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Recovery of Magnetite from Phoenix Tailings

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Map # M082E008 and M082E018
Latitude 5440000
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Owner and Operator: Kettle River Resources Ltd.

Prepared by:
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GEOLOGICAL SURVEY BR

W. Scott Dunbar

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1 Summary

This report presents the results of a study to assess the recovery and potential usage of magnetite from Phoenix tailings for dense media separation. Other minerals such as garnet, gold, silver and chalcopyrite were also identified as minerals of interest. The study primarily considers the potential for magnetite production, but also investigated the feasibility of concentrating other minerals.

Existing reports were reviewed to evaluate the technical feasibility of re-processing Phoenix mine tailings. The reports indicated that re-grinding is necessary to recover magnetite, garnet, copper and precious metals. The reports also indicated that processing Tremblay and Twin Creek tailings may be economically viable, however no substantial data on economic feasibility was included.

A sampling campaign was carried out on the Tremblay tailing ponds to obtain 25-kg samples from different locations. The samples were studied using process mineralogy and beneficiation techniques. The mineralogical studies involved optical and electron microscopy as well as X-ray diffraction which identified chlorite, illite, calcite, garnet, quartz, magnetite and trace amounts of chalcopyrite, pyrite, apatite, goethite, ilmenite, zircon and gold in the Tremblay tailing. The degree of mineral liberation without grinding ranges from 61.4 to 64.5% confirming that grinding is needed to extract magnetite. However, excessive grinding will not increase the liberation as a portion of magnetite occurs as minuscule inclusions (<5μm) in gangue minerals. Garnet consistently contains fine magnetite inclusions and therefore it is very unlikely to generate a saleable product.

The metallurgical testing showed that re-grinding and cleaning stages are necessary to produce a magnetite product with marketable characteristics. Three stages of cleaner magnetic separation were conducted to obtain the marketable product with 96.1% magnetite, specific gravity 4.66 and 67.2% finer than 44μm. Finer grinding would produce a concentrate that meets specifications for magnetite use in coal processing. However, the product yield is likely to be very low at about 1.9%.

Copper sulfide flotation did not render a high grade product. The final product graded 1.2% Cu with very low recovery (3.1%). The gold grade reached 20 g/t in the final flotation concentrate but the recovery was only 3%. These recoveries are too low to be of practical significance.

2 Introduction

A study was under taken at the UBC Department of Mining Engineering to assess the Phoenix tailings for the recovery of magnetite, garnet and other minerals from Tremblay tailings pond. The feasibility of concentrating the identified minerals was also studied.

The sampling, mineralogical studies and metallurgical tests were focused on obtaining products that meet market specifications. Preliminary equipment selection was also included in this study as part of a student exercise.

3 Background

3.1 Phoenix Mine

The Phoenix mine is located east of Greenwood, BC on NTS map sheet 82E/2E. Access to the property is from Greenwood east on the Phoenix road, or west and north from Grand Forks on Highway 3, then west on the Phoenix road (location map is attached).

Terrain is generally moderate, with elevations ranging from 900 to 1600 m. The slopes are generally forested, which on north facing slopes can be quite heavy. Some open grasslands occur on south and west facing slopes. The climate is moderate and semi-arid, with hot, dry summers and winter snow accumulations in the order of 1 – 2 metres. Abundant water for exploration is available from old mine workings, ponds or creeks (L. Caron 1996).

This mine operated from 1900 (1896) to 1978. The mine produced around 30 million tons of copper-gold skarn ore from open pit and underground mining. Silver was also produced by the mine. From 1956 to 1978 approximately 14,778,000 tons of cooper-gold-silver ore was treated. The metal recovery was not very high (precious metals between 55 and 65%), especially in the early years of operation.

Chalcopyrite was the only copper bearing mineral in the ore. Other metallic minerals include pyrite, magnetite, specular hematite and possibly an unidentified silver bearing mineral. Gangue minerals included carbonates, epidote, chlorite, garnet and quartz. Gold and silver were primarily associated with chalcopyrite and pyrite.

The amount of tailings generated by the Phoenix mine was approximately 14,500,000 tonne. Tailings from processing plant, which employed flotation to concentrate the copper sulphide, were deposited into three ponds (Tremblay, Twin, Open Pit). The tonnage and grades of copper, gold and silver in each of these tailing dams is shown in Table 1. Tremblay pond is favorably situated and contains an estimated 4.6 million tons of material with grades of 0.151% Cu, 0.34g/t Au and 3.4 g/t Ag. The Tremblay tailings are composed of waste from the early stages of operation and therefore contain the highest metal grades. Tremblay tailings are situated in the northeast portion of the property, primarily on DL 2116. The map of tailings on claims is attached.

Table 1: Tonnages and grades for the three ponds (Tailings retreatment possibilities at Phoenix B.C., Report, 1990, G.B. Hardwicke, P. Eng.)

Pond	**Grade**			
	Metric Tonnes	**Copper (%)**	**Gold (g/tonne)**	**Silver (g/tonne)**
Tremblay	4,194,490	0.151	0.377	3.737
Twin Creek	7,332,087	0.084	0.274	3.188
Open Pit	1,596,322	0.087	0.171	2.228
Total	**13,122,900.201**			

3.2 Review of Existing Reports

Six previous studies were conducted to examine the technical feasibility of recovering copper and gold from tailings.

Mineralogical studies of the tailings samples revealed that the remaining gold and silver are not liberated and could not be recovered by flotation during the previous operation. The lack of regrinding and the presence of oxidized copper minerals resulted in low copper and precious metal recoveries.

Studies on the Tremblay, Twin and Open Pit ponds showed that the metal values are not evenly distributed throughout the ponds. The Tremblay and Twin Creek ponds were built by hydrocycloning a portion of the tailing flow. The coarser hydrocyclone underflow was deposited on the dam and the overflow, consisting of fine material was allowed to flow to the center of the pond. The coarse and heavy fraction of the tailings settled at the upstream end of the ponds. Under these circumstances the central portion of the pond should contain less metal than the dam or upstream portions.

The Twin Creek pond is an environmentally sensitive area and disposing of retreated tailings below the present site would pose many problems. The metal values in this pond are much lower than those in the Tremblay pond. Report (Tailings retreatment possibilities at Phoenix B.C., Report, 1990, G.B. Hardwicke, P. Eng.) suggests that the treatment of both the Tremblay and Twin Creek tailings may be economically viable because the capital costs could be amortized over the larger tonnage.

Although there is a suitable site to establish a reprocessing plant at the Open Pit, there is no satisfactory area for tailing disposal. Furthermore, tailings are kept flooded under millions of gallons of water and metal concentrations are low. Moreover, there is a high fine particle content and the tailings would pose even greater problem if the flotation process was utilized.

The sampling, drilling and metallurgical tests conducted previously are outlined as follows:

- Noranda (1984) attempted drilling using a small vibratory drill but this was unsuccessful because of impenetrable clay layers.
- In May 1991, Bacon and Donaldson was commissioned to test tailings from the Tremblay pond. Based on tabling and panning tests, the report concludes that is not possible to produce a high grade gold concentrate with acceptable recovery using only gravity separation. From a head sample grading 0.88 g/tonne Au, a concentrate with 64 g/tonne Au was obtained by panning with recovery of 1.56%. Very little free gold was observed and this led to the

conclusion that gold occurs locked in the coarse sulphide mineral grains such as pyrite that may be concentrated by flotation. The cyanidation tests corroborate that grinding is necessary to expose unliberated gold particles. Low Intensity Magnetic Separation testing has concluded that the tailing sample has only 2.2% magnetite. High Intensity Magnetic Separation recovered about 72% of the sample mass that comprises of quartz-locked magnetite and garnet.

- In July 1991, another Bacon and Donaldson study produced a flotation concentrate grading 11.5 g/tonne Au, 18 g/tonne Ag and 0.54% Cu. Pyrite was the predominant mineral in the concentrate. Magnetic separation concluded that the sample had 4.8% magnetite.
- In 1994-1995, Echo Bay Exploration contracted Hazen Research, Golden Colorado to conduct gravity concentration, flotation and cyanidation tests on two tailing samples from Tremblay and one sample from Twin Creek. Head sample grades ranged from 0.41 to 0.97 g/tonne Au. Shaking tables recovered 50% of the gold and produced a concentrate with 3.55 g/tonne Au. This concentrate was cleaned using a Gemini table to reach 3203 g/tonne Au with a recovery of 15%. Flotation of the shaking table tailing recovered from 17 to 83% of the gold, depending on the sample. Cyanidation of the gravity concentrates, after removal of the free gold, recovered 77 to 90% of gold.
- In 1996, Kettle River Resources carried out cyanidation and flotation tests on the surface samples of Tremblay tailings and recovered 67% and 57% of the gold respectively. Magnetic separation testing was also conducted on these samples and showed that the tailings contained 4.8% magnetite and 27.2% +100 mesh garnet.
- Results obtained on samples of Tremblay and Twin Creek tailing, using gravity concentration (Gemini table/shaking tables) produced low gold recovery (from 44 to 55%).
- Caron (1996) reported results from two test holes drilled in the Tremblay tailing ponds. The sample size was 4.25 inch diameter by 77.5 ft deep in the berm, and 61.5 ft deep in the pond. It was found that the base of the tailings is very anaerobic and composed of an organic layer, several feet in thickness above gravel and dirt. Sonic drilling retrieved samples grading 0.14% Cu and 0.64 g/tonne Au, which were sent to International Metallurgical and Environmental Inc. (IME), Kelowna for testing to recover gold and copper by flotation. The copper recovery was between 35 and 45% when samples were ground 65% minus 200mesh. The copper concentrate was very poor with grades of 2 to 3% Cu. The gold concentrate grade reached 25-35 g/t with occasional concentrates of 250 g/t. Recovery ranged from 65 to 80% (ground 65% -200 mesh). Based on microscopic observations, it was possible to determine that chacopyrite comprises only 3% of the concentrate (pyrite is the dominant mineralogical species with 96%) that is basically liberated.

4 Sampling

During September 14 to 15, 2001, a group of researchers from UBC, Dept. of Mining and Mineral Process Engineering sampled the Tremblay tailing pond. Seven samples, TR-01 to 07, were taken from various locations of the pond with the objective of collecting samples for evaluation of the tailings. Figure 1 shows a sketch of the tailings pond and approximate sample locations.

