Cl. Tails	96	0.1	117.0	16.8	0.86	0.56	0.63	0.23	5.10	4.80
1st Knelson Cl. Conc	172	0.2	176.6	20.4	1.06	0.69	0.97	0.23	7.88	7.49
1st Knelson Cl. Tails	584	0.8	94.6	15.5	0.90	0.66	0.84	0.26	2.50	2.23
Falcon Cl. Conc	736	1.0	113.7	16.5	0.94	0.67	0.87	0.24	3.76	3.46
Falcon Cl. Tails	1,986	2.6	75.4	10.7	0.47	0.31	0.63	0.19	2.28	2.14
Total Falcon Ro. Conc	2,722	3.6	85.8	12.3	0.60	0.41	0.69	0.20	2.68	2.50
Falcon Ro. Tails	65,278	86.8	80.9	9.3	0.56	0.40	0.60	0.24	1.68	1.34
Total Falcon Feed	68,000	90.4	81.1	9.4	0.56	0.40	0.60	0.24	1.72	1.39
Oversize Solids	7,190	9.6	78.2	9.1	0.53	0.33	0.75	0.23	1.95	1.60
Calculated Feed	75,190	100.0	80.8	9.4	0.56	0.39	0.62	0.24	1.74	1.41
Assay			79.5	10.1	0.55	0.39	0.68	0.22	1.96	1.69

Product	Distribution %							
	Ag	Fe	Pb (total)	Pb (oxide	Zn (total)	Zn (oxide	S (total)	Sulfides
2nd Knelson Conc	0.3	0.3	0.2	0.2	0.2	0.1	0.7	0.8
2nd Knelson Cl. Tails	0.2	0.2	0.2	0.2	0.2	0.1	0.4	0.4
1st Knelson Cl. Conc	0.5	0.5	0.4	0.4	0.4	0.2	1.0	1.2
1st Knelson Cl. Tails	0.9	1.3	1.3	1.3	1.1	0.8	1.1	1.2
Falcon Cl. Conc	1.4	1.7	1.6	1.7	1.4	1.0	2.1	2.4
Falcon Cl. Tails	2.5	3.0	2.2	2.1	2.7	2.1	3.5	4.0
Total Falcon Ro. Conc	3.8	4.8	3.9	3.8	4.0	3.1	5.6	6.4
Falcon Ro. Tails	86.9	85.9	87.1	88.2	84.3	87.6	83.7	82.7
Total Falcon Feed	90.7	90.7	90.9	92.0	88.4	90.7	89.3	89.1
Oversize Solids	9.3	9.3	9.1	8.0	11.6	9.3	10.7	10.9
Calculated Feed Assay	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.00

Flotation Test One

Sample #2

November 16, 1995

Purpose: To investigate recovery of silver, lead and zinc from a tailing sample.

Procedure: Tailing material was floated in a 2 litre Agitair cell. Agitation speed was set between 700 - 800 rpm. and air flow was set between 10 - 15 litres/min. All rougher concentrates were individually cleaned. A sample from the final rougher tailing was wet screened at 325 mesh and the +325 mesh material was dry screened for fractional analysis. All products were filtered, dried, weighed, and submitted for assay.

Feed: 1 kilogram of Sample #2 tailing material

Stage	Reagent Addition, g/t				Time, minutes			
	CuS04	PAX	NaHS	Dowfroth 250 (drop)	Grind	Condition	Froth Flotation	pH
Ro. Conc. 1	-	-	-	1	-	-	3	6.5
Ro. Conc. 2	-	50	-	1	-	-	3	6.5
Ro. Conc. 3	300	25	-	2	-	-	4	6.6
Ro. Conc. 4	-	25	1000	1	-	-	5	6.5
Cl. Conc. 2							3	6.5
Cl. Conc. 3							3.5	6.5
Cl. Conc. 4							4	6.5
Totals	300	100	1000	5			25.5	

Flotation Test Two

Sample #1

November 16, 1995

<u>Purpose</u>: To investigate recovery of silver, lead and zinc from a tailing sample.

<u>Procedure</u>: Tailing material was floated in a 2 litre Agitair cell. Agitation speed was set between 700 - 800 rpm. and air flow was set between 10 - 15 litres/min. All rougher concentrates were individually cleaned. A sample from the final rougher tailing was wet screened at 325 mesh and the +325 mesh material was dry screened for fractional analysis. All products were filtered, dried, weighed, and submitted for assay.