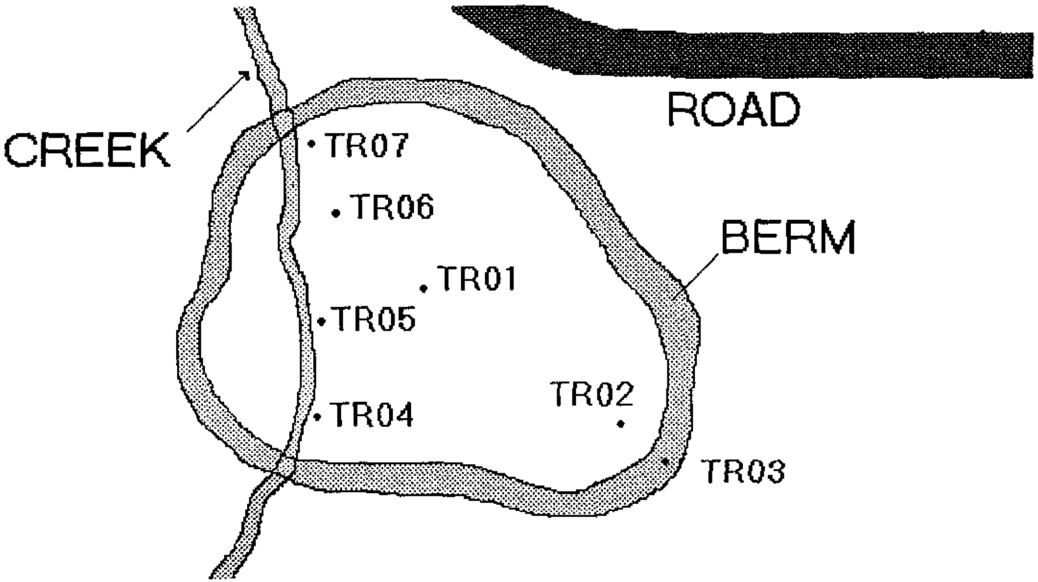


Figure 1: Sampling campaign, taken from the tailings dam at Phoenix Mine

Samples were packed in plastic bags and transported to the UBC - Center for Mineral Processing lab in Vancouver where they were dried and weighed. The weight of samples TR-01, 02, 03, 05, 06 and 07 were 24.6, 26.8, 30.9, 31.6, 25.9 and 29.4 kg, respectively. Each sample was split in two parts: one to be used for mineralogy studies and the other for metallurgical tests, the remaining sample was stored.

In order to simplify the laboratory work and indicate mineable zones, samples from areas of the pond that displayed consistent characteristics were combined and homogenized into Composite samples. Samples TR-01, 02, from the center of the tailings pond were combined to create Composite 1; sample TR-03, was taken from the berm and considered separately from the pond, was called Composite 2; samples TR-05, 06, and 07 from the south side of the pond adjacent a run-off stream were combined to make Composite 3. Unfortunately, sample TR-04 was lost during transportation. Each Composite was split in 2 kg sub-samples and stored in bags.

5 Process Mineralogy

5.1 Introduction

The 3 Composite samples were subjected to a number of procedures to determine the potential for extracting values from the tailings. The study was primarily concerned with the potential for magnetite production, but also examined the feasibility of recovering other minerals.

The process mineralogical analyses included chemical analysis, magnetic separation, heavy liquid separation, evaluation of the degree of liberation, mineral characterization by x-ray diffraction and scanning electron microscopy. The purpose of each test is described in the corresponding section. Figure 2 shows an overview of the sample preparation and mineralogical procedures.

5.2 Sieve Analysis

A sieve analysis was performed to obtain the particle-size distribution information and to supply samples from each size fraction for mineralogical analyses. The retained percentage in different sieves is shown in Figure 3.

Composite 1 sample comprises material that is finer than Composites 2 and 3. The sieve analysis shows that the majority of the material for Composites 2 (66.9%) and 3 (56.8%) is in the +100 mesh size fraction (150 μm) where as Composite 1 has about 21% of material coarser than 100 mesh and 46% of material passing through the 400 mesh screen (37 μm).

5.3 Magnetic and Heavy Liquid Separations

Magnetic and heavy liquid separation tests were performed on the samples with the intent of determining the amounts of magnetite and other valuable minerals and to evaluate the liberation of these minerals. By analyzing separation products, a more accurate qualitative mineralogy was obtained. The results of these tests are reported in the following sections.

5.3.1 Magnetic Separation

5.3.1.1 Magnetic Separation Using Wet Low Intensity Magnetic Separation on Individual Samples:

Low intensity magnetic separation was performed on each of the six individual samples from the tailings pond, in order to quantify the amount of magnetite. The samples were pulverized in a BICO disk pulverizer to below 200# (75μm) to liberate the magnetite. Each sample was pulped to 20% solids and then passed through a Davis tube wet low intensity magnetic separator (WLIMS). The results of the Davis tube tests are shown in the Figure 4.

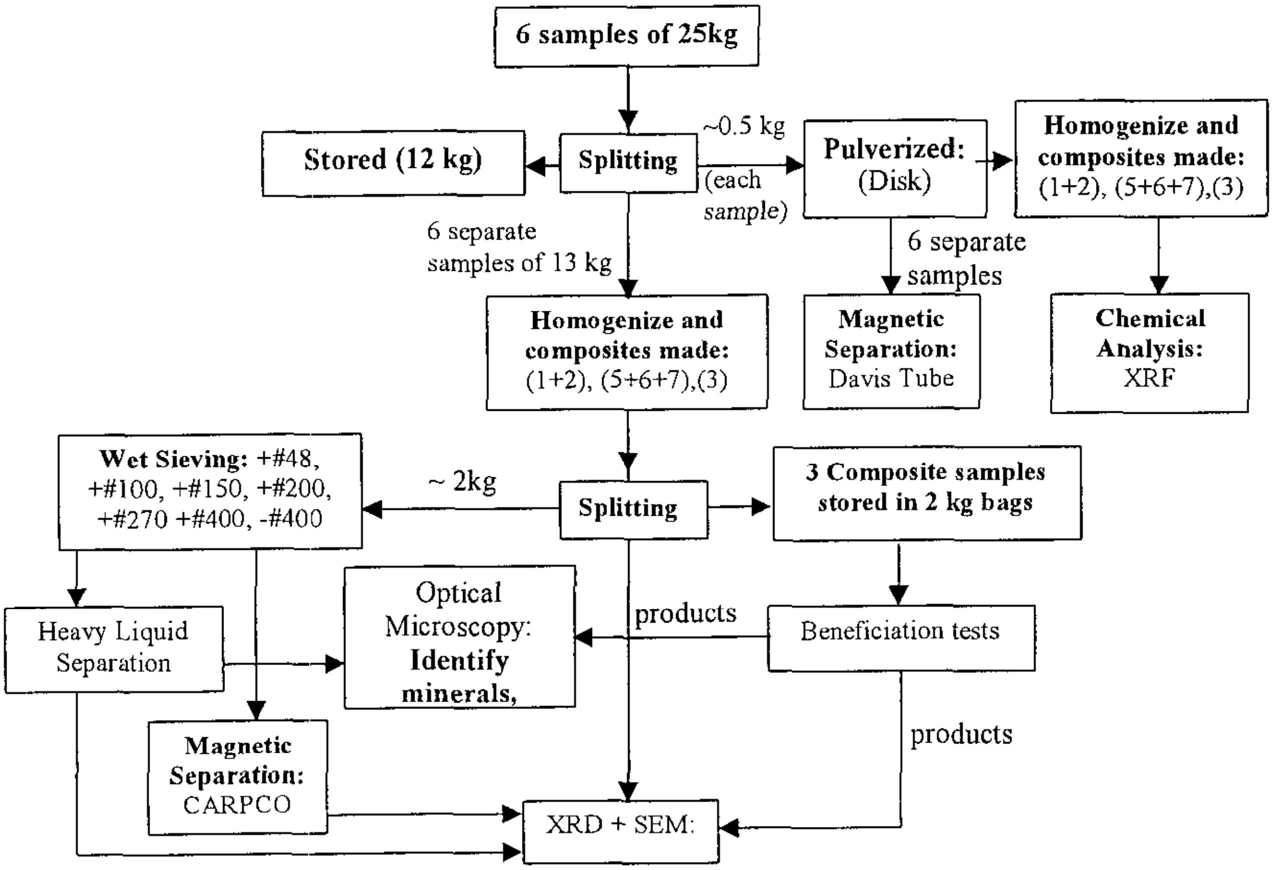


Figure 2: Phoenix Tailings Project: Overall Process Mineralogy Flowsheet

These results provide the most accurate indication of the magnetite grade in different areas of the pond. Although the samples were pulverized to 80% -200# (75µm), much of the gangue material still contained small magnetite inclusions (this will be discussed in the optical microscopy section). The gangue material with magnetite inclusions often contains enough magnetite to be magnetic (10-30%), making it susceptible to magnetic separation, and thereby lowering the magnetic concentrate grade (and increasing the concentrate weight). This explains why the Davis Tube retained a greater weight percentage than the weight of magnetite calculated in Table 5. The maximum % weight retained (5.5%) occurred in Sample #2, and the minimum % weight retained (0.9%) occurred in Sample 1. Both of these samples were taken from the same creek bed, and homogenized to form Composite 1. This result indicates that the assumption of uniform distribution of magnetite in the creek bed may be false.

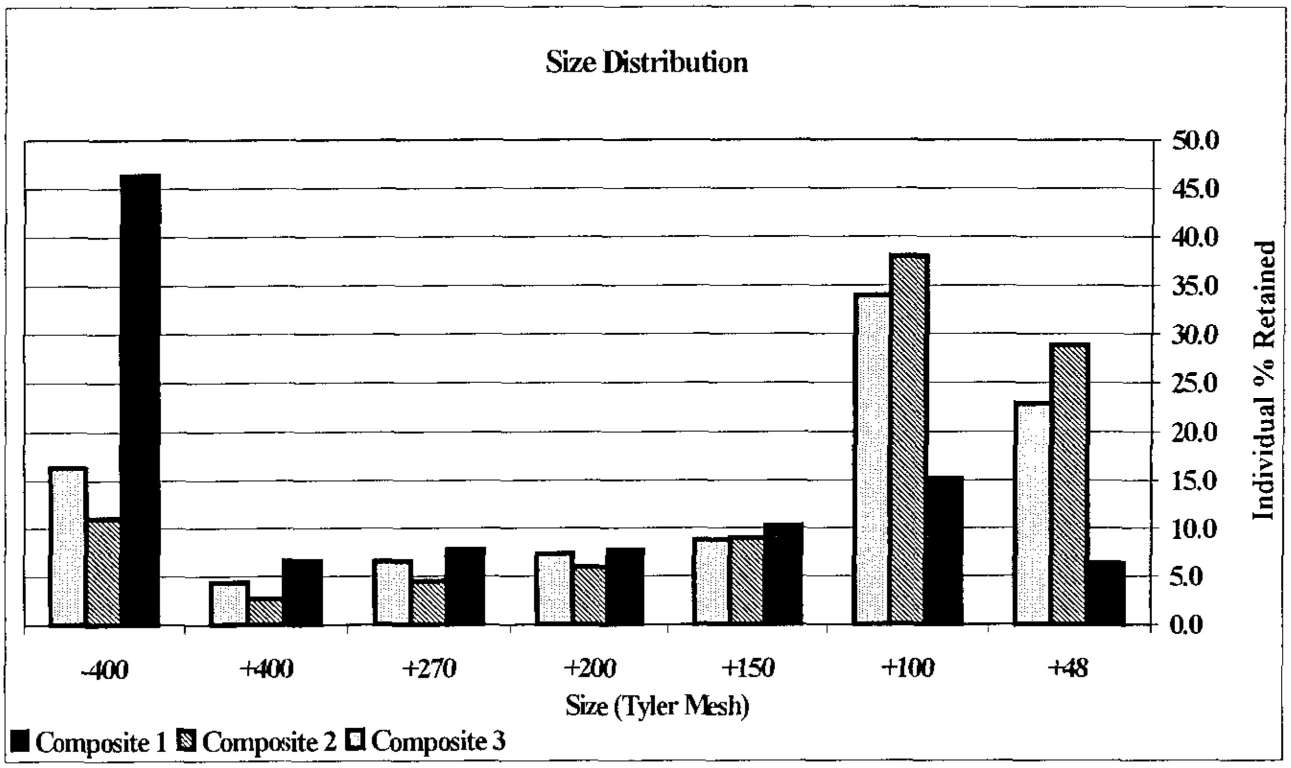


Figure 3: Percent weight retained in each size fraction for Composites 1, 2 and 3