<u>Feed</u>: 1 kilogram of Sample #2 tailing material

Stage	Reagent Addition, g/t				Time, minutes			
	CuS04	PAX	NaHS	Dowfroth 250 (drop)	Grind	Condition	Froth Flotation	pH
Ro. Conc. 1	-	-	-	-	-	-	2.5	6
Ro. Conc. 2	-	-	-	1	-	-	4	6
Ro. Conc. 3	-	50	-	1	-	-	4.5	6.2
Ro. Conc. 4	300	25	-	1	-	-	5	6.2
Ro. Conc. 5	-	25	1000	1	-	-	3.5	6.2
Cl. Conc. 2							2.5	6.2
Cl. Conc. 3							2.5	6.2
Cl. Conc. 4							2	6.2
Cl. Conc. 5							2	6.2
Totals	300	100	1000	4			28.5	

Flotation Test Three

Sample #1

November 23, 1995

Purpose: To investigate recovery of silver, lead and zinc from a tailing sample, using a 30 second re-grind prior to flotation.

Procedure: Tailing material was reground in a rod mill for 30 seconds. The slurry was then floated in a 2 litre Agitair cell. Agitation speed was set between 700 - 800 rpm. and air flow was set between 10 - 15 litres/min. The material was conditioned for one minute with copper sulfate. Stage additions of collector and frother were used to collect a first rougher concentrate. Sulfidizing was done in stages to collect a second rougher concentrate. The first rougher concentrate was cleaned. All products were filtered, dried, weighed, and submitted for assay.

Feed: 1 kilogram of Sample #1 tailing material

Stage	Reagent Addition, g/t				Time, minutes			
	CuS04	3418A	NaHS	Dowfroth 250 (drop)	Grind	Condition	Froth Flotation	pH
Ro. Conc. 1	250	50	-	3	0.5 -	1	5	7.4
Ro. Conc. 2	-	50	2600	3	-	-	8	7.1
Cl. Conc. 2							3	7
Totals	250	100	2600	6	0.5	1	16	

Flotation Test Four

Sample #1

November 27, 1995

Purpose: To investigate recovery of silver, lead and zinc from a tailing sample, using a 5 minute re-grind prior to flotation.

Procedure: Tailing material was reground in a rod mill for 5 minutes. The slurry was then floated in a 2 litre Agitair cell. Agitation speed was set between 700 - 800 rpm. and air flow was set between 10 - 15 litres/min. The material was conditioned for one minute with copper sulfate. Stage additions of collector and frother were used to collect a first rougher concentrate. Sulfidizing was done in stages to collect a second rougher concentrate. The first rougher concentrate was cleaned. All products were filtered, dried, weighed, and submitted for assay.

Feed: 1 kilogram of Sample #1 tailing material

Stage	Reagent Addition, g/t				Time, minutes			
	CuS04	3418A	NaHS	Dowfroth 250 (drop)	Grind	Condition	Froth Flotation	pH
Ro. Conc. 1	250	50	-	3	5 -	1	4	8
Ro. Conc. 2	-	50	2400	3	-	-	8	7.4
Cl. Conc. 2							2	7.4
Totals	250	100	2400	6	5	1	14	

Appendix D-4

Historic Tailings Reprocessing Assessment Reports

14-67

canada/yukon economic
development agreement

INDIAN AND NORTHERN AFFAIRS CANADA
NORTHERN AFFAIRS: YUKON REGION

Open File 1996-3(T)

INVESTIGATION INTO THE REPROCESSING OF ELSA TAILINGS

for

UNITED KENO HILL MINES LIMITED
ELSA, YUKON

by

G. HAWTHORN, P. ENG.

Canada

Yukon
Government

Exploration and Geological Services
Indian and Northern Affairs Canada
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UNITED KENO HILL MINES LIMITED

INVESTIGATION

INTO THE

REPROCESSING OF ELSA TAILING

ELSA, YUKON

G.HAWTHORN, P.ENG

MARCH, 1996

ABSTRACT

This project was undertaken to evaluate the potential to economically recover additional metal values from the 4.6 million tons of Keno Hill flotation tailing which grades 3 - 4 oz/t Ag, 0.8 % lead, and 0.9 % Zn.

The study determined that the higher grade portion of the tailing, containing some 1.0 million tons at 5.35 oz/t Ag, responds well to cyanide heap leaching with a silver recovery of 50 - 60 %. The testing data suggests that heap leaching is economically feasible at the current silver price of $ US 5.50 / ounce and a currency exchange rate of 1.35 in favour of the US dollar.

The response to gravity and flotation concentration was poor.

THIS PROJECT WAS FUNDED BY UKHM (50 %), DEPARTMENT OF INDIAN AND NORTHERN AFFAIRS (35 %), AND GOVERNMENT OF YUKON (15 %).

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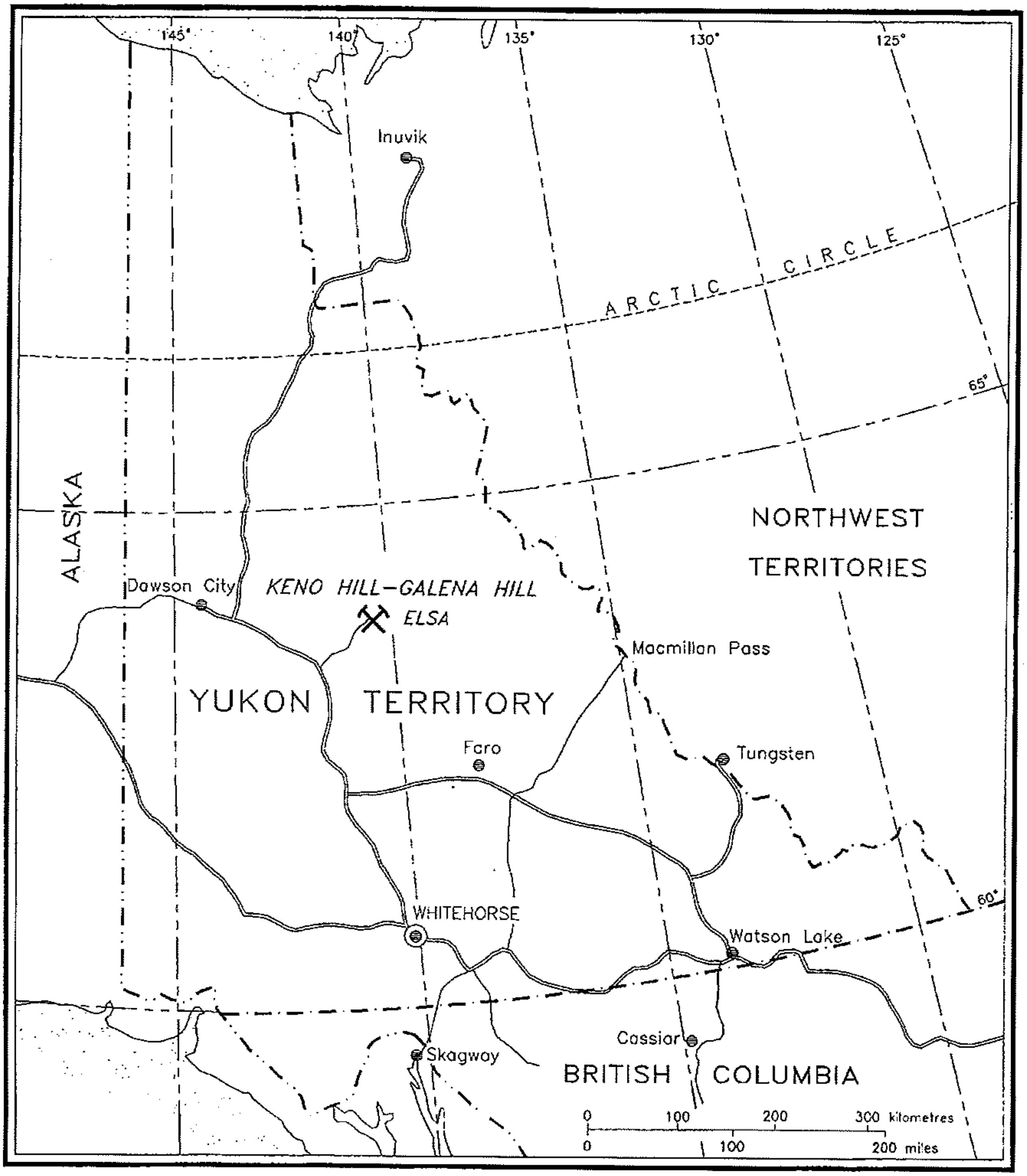
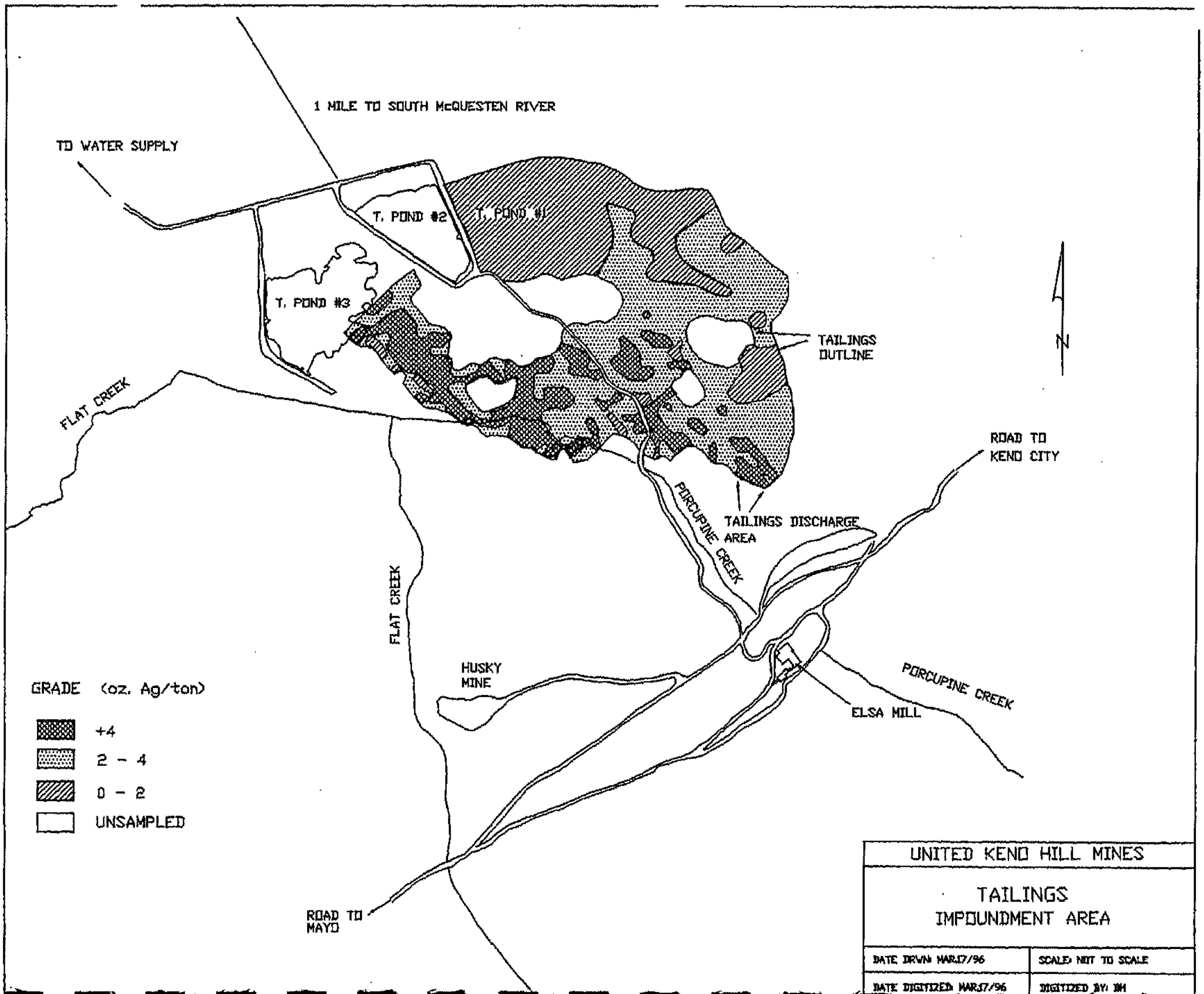