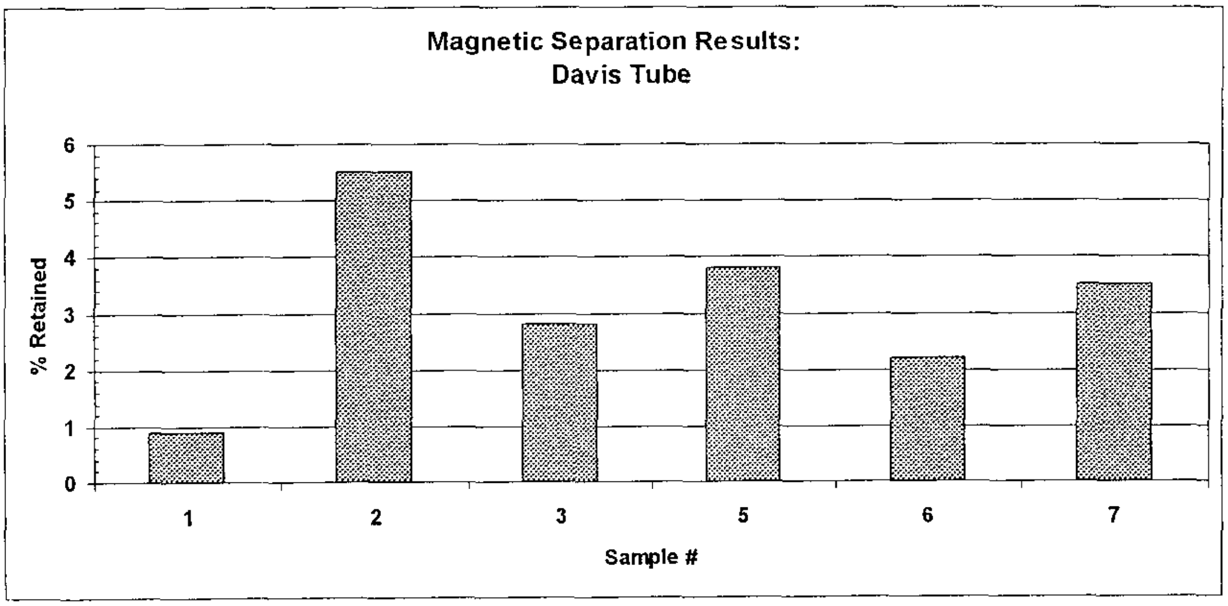


Figure 4: Wet Low Intensity Magnetic Separation (Davis Tube) results from samples TR-01 to 07.

5.3.1.2 Magnetic Separation Using Wet High Intensity Magnetic Separation on Composite Samples

In an attempt to simplify the magnetic separation process, a Carpco wet high intensity magnetic separator (WHIMS) was used on the Composite samples. For these tests, magnetic separation was performed on each size fraction of each of the 3 composite samples. In this process, any particle containing magnetic (magnetite) or paramagnetic (hematite and goethite) mineral is recovered. Figure 5 summarizes the results of magnetic separation using the WHIMS. The graph shows a clear trend of decreasing weight percent retained with decreasing particle size. This is due to an increasing degree of liberation of the magnetic minerals in the smaller size fractions.

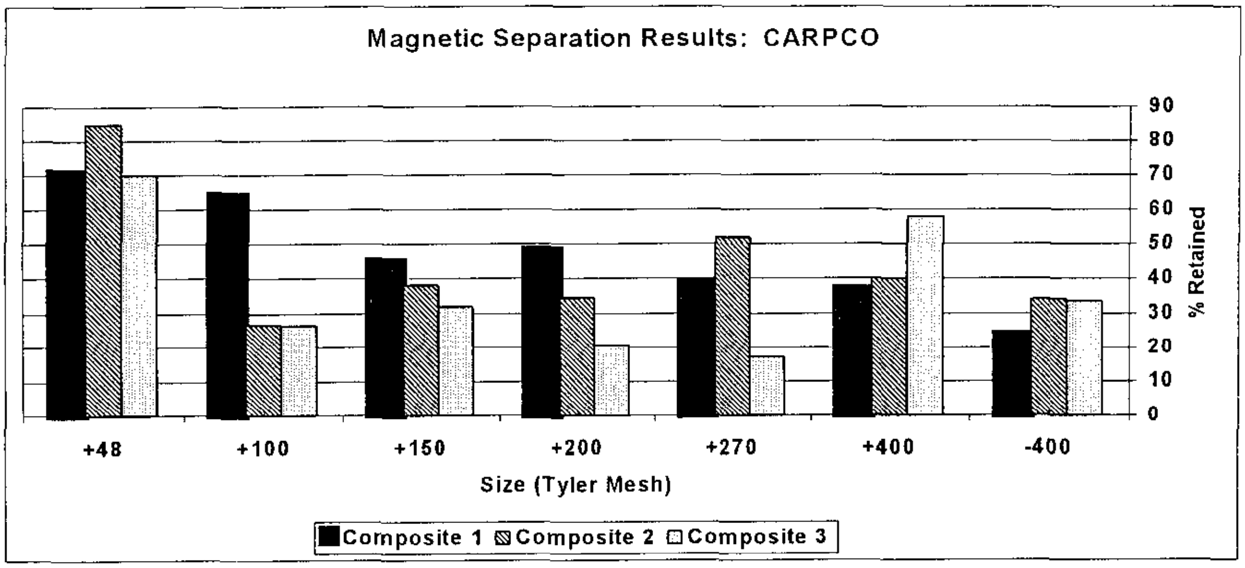


Figure 5: Wet High Intensity Magnetic Separation results

Table 2 shows the weight and magnetic distributions for the three composite samples.

Table 2: Wet High Intensity Magnetic Separation Results on Composite Samples

Size Fraction (Tyler Mesh)	%Wt. (grain size)			% Magnetics			$\%D_{magnetics}$		
	1	2	3	1	2	3	1	2	3
+48	6.3	28.9	22.8	70.8	84.9	70.0	11.5	51.8	41.8
-48 +100	15.1	38.0	34.0	64.0	26.8	26.5	24.8	21.5	23.4
-100 +150	10.2	9.0	8.8	45.1	38.1	32.1	11.8	7.2	7.3
-150 +200	7.7	5.9	7.3	48.1	34.4	20.5	9.5	4.3	3.9
-200 +270	7.8	4.5	6.5	38.4	51.9	17.0	7.7	4.9	2.9
-270 +400	6.6	2.7	4.4	36.8	39.9	57.8	6.2	2.3	6.6
-400	46.3	11.0	16.3	24.0	34.0	33.4	28.5	7.9	14.1
Feed	**100**	**100**	**100**	**39.0**	**47.3**	**38.4**	**100**	**100**	**100**

The distribution data indicates that most of the magnetite is in the coarser fractions (+48#, +100#). This may be a result of lack of liberation of the magnetic minerals (magnetite, hematite, goethite) in the coarse fraction. The high distribution of magnetics in the fines (-400#) indicates that the clay fraction was not effectively washed out in the operation of the high intensity magnetic separator.

The last row of Table 2 indicates that Composite 2 (berm sample), has the highest percentage (47.3%) of magnetic material. This sample has about 3% of magnetite, as indicated in Fig. 4, (sample 3) and comprises the coarsest material (76% above 150 mesh). It is expected that most of the material retained by magnetic separator consists of unliberated magnetic and paramagnetic minerals.

5.3.2 Heavy Liquid Separation

Heavy liquid separation was performed in order to concentrate minerals and study their liberation behavior. Methylene iodide (CH_2I_2) was chosen as the heavy liquid, since its density (SG=3.0) is intermediate between the magnetite/garnet and the gangue. Table 3 shows the specific gravities of the primary minerals in the samples. The sinks would contain magnetite and garnet, since these minerals have a specific gravity well above that of methylene iodide.

Table 3: Specific Gravities of Primary Minerals Involved in Heavy Liquid Separation

Mineral:	**Magnetite**	**Garnet**	**Calcite**	**Chlorite**	**Quartz**
Formula:	Fe_3O_4	variable	$CaCO_3$	$(Mg, Al, Fe)_6[(Si,Al)_4 O_{10}]OH_8$	SiO_2
Specific Gravity:	~5.1	3.6-4.3	2.7-3.4	~2.4	~2.7

The Figure 6 shows the results of heavy liquid separation of each of the 3 Composite samples.