145°
140°
135°
130°
125°
Inuvik
ARCTIC CIRCLE
65°
ALASKA
NORTHWEST
TERRITORIES
Dawson City
KENO HILL–GALENA HILL
ELSA
Macmillan Pass
YUKON TERRITORY
Faro
Tungsten
60°
WHITEHORSE
Watson Lake
Cassiar
Skagway
BRITISH COLUMBIA
0 100 200 300 kilometres
0 100 200 miles



1 MILE TO SOUTH McQUESTEN RIVER
TO WATER SUPPLY
T. POND #2
T. POND #1
T. POND #3
TAILINGS OUTLINE
FLAT CREEK
ROAD TO KENO CITY
PORCUPINE CREEK
TAILINGS DISCHARGE AREA
FLAT CREEK
HUSKY MINE
PORCUPINE CREEK
ELSA MILL
GRADE (oz. Ag/ton)
+4
2 - 4
0 - 2
UNSAMPLED
ROAD TO MAYO
N
UNITED KENO HILL MINES
TAILINGS IMPOUNDMENT AREA
DATE DRWN: MAR.17/96
SCALE: NOT TO SCALE
DATE DIGITIZED: MAR.17/96
DIGITIZED BY: JH

Both flotation and gravity concentration were evaluated. Although some encouragement was provided in previous testing, the current investigators are of the opinion that, from a global perspective, lead and zinc cannot be economically recovered using contemporary technology. This has occurred principally because of high levels of oxidation of both the lead (at 65 - 100 %) and zinc (at 37 - 64 % based upon limited data). The data suggests that the majority of the oxidation occurred prior to mining with perhaps a small increment subsequent to processing.

Because of this, the intended program was substantially diminished, and was subsequently limited to the investigation of cyanide heap leaching.

2.0 HISTORICAL REVIEW

During the 1980's, UKHM conducted two internal studies (Watson et al, and Lockstein) to determine the characteristics of the tailing and to identify the distribution of metal values.

The following data is from the Watson report.

Source	Tons	Ag oz/t	Ag Ounces
1987/88 Drilling	1,699,405	4.45	7,562,352
1970 Drilling	2,156,175	1.91	4,118,294
1950 Terraced	15,500	10.70	165,850
New Discharge	70,000	5.50	385,000
Under 2 nd Pond	108,590	4.63	502,772
Total	4,049,670	3.14	12,734,268
Production reports	4,049,670	3.98	16,117,687

Note that the above data indicates a lower silver grade from the field sampling than from the production reports. There is no adequate explanation for this discrepancy. Both the silver grade and the available tonnages will be confirmed by UKHM prior to any production decision.

Also from the Watson report, the following metal analyses were obtained.

Source	Ag oz/t	Pb %	NSPb %	Zn %	Fe %	Cd %
1987/88 Drilling	4.45	1.06	0.84	1.27	12.03	0.03
1970 Drilling	1.91	0.50	0.39	0.55	5.20	0.01
Total	3.03	0.75	0.49	0.87	8.21	0.01

There is an error in the tabulation of the NSPb data, which indicates that for the 1987/88 drilling program the lead was 78 % oxidized. The correct value is about 65 %.

During the 60 year mine life which is represented by the impounded tailing, ores from 35 mines were processed at the same millsite. Although all of these ores were exploited for their silver, lead, and zinc content, there was considerable variation in the metallic mineralogy.

In addition to clean sulphide ores from underground mining, several of the mines had open pit production of partially oxidized ores. The metallic mineralogy consisted of galena (lead sulphide), sphalerite (zinc sulphide), and pyrite (iron sulphide) as the dominant metallic sulphides, with the "oxides" represented by cerussite (lead carbonate) and minor anglesite (lead sulphate). The oxide zinc minerals included hemimorphite and smithsonite.

Silver was represented by several minerals: major electrum (native silver), argentite (silver sulphide), and to a lesser extent as solid solution in anglesite and limonite/manganite. The poor response of the silver in the flotation tailing to cyanidation, with a 50 % recovery, is principally explained by solid solutions. The silver manganese minerals are particularly resistant to direct cyanidation, but after treatment with sulphurous acid, leach very well with silver recoveries exceeding 90 %.

The mining of open pit ores was accelerated during the last 10 years of operation due to a shortage of higher grade and more readily processed sulphide ores from underground mining.