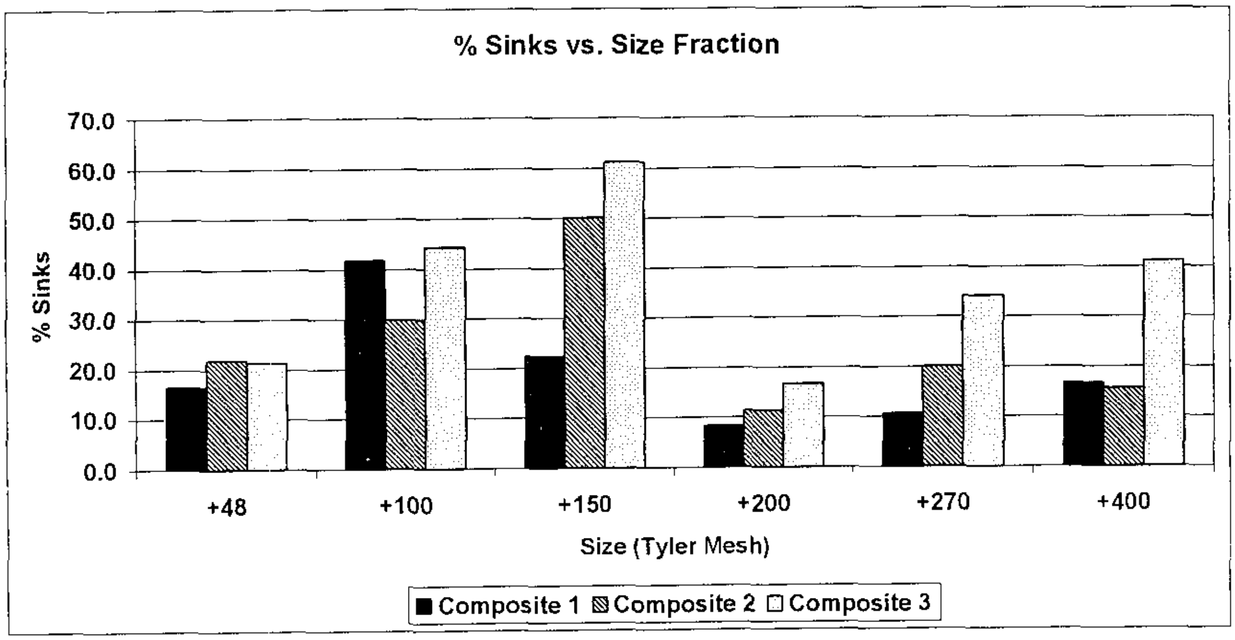


Figure 6: Heavy liquid separation results

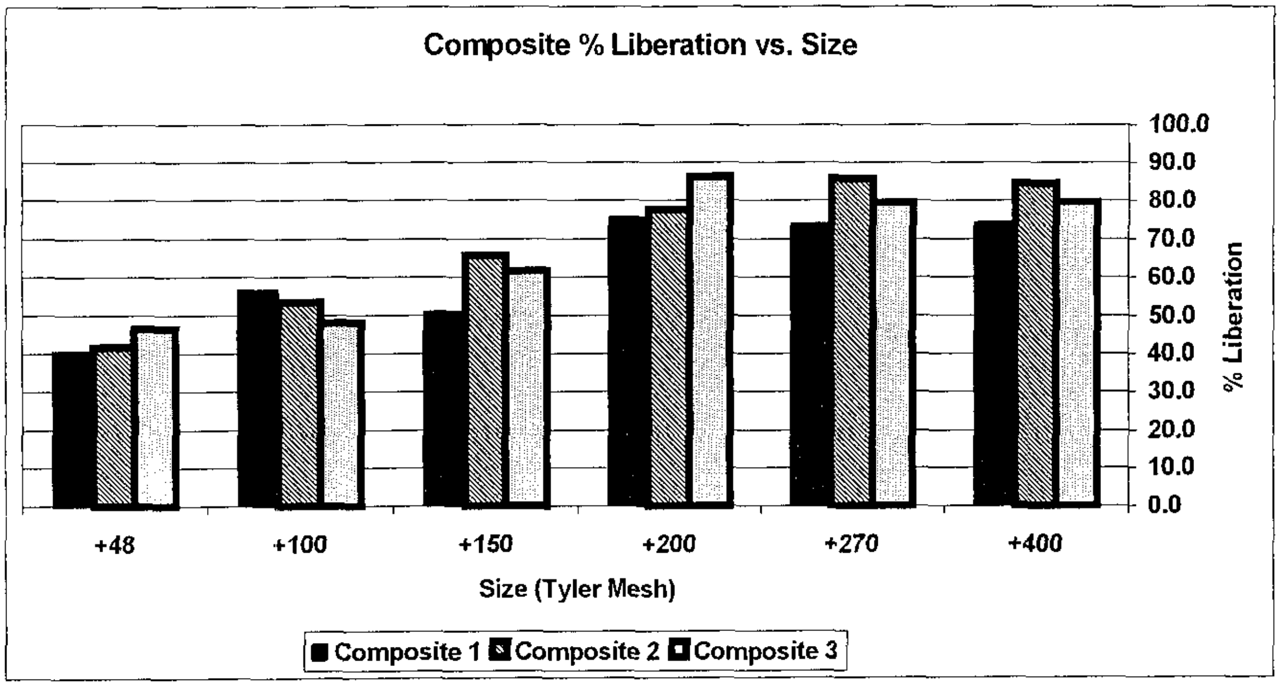
Figure 6 indicates that Composite 3 has more heavy minerals in all size fractions than Composites 1 and 2. It is also noticed that the % sinks decreases below 200#. This result indicates that the liberation of the magnetite and other heavy minerals is at this grain size. Heavy liquid separation was not performed on the –400# fraction due to low effectiveness of this procedure for very fine particles.

The distribution of sinks indicates that heavy minerals appear mostly in the coarser fractions and a gravity separation process would not produce high grade concentrates as most minerals are not fully liberated in grain sizes above 200 mesh.

5.4 Degree of Liberation

The degree of liberation of magnetite was determined by optical microscopy on the sink products. The liberation was determined in each individual size fraction using a simplified Gaudin Method. The method divides the amount of free particles in each sized fraction by the amount of free and locked particles in the same fraction. A transmitted condensed light, microscope was used and it assumed that the opaque minerals were magnetite. It was possible to distinguish magnetite from sulfides using reflected light during the microscopic analysis. Figure 7 summarizes the degree of liberation obtained for each sized fraction.

Figure 7: Summary of % liberation calculated for each Composite in each size fraction



The graphic shows that 80% liberation occurred at approximately 200#, or 74μm. For all Composite samples, the degree of liberation did not seem to increase beyond 80% as the particle size decreased down to 400#. Incomplete liberation is likely due to a significant portion of very small magnetite inclusions locked in the gangue minerals (1 to 3% of the particle volume). This characteristic of the material is better visualized in the Scanning Electron Microscope.

An important result from the liberation study was that at least two different types of magnetite particles are present. The first consists of mostly liberated particles, i.e. particles with degree of liberation above 90%. The second type comprises particles with minuscule inclusions of magnetite in gangue minerals accounting for a great number of grains with less than 40% of magnetite. Figures 8 and 9 exemplify this characteristic that is more evident in the -100+150 mesh fraction. The classes of liberation from 1-10 in the graphics represent the magnetite particles with 10% to 100% liberation, respectively. It is observed that the frequency of free particles in the fraction -35+48 mesh is 26%, which is not a very high difference from the 35% of free particles in fraction -100+150 mesh. Fraction based on visual estimates the fine magnetite particles would be liberated by grinding to below 5 μm, which is not economic. Therefore, it is recommended to recover only the first type of magnetite grains. In this case, it is expected that the recovery would not be high but, rejecting the locked particles, the final grade of the magnetic concentrate can meet product specification.

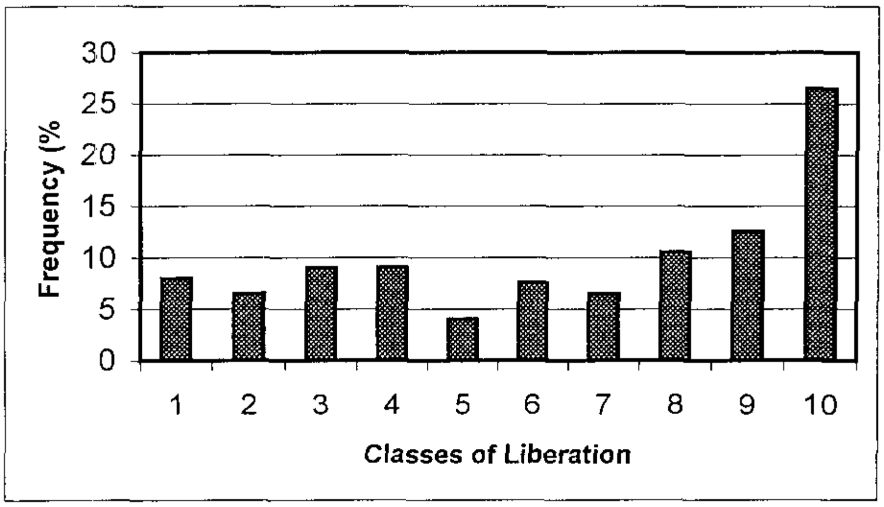


Figure 8: Classes of liberation of the magnetite particles in fraction -35 +48 mesh of the Composite 1

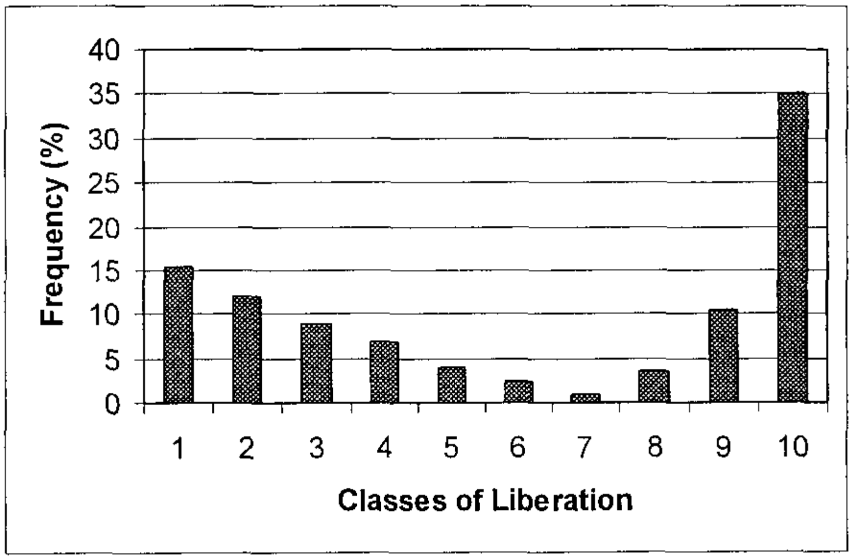


Figure 9: Classes of liberation of the magnetite particles in fraction -100 +150 mesh of the Composite 1

The cumulative liberation yield (CLY) was also determined from the degree of liberation. The CLY is the proportion of a mineral that are present as a free grains plus the proportion of middling particles that can be recovered. This is based on the assumption that grains with an individual liberation above the overall product liberation can be recovered. The CLY is a mathematical approximation of the actual recovery that can be expected from a concentration process. Tables 4, 5 and 6 show the CLY and product liberation calculations.

Table 4: Cumulative liberation yield for Composite 1

Fraction	**Magnetite Mass (g)**	**Magnetite Dist'n (%)**	**% wt Free Magnetite**	**% wt locked Mag > Lp**	**% Free Mag in Concentrate**	**% Middling Particles Liberation > Lp**
+48#	37.5	14.3	40.7	66.8	5.8	9.5
+100#	58	22.1	56.6	79.0	12.5	17.5
+150#	58.2	22.2	61.3	51.8	13.6	11.5
+200#	38.4	14.6	75.6	87.5	11.1	12.8
+270#	46.4	17.7	74.1	79.2	13.1	14.0
+400#	23.9	9.1	74.3	79.2	6.8	7.2
Total	262.4	100.0			**62.9**	**72.5**

Table 5: Cumulative liberation yield for Composite 2

Fraction	**Magnetite Mass (g)**	**Magnetite Dist'n (%)**	**% wt Free Magnetite**	**% wt locked Mag > Lp**	**% Free Mag in Concentrate**	**Middling Particles Liberation > Lp**
+48#	15.7	6.6	41.7	65.7	2.7	4.3
+100#	95.8	40.3	53.4	72.7	21.5	29.3
+150#	57.4	24.1	65.6	79.0	15.8	19.1
+200#	9.6	4.0	77.5	91.5	3.1	3.7
+270#	47.5	20.0	85.6	90.0	17.1	18.0
+400#	11.9	5.0	84.5	94.8	4.2	4.7
Total	237.9	100.0			**64.5**	**79.1**

Table 6: Cumulative liberation yield for Composite 3

Fraction	**Magnetite Mass (g)**	**Magnetite Dist'n (%)**	**% wt Free Magnetite**	**% wt locked Mag > Lp**	**% Free Mag in Concentrate**	**Middling Particles Liberation > Lp**
+48#	23.2	9.8	46.5	87.1	4.6	8.5
+100#	75	31.7	47.9	78.8	15.2	25.0
+150#	70.4	29.8	61.4	72.5	18.3	21.6
+200#	16.5	7.0	86.2	94.5	6.0	6.6
+270#	14.5	6.1	79.5	82.5	4.9	5.1
+400#	36.9	15.6	79.6	80.2	12.4	12.5
total	236.5	100.0			**61.4**	**79.3**

The CLY for Composites 2 and 3 was 79%, and it was 72.5% for Composite 1. This indicates that processing material, as it is, from the area represented by Composite 1 would result in lower recoveries than by Composites 2 and 3. The degrees of liberation of the products were similar, ranging between 61% and 65% for all three composites. The CLY increases if these samples are submitted to comminution, i.e. grinding.

5.5 Mineralogy

The mineralogical analysis of the tailing Composites of Phoenix Mine was performed by combining results from Scanning Electron Microscopy (SEM) coupled with Energy Dispersive X-ray Analysis System (EDS) and X-ray Diffraction (XRD).

The mineralogical examination was conducted on several representative Composite samples and their respective separation products (heavy liquid and magnetic separation). Products from metallurgical tests were also analyzed which are described in section 6 of this report.

In order to confirm the mineralogical compositions of the composite samples, the following samples were submitted to SEM and XRD analyses:

- Heavy fraction from the +150# fraction of Composite 1
- Heavy fraction from the +270# fraction of Composite 3
- Magnetic concentrate (WHIMS) of the +270# fraction of Composite 3
- Magnetic concentrate (WHIMS) of the -400# fraction of Composite 3

The XRD analyses indicated that all Composite head samples contain quartz, chlorite, calcite, illite, garnet, magnetite and traces of hematite.

The XRD pattern of the heavy fraction from the +150# fraction of Composite 1 (Appendix I-A) shows the presence of magnetite, garnet, calcite, chlorite and quartz. The x-ray mapping obtained from SEM (Appendix II-A) shows that there are a few free particles of magnetite indicated by the Fe distribution. Fe associated with Si, Al and Ca is indicative of the presence of ferruginous chlorite. Garnets also are formed by Fe or Al associated with Ca and Si (andradite,$Ca_3Fe_2Si_3O_{12}$ or grossular $Ca_3Al_2Si_3O_{12}$). The garnet minerals should be concentrated in the heavy fraction as their densities range from 3.6 to 4.3. Quartz (SiO_2) and calcite ($CaCO_3$) also appear in large concentrations in the sample as can be seen in the x-ray mapping. Most of magnetite mineral was locked inside gangue minerals. The x-ray mapping suggests that magnetite inclusions are smaller than 10 µm.

The XRD pattern (Appendix I-B) from the heavy fraction of +270 mesh size fraction (53 µm) of Composite 3 shows the presence of magnetite, hematite, garnet, calcite, chlorite and quartz. The x-ray mapping, (Appendix II-B1 & B2) shows liberated magnetite particles and the presence of a few grains of iron sulfide indicated by the sulfur and iron association. Some of the sulfide particles show superficial oxidation as sulfur is depleted on the surface. However, the magnetite is still not fully liberated occuring as fine inclusion in some particles. From the XRD pattern, it is possible to infer that the amount of quartz and calcite are approximately the same in the sink product from the +150# fraction of Composite 1.

The XRD analysis from the magnetic concentrate (Appendix I-C) obtained by WHIMS of the +270# fraction (53 µm) of the Composite 3 shows the presence of chlorite, calcite, magnetite, hematite, quartz and ilmenite. The x-ray mapping (Appendix II-C) shows that this concentrate is much richer in magnetite (Fe distribution) than the previous samples. Most magnetite is liberated despite some small (~10 µm) inclusions of this mineral in the gangue minerals. The x-ray mapping also shows that there is a large majority of the iron in chlorites, characterized by the association of Fe with Si.

The XRD pattern (Appendix I-D) of the magnetic concentrate obtained by WHIMS of the -400# fraction of Composite 3 indicates the presence of chlorite, calcite, quartz, magnetite, hematite and garnet. Despite, the high degree of liberation, over 80%, in the –400# fraction size, there are still small inclusions of magnetite in other minerals. The x-ray mapping (Appendix II-D1 & D2) also shows the presence of free particles of calcite and quartz with virtually no inclusion of magnetite or other iron-bearing minerals. The results indicate that the washing process during the magnetic separation was not very effective at removing very fine particles aggregated to the magnetic minerals. Further evidence of this fact is obtained from the XRD patterns. Comparing the diffratograms of magnetic concentrate of -400# and +270# fractions, it is noticed that peaks of calcite and chlorite were higher in the -400# fraction than in the +270# fraction.

5.6 Chemical Analysis and Quantitative Mineralogy

A mass balance was used to quantify the mineralogy of the samples. Chemical analysis by X-Ray Fluorescence (XRF) was performed at TeckCominco Research & Exploration Laboratory on each of the 3 Composite samples. XRF provides data on the relative amounts of the major elements in the sample. The results of the XRF study are shown in Table 7.

Table 7: Chemical analysis by XRF of composite samples

Comp.	SiO_2 (%)	Al_2O_3 (%)	MgO (%)	Na_2O (%)	MnO (%)	Fe_2O_3 (%)	TiO_2 (%)	P_2O_5 (%)	CaO (%)	K_2O (%)	LOI* (%)
1	39.24	6.71	3.30	0.18	0.34	11.03	0.34	0.27	19.71	0.67	15.81
2	42.97	3.50	3.34	0.07	0.31	11.56	0.20	0.23	18.70	0.17	15.85
3	42.79	6.01	2.80	0.18	0.34	13.73	0.38	0.23	16.45	0.46	12.84

* Loss on ignition: material vaporized when the sample is fused prior to analysis (usually H_2O)

The analytical results for Al_2O_3 obtained by XRF of Composite 2 seemed to be low when compared to Composites 1 and 3. This result was confirmed using atomic absorption indicating 3.49% Al_2O_3, i.e. identical to the XRF analysis.

The composition of the chlorite was determined by micro-analysis using SEM-EDS. The micro-analysis and the XRD pattern for chlorite match with the iron-rich ripiolite composition with generic composition $Mg_{6-x-y}\ Fe_y\ Al_x\ (Si_{4-x}\ Al_x)O_{10}\ (OH)_8$. The EDS determined the following average composition: Al = 10.7%, Si = 13.9%, Fe = 25.2% and Mg = 10.6%. This generates the following probable composition: $Mg_{2.77}\ Fe_{2.84}\ Al_{0.39}\ (Si_{3.13}\ Al_{0.86})\ O_{10}\ (OH)_8$.

Garnet was also analyzed by SEM-EDS and it was found that the composition is much more variable than the chlorite. The predominant garnet seems to have an intermediate compsition between grossular and andradite. The most probable formula is $Ca_{2.13}\ (Al_{1.98}\ Fe_{1.02})\ Si_{2.86}\ O_{12}$.

In order to obtain a quantitative mineralogical analysis, the chemical analysis of the major oxides was combined with qualitative mineralogical data. The stoichiometric balance was based on the most probable formula determined either by XRD or semi-quantitative microanalysis by SEM/EDS. Descriptions of, and procedures for XRD and SEM are shown in Appendix C. SEM showed that iron was in all of the samples in the magnetite and chlorite. There were also trace amounts of chalcopyrite, pyrite, apatite, goethite, ilmenite, zircon and gold.

Table 9 shows the results of the stoichiometric balance for the major minerals. The percentages of minerals reported in Table 8 are very approximate as they are based on the most probable mineralogical formula and distribution of the major oxides in these minerals. The amount of garnet in Composite 2 was considered very low as this mineral was calculated based on the remaining CaO and Al_2O_3 after allocating to calcite and chlorite, respectively. Despite the imprecision, the calculated mineralogical composition is in agreement with the results from heavy liquid separation indicating that Composite 3 has a higher concentration of heavy minerals (garnet and magnetite) than the other composites. This characteristic associated with the fact that Composite 3 has the highest Cumulative Liberation Yield and the highest percentage of fines (Fig. 3) makes this sample, i.e. this part of the Tremblay tailing pond, the most attractive for re-processing. For this reason, the metallurgical tests were focused on this composite sample.

Table 8: Mineralogical Compositions Calculated by Stoichiometry

MINERAL:	**Probable Formula**	**comp 1 (%)**	**comp 2 (%)**	**comp 3 (%)**
Chlorite	$(Mg, Al, Fe)_6[(Si,Al)_4 O_{10}]OH_8$	19	19	16
Illite	$KAl_2(Si_3AlO_{10})(OH)_2$	5.5	1.5	4
Calcite	$CaCO_3$	34	33	27
Garnet	$Ca_3(Fe,Al)_2Si_3O_{12}$	3.5	low	6
Quartz	SiO_2	30	37	34
Magnetite	Fe_3O_4	4	5	7

6 Metallurgical Testing

6.1 Introduction

This section constitutes a discussion about the processing tests, including grinding, wet drum magnetic separation and flotation. It reports the procedures and results of the metallurgical testwork and sizes appropriate equipment. All metallurgical tests were conducted on the Composite 3 sample *(combination of the individual samples 5, 6 and 7).*

6.2 Procedures

6.2.1 Overview

Three main groups of tests, as well as a Bond Work Index test, were undertaken as follows:

Test #1

The objective of Test #1 was to determine recoveries from rougher magnetic and flotation separation. A 3.1 kg sample was fed through a Sala wet drum magnetic separator (WDMS), producing a rougher magnetic concentrate. The tailing from the magnetic concentration was submitted to flotation at pH 9.3 using 100 g/t of Potassium Amyl Xanthate and Dowfroth 250 as a frother. After 3 minutes of flotation, the pH was reduced to neutral condition and more PAX was added. This was aimed at evaluating whether copper, silver and gold could be recovered in rougher flotation. The flotation tailings were analyzed for garnet. A summary of test procedures and conditions are reported in Table 9.

Table 9: Summary of test #1 procedures and conditions

Stage	Equipment	Description
Wet Drum Magnetic Separation	Sala lab separator at 30% Solids	Davis tube to establish magnetics content
Bulk Sulphide Flotation	Denver Flotation Cell at ~30% Solids	100g/t PAX @2% solids + 1 drop DowFroth 250, float 3min at pH=9.3; Lower pH=6.9, 50 g/t PAX, 1 drop frother, float 3 min; pH=7.0 lower to pH=6.3, 50g/t PAX, 1 drop frpther, float 3 min.

The results of Test #1 reported in the next section indicate that cleaning stages are required to produce saleable magnetite and metal products.

Test #2

The objectives of Test #2 were to observe the metallurgical performance of cleaning magnetic separation and cleaning flotation as well as the effect of grinding on magnetite and metal recovery.
Test #2 constituted a 15 minute pre-grinding of 8.6 kg of feed, followed by wet drum magnetic separation (WDMS). The magnetite concentrate was reground and several cleaning stages of

magnetic separation were applied. The tailing of the rougher magnetic separation stage was floated using xanthate and frother to determine the possibility of recovering copper, silver or gold. Two additional cleaning flotation stages were performed to determine if the metal product grade can be improved. No high intensity magnetic separation tests were conducted because the material was too fine to generate a garnet product. A summary of test #2 procedures and conditions are reported in Table 10. The rougher flotation test was performed at neutral pH, 100 g/t of PAX, Dowfroth 250 as frother in three stages. The rougher concentrate was cleaned in two 3-min flotation steps.

Table 10: Summary of test 2 procedures and conditions

Stage	**Equipment**	**Description**
Preliminary Grinding	65% Solids in lab rod mill	Grind time of 15 minutes
Wet Drum Rougher Magnetic Separation	Sala lab separator at 30% solids	Davis Tube to calculate magnetics content
Magnetite Regrind	65% solids in lab rod mill on rougher magnetite concentrate	Grind time of 15 minutes
Wet Drum Cleaner Magnetic Separation	Sala lab separator at 30% solids	Davis Tube to calculate magnetics content; 4 cleaner stages
Bulk Sulphide Flotation	Denver flotation cell at ~30% solids	pH=6.9; 3 min float with 100 g/t PAX, 1 drop Dowfroth 250, 3 stages
Cleaner Sulphide Flotation	Denver flotation cell at ~30% solids	pH=6.9; 2 cleaner stages; 3 min float with 1 g/t PAX, Dowfroth on rougher flotation concentrate.