In the last full year of operation, for example, 25 % of the lead in the mill feed was present as oxides, mainly as cerussite.

By the early 1980's, the recovery of zinc was no longer practised, so the operators had no incentive to determine the mineral distribution of zinc in the plant feed or tailing.

The presence of oxidized lead became an important consideration, so in the later years the metallurgical statement included oxide lead analyses, presented as NSPb (non-sulphide lead).

For the current study, samples were collected from the accumulated tailing, and were analyzed for oxide lead and zinc using the acetic acid soluble procedure. These analyses (see Sveinson) indicated that 80 - 100 % of the lead is oxidized as is 37 - 64 % of the zinc.

From a processing perspective, the common sulphide minerals galena and sphalerite respond well to flotation, with typical flotation recoveries of + 90 %. Cerussite, after activation with sodium sulphide, responds to flotation, but not as well as the primary sulphides. By way of comparison with the typical 1950's tailing, when the ores were all of the clean sulphide type, and 1988, the lead grade in the plant tailing increased from 0.6 % Pb to 1.2 % Pb.

Anglesite, a much less common "oxide" lead mineral, actually a lead sulphate will not float, but it has the potential to respond to gravity concentration.

A comparison of zinc mineralogy and process metallurgy is more challenging since for the last 8 years of operation a separate zinc concentrate was not produced because there was insufficient zinc in the ore to justify operation of the zinc flotation circuit. During the 1950's the zinc feed grade was about 8 %, but by the late 1970's had decreased to < 1 %, and in 1988 was 1.2 %. A review of the site data did not reveal any oxide zinc analyses, but undoubtedly the proportion of oxide zinc did increase at the same time as did the content of oxide lead. This is apparent from the current study (Sveinson) which included analysis of both oxide lead and zinc. Note that unlike some of the lead oxide minerals which, after sulphidization, respond to flotation, the zinc oxide minerals will not float.

When the above knowledge is combined with the fact that some 50 % of the ore was cyanided, and there appears to have been only a single point at which tailing was discharged into a meandering and frequently self-eroded alluvial fan, it becomes quite apparent that the settled tailing lacks homogeneity. This is indeed the case, although the 1988 report and the current investigation do indicate several perhaps useful features of the accumulated tailing, as follows:

(1) From a silver grade perspective there are two readily identified areas, as follows:

   (a) a low grade silver area, in the valley bottom which contains about 50 % of the total tailing. This material is elevated in slimes content by virtue of the ease of hydraulic transportation of mineral slimes. This lower area is underlain by organics, ie. muskeg, which is not trafficable. The 1970 drilling program (Watson) indicated 2.2 million tons grading 1.9 oz/t Ag.

   (b) a higher grade area which is represented by the alluvial fan on the gentle side slope of the valley. This material is both coarser and of higher grade than the slimes. The 1987/88 drilling program identified 1.7 million tonnes grading 4.45 oz/t Ag.

3.0 GRAVITY CONCENTRATION

3.1 UKHM - 1988

Gravity Concentration - Test 1

		oz/t - Assays - %			Distribution - %		
Product	Wt %	Ag	Pb	S	Ag	Pb	S
Mozely conc	0.4	103	10.2	37	11	3	10
Table conc	9.5	14	2.2	9	36	18	57
Tails	90.5	2.7	1.1	0.7	64	82	43
Feed	100.0	3.8	1.2	1.5	100	100	100

Gravity Concentration - Test 2

		oz/t - Assays - %		Distribution - %	
Product	Wt %	Ag	Pb	Ag	Pb
Mozely conc	0.5	213	7.0	25	3
Knelson conc	1.9	67	3.0	28	5
Tails	98.1	3.3	1.2	72	95
Feed	100.0	4.5	1.2	100	100

The Mozely concentrate has been separated from the balance of the statement, since it represents the incremental effect of cleaning the rougher concentrate.

A comparison of the two test results is somewhat challenging, since the calculated feed grade between the two tests is not in good agreement. The data, however, offers encouragement in the possible utilization of gravity concentration either alone or with subsequent flotation to produce a marketable concentrate.