The results of Test #2 indicated that metal concentrates need to be upgraded to produce saleable products.

Test #3

The objectives of Test #3 were to observe if different operating conditions could improve the grade of the cleaner flotation product and if a saleable garnet product could be obtained. Test #3 constituted a 3 minute pre-grinding of 2.2 kg of feed, followed by WDMS, producing a rougher magnetic concentrate. The concentrate from the magnetic separation was not further tested because Test #2 produced adequate results for a cleaner magnetite product. The WDMS tailings were floated in a rougher stage. The rougher flotation concentrate was reground to clean-up the sulfide surfaces and provide further liberation. After regrinding, the rougher concentrate was subjected to two stages of cleaner flotation tests conducted at pH 11 to determine if a high grade metal concentrate could be produced. Additionally, the rougher flotation tailings were screened at 100 mesh. The plus +100 mesh fraction material was concentrated in a high intensity magnetic separator (HIMS) to determine if garnet (paramagnetic) could be recovered in the non-magnetic product. To upgrade the garnet product, dry low intensity magnetic separator and electrostatic separator tests were conducted on the garnet product. A summary of test #3 procedures and conditions are reported in Table 11.

Table 11: Test #3 procedures and conditions

Stage	Equipment	Description
Preliminary Grinding	65% Solids in lab rod mill	Grind time of 3 minutes
Wet Drum Rougher Magnetic Separation	Sala lab separator at 30% solids	Davis Tube to calculate magnetics content
Bulk Sulphide Flotation	Denver flotation cell at ~30% solids	pH=6.9; 3 min float with 100 g/t PAX, 1 drop Dowfroth 250, 3 stages
Flotation Regrind	Lab Rod Mill	Grind time of 15 minutes
Cleaner Sulphide Flotation	Denver flotation cell at ~30% solids	pH=6.9; 2 cleaner stages; 3 min float with 1 g/t PAX, dowfroth on rougher flotation concentrate.
Screening	Wet screen rougher flotation tails	100 mesh screen size
High Intensity Magnetic Separation	+ 100 mesh material	

The Bond Work Index was determined by the standard test procedures[1].

6.2.2 Details of Testwork Procedures

All of the grinding procedures took place in the bench-scale rod mills. The samples were ground at 65% solids by weight.

The rougher flotation tests were conducted by bench-scale batch tests. All three tests floated the WDMS tails for 9 minutes, at a pH of 6.9, with 100g/t of PAX added in three separate stages. One or two drops of Dowfroth 250 were used as a frother for all the tests. The concentrate and tails were filter pressed and dried in the oven.

After drying, the samples were pulverized and sent for assay. Copper was analyzed by atomic absorption spectrometry and gold and silver were analyzed by fire assay.

Screening analyses were conducted either wet or dry. For dry screening a Roto-Tap was run 15 minutes. The various size fractions were then weighed and recorded.

Magnetite concentrates were assayed using a Davis Tube low-intensity magnetic separator. About 20 to 30 grams of representative material from various products were riffled, slurried, and slowly passed through the Davis Tube. After all the slurry was passed through the Davis tube, wash water was introduced to clean the retained magnetite. Then the magnetic field was discontinued and the magnetic fraction was collected, dried, and weighed. This method also collects unliberated magnetite, so the method may overestime the magnetite content.

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[1] Weiss,N. (Ed) SME Mineral Processing Handbook, SME-AIME, Littleton, CO, 1986

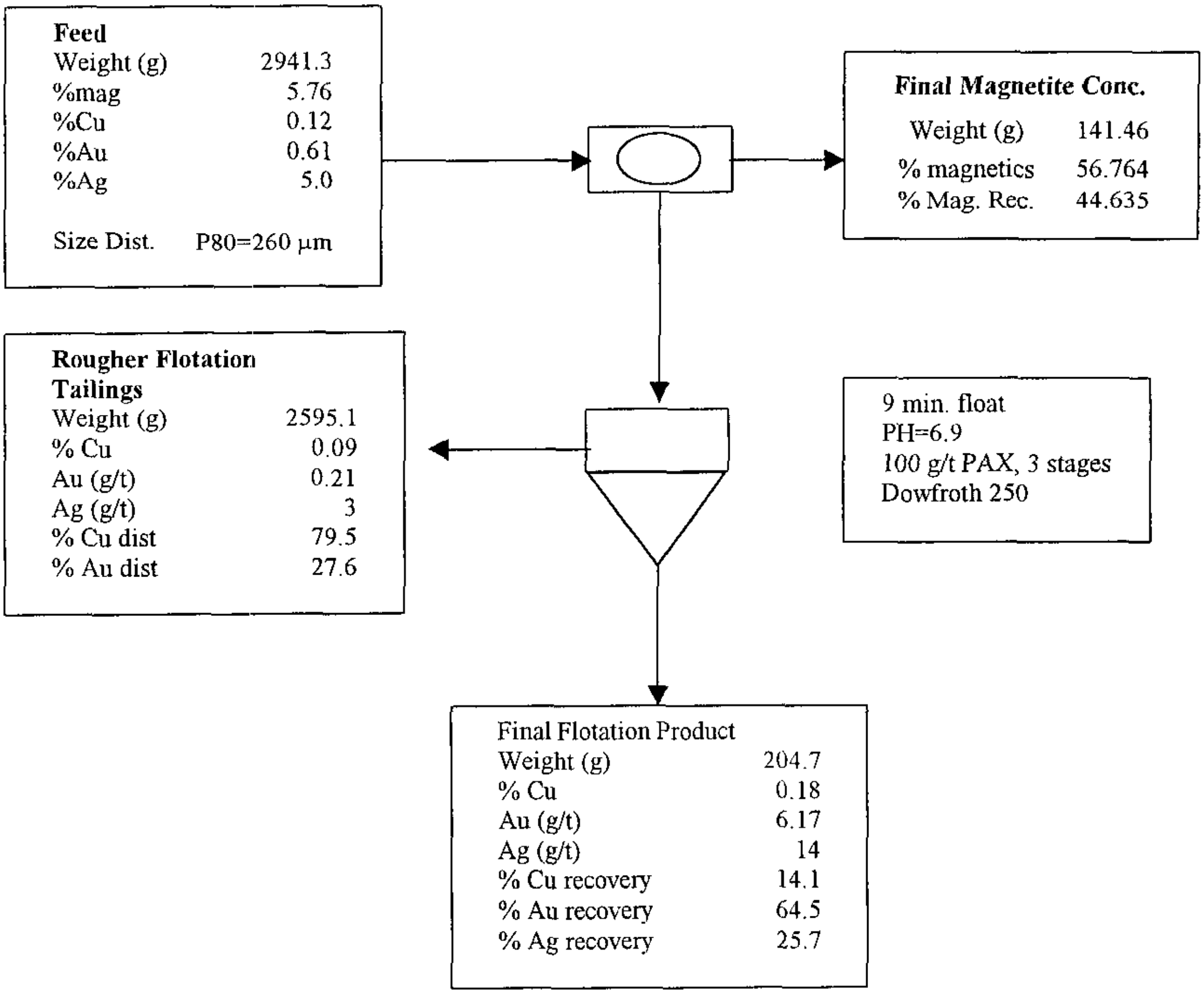
6.3 Results & Discussion

6.3.1 Bond Work Index

The Bond Work Index test determined to be 14.05 kWh/tonne.

6.3.2 Test #1

Figure 10 shows that rougher magnetic separation of Composite 3 using WDMS proceduces a concentrate with 56.7% magnetite and a recovery of 44.6%. Flotation produced a concentrate with 0.18% Cu and 6.17g/t Au. The low copper recovery is an indication that grinding is needed. The gold recovery of about 63% is an indication of the presence of free gold as highlighted elsewhere in previous reports.



Feed

Weight (g)	2941.3
%mag	5.76
%Cu	0.12
%Au	0.61
%Ag	5.0
Size Dist.	P80=260 µm

Final Magnetite Conc.

Weight (g)	141.46
% magnetics	56.764
% Mag. Rec.	44.635

Rougher Flotation Tailings

Weight (g)	2595.1
% Cu	0.09
Au (g/t)	0.21
Ag (g/t)	3
% Cu dist	79.5
% Au dist	27.6

9 min. float
PH=6.9
100 g/t PAX, 3 stages
Dowfroth 250

Final Flotation Product

Weight (g)	204.7
% Cu	0.18
Au (g/t)	6.17
Ag (g/t)	14
% Cu recovery	14.1
% Au recovery	64.5
% Ag recovery	25.7

Figure 10: Flowsheet and Results of the Concentration Stages Used in Test #1

6.3.3 Test #2

A marketable magnetite product must first meet the requirements of current consumers. For the purpose of this study, several Elk Valley coal producers were consulted in order to estimate magnetite consumption rates. Magnetite product specifications are determined by the required characteristics for dense medium in coal plants.

It was the goal of the magnetic separation to produce a magnetite concentrate suitable for dense medium with the following characteristics:

% passing #325 (or 44μm) = 90
Magnetite Content = 93-95%
Specific Gravity > 4.7

It was evident that the rougher magnetic separator concentrate obtained in Test #1 was not of acceptable product quality. The magnetite content was too low, in addition to specific gravity and size specifications. Further treatment of the rougher concentrate was required to upgrade the product quality. A sizeable amount of material was used to produce an adequate amount of feed for the cleaner circuit and a subsequent final magnetite product.

The wet drum magnetic separation was conducted with material ground in a lab rod mill for 15 min. Grain size analysis of 15 min grinding of the rougher magnetic separation feed is shown in Table 12 and the P80 obtained was 58 μm. Regrinding of the concentrate included to liberate magnetite particles from gangue material. The magnetic product was reground for further 15 min (grain size analysis shown in Table 13) generating a product with a P80 = 25 μm. This material was resubmitted to cleaning stages with WDMS.

Table 12: Size Distribution of 15 min grinding (Test #2)

Tyler Mesh	**Size (μm)**	**Weight (%)**	**Cumulative % Retained**	**Cumulative. % Passing**
48	300	0.02	0.02	99.98
65	210	0.03	0.04	99.96
100	150	0.09	0.13	99.87
150	105	1.62	1.75	98.25
200	74	11.03	12.78	87.22
270	52	9.99	22.77	77.23
325	37	14.30	37.07	62.93
-325	-34	62.93	100.00	0.00
Total		100.0		

Table 13: Size Distribution of 15 min re-grinding (Test #2)

Tyler Mesh	**Size (μm)**	**Weight (%)**	**Cumulative % Retained**	**Cumulative % Passing**
270	52	6.85	6.85	93.15
325	34	2.60	9.45	90.55
400	24	12.07	21.52	78.48
-400	-24	78.48	100.00	0.00
Total		100.00		

The subsequent cleaner stages were necessary to achieve the desired specification. Specific gravity and grade targets can be attained through the inherent desliming action of the cleaner stages, in addition to rejection of liberated non-magnetic middling particles. Cleaning procedures are summarized as follows:

- First cleaner stage using the Sala low intensity wet drum magnetic separator (WDSM)
- First cleaner concentrate was analyzed for product specifications
- First cleaner tailings was analyzed for magnetite
- When product did not meet specification, cleaning procedure was applied
- Three more cleaning stages with the wet drum magnetic separator were applied
- Final magnetite product was obtained

The number of cleaning stages required is related to the amount of slimes and liberation of magnetite in the feed.