3.3 Keno Hill - 1995

A 68 kg sample from the higher grade portion of the Keno Hill tailing, grading 2.4 oz/t Ag, was collected at site Z-8 (see Watson) and was subjected in entirety to centrifugal concentration using both Falcon and Knelson concentrators (see Sveinson).

The feed sample was first passed through a model B-6 (6") Falcon concentrator. The concentrate was advanced through 3 cleaning stages, the first of which was the same Falcon B-6, which was replaced by a smaller 3" Knelson concentrator for the subsequent cleaning stages.

As shown in Sveinson's Materials Balances, the results were very poor. The rougher concentrate contained 4 % of the feed weight at essentially the same grade as the feed. Since these results were not known until the products had been assayed, this low grade rougher concentrate was cleaned through 3 stages. The 3 rd cleaner concentrate contained approximately 20 % sulphides and graded only 7 oz/t Ag. Less than 0.5 % of the silver, lead, and zinc were recovered in this product.

This poor result does not emphatically condemn gravity concentration, but it does not encourage the retreatment of tailing using gravity concentration.

4.0 FLOTATION CONCENTRATION

4.1 Lakefield Research - 1994

Two flotation tests evaluated sequential flotation to produce four rougher concentrates, with intermediate slimes removal. Only the first rougher concentrate was cleaned. The test utilized very long rougher flotation times, totalling + 60 minutes.

Flotation Concentration - Test 3

		oz/t - Assays - %			Distribution - %		
Product	Wt %	Ag	Pb	Zn	Ag	Pb	Zn
Cleaner conc	4.8	25	2.0	2.6	32	8	18
Rougher conc 1	10.2	16	2.3	1.9	43	21	28
" " 2	3.8	16	5.1	3.9	16	17	21
" " 3	0.7	7	3.3	1.0	1	2	1
" " 4	1.3	8	3.6	1.2	3	4	2
Total rough conc	16.0				63	44	52
Slimes	23.0	3.2	1.9	0.6	19	37	21
Tails	61.0	1.1	0.4	0.3	18	20	27
Feed	100.0	3.8	1.2	0.7	100	100	100

In test 4, the feed was reground from 63 % - 200 mesh, to 80 % - 200 mesh. The results were only marginally better. Note that by virtue of a lower ratio of concentration than was achieved in the gravity concentration tests, the concentrate grade, at 25 oz/t Ag, was very low. The concentrate analysis suggests a high slimes content.

Although the reported silver recoveries were "acceptable" for tailing retreatment, the grades of the cleaned flotation concentrates were only equivalent to the average ore grade and were much lower than could be marketed.

The data suggests that gravity concentration alone (see Section 3.2) or with secondary flotation may be more favourable than flotation alone, since gravity concentration appears to be superior in its ability to discard gangue slimes than is flotation.

The sum total of this investigation perhaps suggests potential processing options, but it certainly has not come close to achieving technical or financial success.

5.0 CYANIDATION

5.1 Existing Cyanidation Plant

The Keno Hill plantsite includes a nominal 400 tpd tailing cyanidation plant which is attached to the flotation concentrator. This plant has been idle since about 1982.

Although the plant is reasonably intact, and probably could be placed back into operation, it is improbable that this could be justified for the cyanidation of tailing from the proposed mining of ores from the Bellekeno or Silver King mines.

The plant equipment is mainly of an obsolete design, but a few components could potentially be used if retreatment of the tailing is economically viable.

5.2 Agitation Cyanidation

There is considerable operating and laboratory data to indicate that about 50 % of the silver which is contained in the flotation tailing is amenable to direct cyanidation.

Undoubtedly, the long term volatility of the silver price constrained the economic ability to reprocess all of the flotation tailing subsequent to the construction of this plant in 1952.

Based upon this knowledge, there was no necessity to undertake further evaluations of agitation leaching, so the focus of the current study was directed to heap leaching, which is less costly to operate than is agitation cyanidation.

5.3 Heap Leaching

Heap leaching was investigated by Candorado Operating Company Ltd, which for several years has operated a tailing heap leaching operation at Hedley, B.C. for the recovery of gold.

The testing, which included both bottle roll and column leaching, determined that the tailing can be leached using cyanide heap leaching as effectively as can be achieved using the more costly agitation leaching procedure.