The magnetic concentrate (test #2) following one cleaning stage was analyzed to evaluate the recovery of magnetite. This product contained 50.4% magnetite (still unliberated) as evaluated by the Davis Tube method and the recovery was around 10%. The product had a specific gravity of 3.7 and a grain size of 83% passing 325 mesh. The XRD pattern (Appendix I-E) shows that the magnetic concentrate contained high amounts of magnetite and quartz, some calcite, garnet, pyrite, hematite and chlorite. The x-ray mappings obtained with the SEM/EDS (Appendix II-E) confirmed that magnetite is the major mineral in the product. The mappings also illustrate that the concentrate still had calcite, chlorite and some unliberated magnetite. The magnetite is associated with chlorite and garnet. This suggested that the concentrate could be cleaned once more.

The characterization of the magnetite concentrate from test #2 after four cleaning stages was aimed at evaluating if a product containing between 93% and 95% of magnetite could be produced. The XRD and SEM/EDS analysis (Appendices I-F and II-F respectively) showed that the concentration of magnetite was around 96% (calculated by simplified quantitative XRD). Quartz, chlorite, pyrite, garnet and calcite were still present as impurities in the product. This final product had the following specification:

% passing #325 (or 44μm) = 67.18
Magnetite Content = 96.05%
Specific Gravity = 4.66
Product Yield as a total of plant feed = 1.89%

Immediate improvements of the magnetite product can be made with respect to the particle size. These requirements can easily be met by increasing the regrinding time of the feed prior to cleaning. This can also increase the S.G. by liberating middling particles and improve the magnetite content. From this modification, a marketable magnetite product can be obtained.

As mentioned, the number of cleaning stages is an issue that needs to be addressed. This problem can be attributed to slimes in the feed. The plant feed would be fine (80% -250μm), and any grinding would add additional slimes. It was observed in the lab tests that high amounts of slimes carried over into the concentrate of rougher separation, but cleaning stages were able eventually removed the slimes.

Slimes in the slurry pose several hindrances in achieving high product quality:

- The higher the slimes content, the more difficult it is to clean the concentrate
- S.G. and magnetite grade targets are more difficult to achieve
- Particle size measurements include slimes which can misrepresent the actual size of the magnetite

To remedy the slime problem, a possible magnetite cleaner stage can be implemented prior to regrinding to improve liberation and reduce equipment size as well as reduce number of cleaning stages downstream.

Given the Test #2 results and final magnetite product characteristics, a marketable product can be achieved. More work has to be done in order to determine the most effective method of recovery, such as cleaning the magnetite concentrate prior to regrinding.

The final flotation product graded only 1.2% Cu with very low recovery of 3.1%. Gold grade reached 20 g/t in the final concentrate but the recovery was also only 3%. Figure 11 shows the flowsheet and results of the concentration stages used in Test #2.

The rougher flotation tailing from test #2 was analyzed by XRD and SEM. The XRD pattern (Appendix I-G) shows the presence of quartz, calcite, chlorite and garnet. The x-ray mapping (Appendix II-G) also shows the presence of a small amount of locked magnetite that could not be detected by XRD analysis as it represents less than 1% of the overall sample weight. Copper, silver, or gold, at very low concentration in the sample, 0.08%, 0.16 g/t and 3 g/t respectively, were not found in the SEM/EDS analysis or detected by XRD.

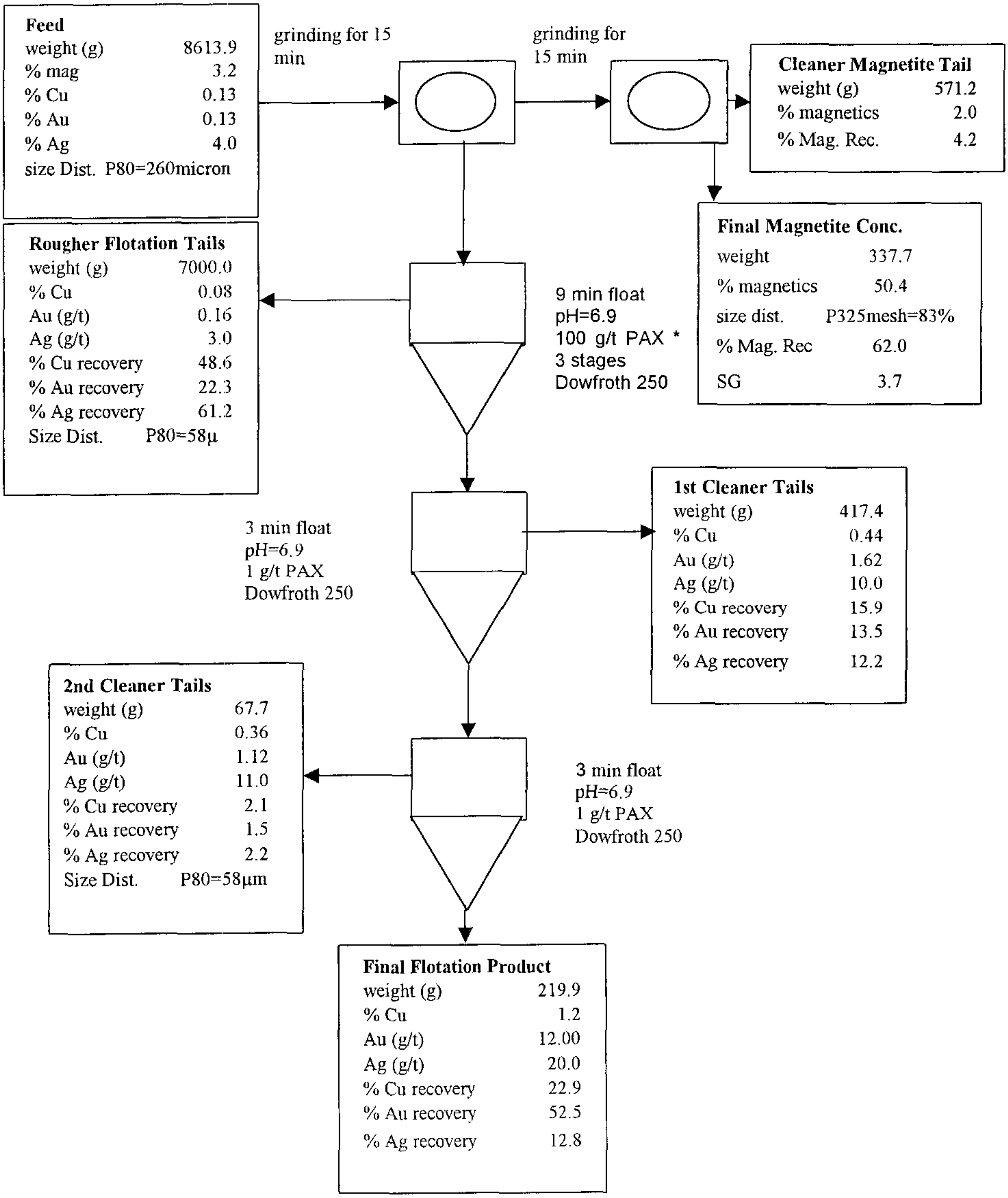


Figure 11: Flowsheet and Results of the Concentration Stages Used in Test #2

6.3.4 Test #3

As mentioned before the objectives of Test #3 were to improve flotation of metals and to attempt to obtain a saleable garnet product. The material was ground for 3 minutes obtaining P80=139 µm. The grain size distribution is shown in Table 14. The results of the wet screening, dry high intensity magnetic separation, dry low intensity magnetic separation, and electrostatic separation tests are shown in Table 15. These results show that the +100 mesh fraction of rougher flotation tailings contains 20.6% garnet (yield = 15.9%) and less than 0.2% magnetite.

The +100 mesh garnet concentrate from test #3 rougher flotation tailing was analyzed to determine if the product contains saleable garnet mineral. The XRD and SEM/EDS analyses (Appendices I-H and II-H) show the presence of quartz, chlorite, calcite and garnet. Magnetite is still locked inside others minerals. Optical microscopy was used to distinguish garnet and establish garnet grade by gross-counting method. The Figure 12 illustrates a common feature in this sample, fine inclusions of magnetite inside green garnet grains.

The rougher flotation concentrate was reground and submitted to two stages of cleaner flotation tests with pH higher than the other tests. The XRD analysis of the second cleaner flotation concentrate (Appendix I-I) shows the presence of pyrite, calcite and quartz. The x-ray mapping (Appendix II-I1 & I2) shows that Fe and S are combined indicating the predominance of pyrite as the floated sulfide. The map also indicates the elements that constitute calcite, quartz and magnetite. In some regions, it is possible to find magnetite particles locked inside pyrite.

Table 14: Size Analysis of Material ground for 3 min. (Test #3)

Tyler Mesh	Size (µm)	Weight (%)	Cumulative % Retained	Cumulative % Passing
100	150	15.64	15.64	84.36
150	106	18.16	33.80	66.20
200	75	11.24	45.04	54.96
270	53	17.04	62.08	37.92
325	45	4.22	66.30	33.70
-325	-45	33.70	100.00	0.00
Total		100.00		

Table 15: Mass Balance of the Garnet Concentration Process

Process		% in Flotation Tailing	% in Total Feed
Screening	+100#	20.3	17.0
	-100#	79.7	66.5
	Total	100	83.4
DHIMS	magnetic	15.9	13.3
	non magnetic	4.4	3.7
	Total	20.3	16.9
DLIMS	magnetic	15.6	13.0
	non magnetic	0.3	0.2
	Total	15.9	13.3
Electrostatic	non-conductive	15.8	13.2
Separation	conductive	0.1	0.1
	Total	15.9	13.3

Elimination of 79.7% of the rougher flotation tailings by screening resulted in low garnet recovery. Low garnet grade (20.6%) in the magnetic product indicates that this mineral is not fully liberated and other minerals have a magnetic susceptibility similar to garnet. Less than 1% of mass can be eliminated from the high intensity magnetic product in subsequent low intensity magnetic separation or electrostatic separation unit process. However, 0.2% ferromagnetic and 0.1% conductive materials in +100 mesh of the rougher floatation tailings indicate high recovery of magnetite in wet drum magnetic separator. Figure 13 shows the flowsheet and results of the concentration stages used in test #3.