The testing determined that the material benefits from agglomeration with cement using fine waste or low grade ore as a nucleus. The use of nucleation is uncommon, but the higher grade portion of the Elsa tailing is essentially devoid of slimes, and does not agglomerate adequately using economical additions of cement. Testing to date suggests that agglomeration is effective at a weight ratio of 3:2 tailing to fine rock.

At this time Candorado has proposed to continue the testing on a 150 ton sample, at the Hedley site, to determine the final Design Criteria.

Analysis of the testing data by Candorado has indicated that the higher grade portion of the tailing can be economically processed at a rate of 1,500 mtd, with mining and leaching limited to an anticipated 6 month summer season. This is consistent with Watson's report which identified a higher grade portion comprising 1.0 million tons grading 5.35 oz/t Ag. At the proposed processing rate, it will require 4 years to process this inventory. If economics remain favourable, Candorado would continue to process some portion of the remaining accessible 0.7 million tons of higher grade material, at + 3 oz/t Ag.

6.0 CONCLUSIONS

6.1 Environmental

From an environmental perspective, there is no incentive to reprocess the tailing to minimize future environmental concerns. The annual requirements for liming of the tailing pond supernatant can be met with a modest, few ton, addition of lime to the supernatant during the summer, only.

The year-round discharge from the Bellekeno and Galkeno 800 portals contain significantly more zinc and are being continuously limed, for which there is an operating crew on site.

Although there is evidence from water analyses of the solid tailing surface moisture (see Sveinson) that zinc is being leached from the tailing, there is no indication that this is caused by acid generation, and there is no indication that relocating the tailing will diminish the rate of solubilization of the zinc.

6.2 Reprocessing of Accumulated Tailing

6.2.1 Silver Recovery

From a technical perspective, additional silver can be recovered from the approximate 50 % of the tailing which has not already been leached with sodium cyanide. From a practical perspective, however, the flotation and cyanidation tailing have been commingled and only that portion of the tailing which has deposited between the tailing discharge point and the valley bottom is potentially of sufficient grade and is sufficiently accessible to be worthy of economic consideration.

The use of a single discharge point for the tailing slurry, on the upper flank of the valley, has resulted in an elevated silver grade in the alluvial fan which has developed above the valley bottom. The coarser and higher specific gravity portion of the tailing settled in this area, and the much finer, lower specific gravity, and lower grade "slimes" flowed to the valley bottom where they overly muskeg which is not trafficable.

The coarser portion of the tailing, at a maximum depth of about 15 feet (4.9 meters) overlies the original soil which, by virtue of the shallow side slope has not been eroded, and which is trafficable.

This suggests that from a technical perspective the coarse tailing, representing some 1.7 million tonnes grading 4.45 oz/t Ag, can be excavated and transported to a heap leaching site where it can be percolation leached with cyanide.

At the current silver price of $ US 5.50 / oz, the economics of heap leaching is encouraging. However, to avoid deflecting the primary focus of UKHM's attention from proposed underground mining of ores from the Bellekeno and Silver King deposits, UKHM may defer any tailing retreatment "production" decision until the mining and processing operation has been established once again. This decision does not diminish the feasibility of tailing retreatment at the current silver price, but from UKHM's perspective, the economics of retreatment are very modest compared with that of ore processing.

At this time the testing has not been sufficiently advanced to entertain any Environmental Permitting.

6.2.2 Base Metal Recovery

Although some of the laboratory testing has offered encouragement for economical recovery of lead from the plant tailing, the results have not been sufficiently consistent to suggest that retreatment of the tailing for base metal recovery is economically viable.

To a varying extent, lead and zinc are present in the accumulated tailing as oxides which have both derived from open pit operations and to a much lesser extent from oxidation within the tailing pond.

When the mine is back in operation, it is UKHM's intention to evaluate gravity concentration of the ongoing plant tailing in an attempt to improve the plant recovery.

In all probability, the proposed study will indicate that little or no improvement can be made as long as only fresh sulphides are being processed. However, once oxide ores are introduced into the plant feed, as may occur after establishing sustainable mining and processing of Bellekeno and Silver King ores, the technical and financial opportunities for processing beyond flotation will increase considerably.

Perhaps this investigation will indicate further processing economies with respect to the ongoing future operation, but it is improbable that it will indicate any economic potential for the retreatment of the existing plant tailing for base metal recovery using contemporary process technology.

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