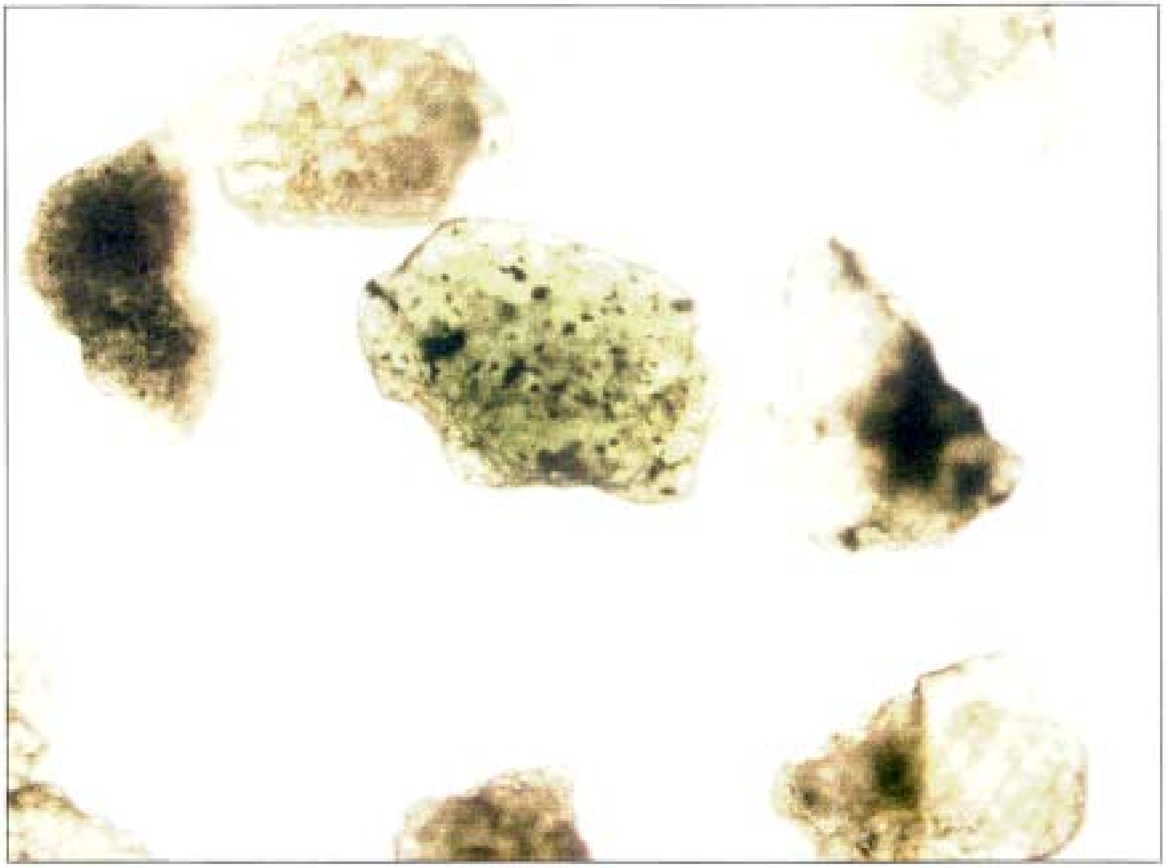
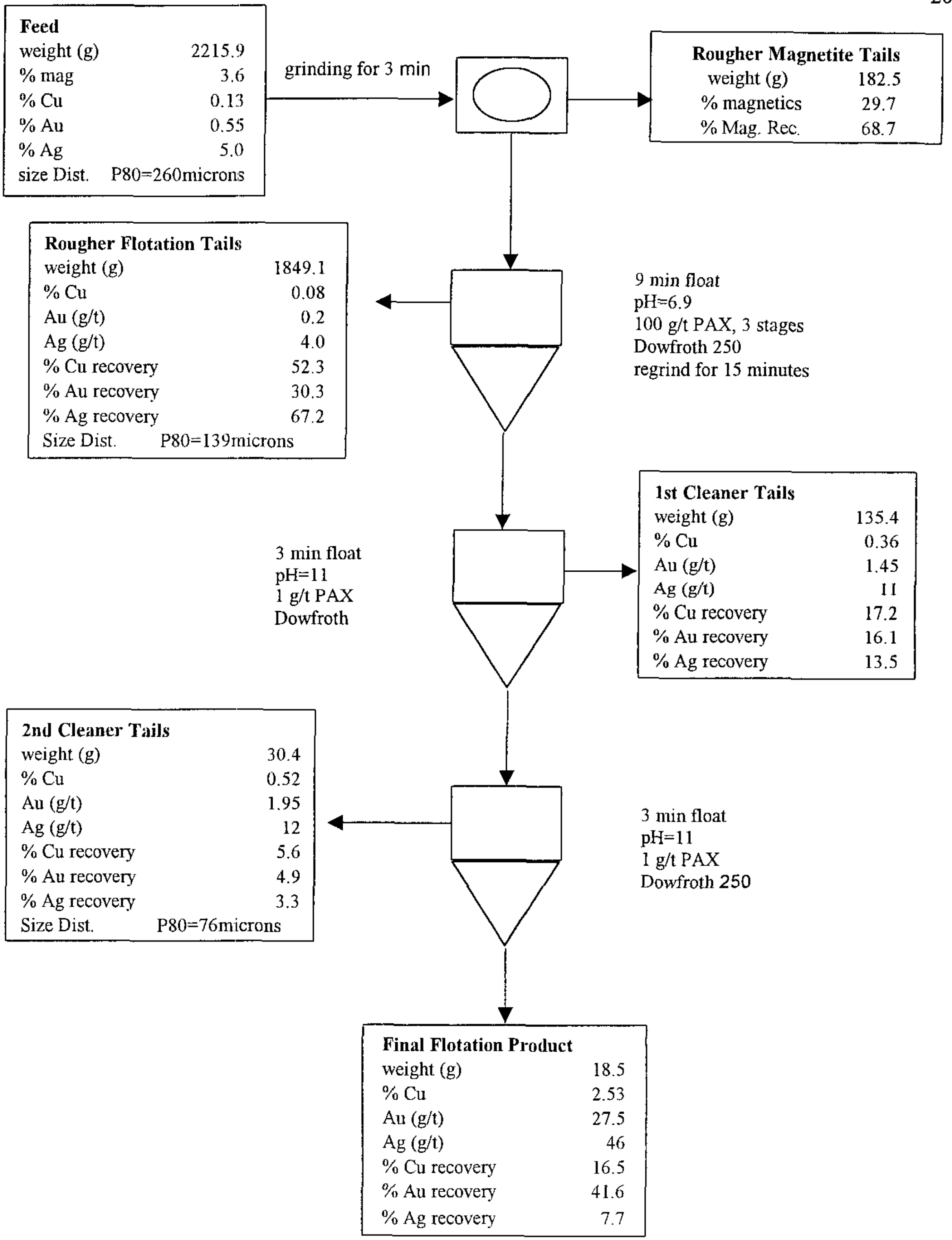


Figure 12: Optical micrograph of unliberated garnet. Black dots are fine magnetite impurities in side green garnet grains.



Feed	
weight (g)	2215.9
% mag	3.6
% Cu	0.13
% Au	0.55
% Ag	5.0
size Dist.	P80=260microns

Rougher Magnetite Tails	
weight (g)	182.5
% magnetics	29.7
% Mag. Rec.	68.7

Rougher Flotation Tails	
weight (g)	1849.1
% Cu	0.08
Au (g/t)	0.2
Ag (g/t)	4.0
% Cu recovery	52.3
% Au recovery	30.3
% Ag recovery	67.2
Size Dist.	P80=139microns

1st Cleaner Tails	
weight (g)	135.4
% Cu	0.36
Au (g/t)	1.45
Ag (g/t)	11
% Cu recovery	17.2
% Au recovery	16.1
% Ag recovery	13.5

2nd Cleaner Tails	
weight (g)	30.4
% Cu	0.52
Au (g/t)	1.95
Ag (g/t)	12
% Cu recovery	5.6
% Au recovery	4.9
% Ag recovery	3.3
Size Dist.	P80=76microns

Final Flotation Product	
weight (g)	18.5
% Cu	2.53
Au (g/t)	27.5
Ag (g/t)	46
% Cu recovery	16.5
% Au recovery	41.6
% Ag recovery	7.7

Figure 13: Flowsheet and Results of the Concentration Stages Used in Test #3.

6.3.5 *Comparison of the Rougher Flotation Tests*

Table 16 shows the results of the 3 rougher flotation tests. The grades of copper and silver results are very low. The longer grind times do not seem to increase gold recovery, although they do increase copper recovery. None of the three tests produced a clean copper product; however, International Metallurgical and Environmental Inc.'s test results show that it is possible to achieve a copper recovery of 17 to 18 %, as well as a gold grade of 200 g/t.[2] Thus more test work should be done on the other samples to evaluate if a copper concentrate could be produced.

Table 16: Results of the 3 rougher flotation tests

Rougher Flotation Concentrate							
		Grade (%)			**Distribution (%)**		
Tests	**Weight (%)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Au (%)**	**Ag (%)**
Test 1	7.0	0.18	6.17	14.0	14.1	64.5	25.7
Test 2	8.2	0.67	4.81	13.2	41.0	67.5	27.2
Test 3	8.3	0.60	4.14	14.7	39.3	62.5	24.5

6.4 Equipment Selection

A preliminary equipment selection and sizing was conducted for a possible flowsheet.

A simplified mass balance is shown in Figure 14.

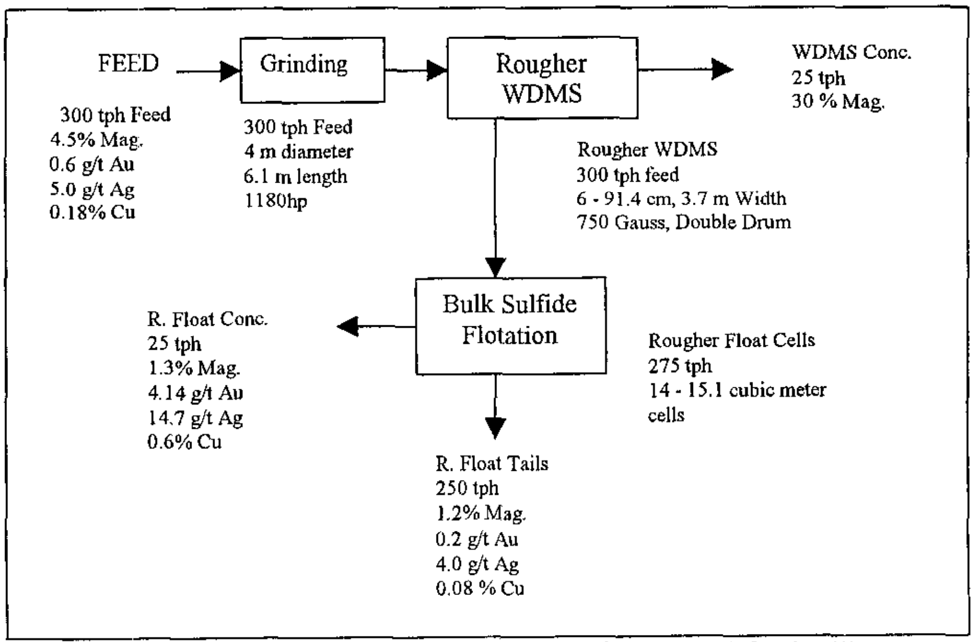


Figure 14: Simplified Mass Balance of a 300 tpd-feed mill

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[2] These results are from two undated reports prepared for Kettle River Resources Ltd.

The design criteria of the project are summarized in Table 17.

Table 17: Design Criteria

Throughput	1,282,000 t/a
Operating Days	180 d/a
Feed Rate	300 tph
Magnetite Recovery	65%
Magnetite Feed Grade	4.5%
Recoverable Magnetite	2.93
Magnetite Production	37,500 tpa
Rougher Ball Mill	4 m∅ x 6.1 m length 1180 hp
Rougher Wet DrumMagnetic Separator	91.4 cm ∅ 750 Gauss, Double Drum
Rougher Flotation Cell Bank	14 of 15.1 m^3

7 Conclusion

The mineralogical studies identified chlorite, illite, calcite, garnet, quartz, magnetite and trace amounts of chalcopyrite, pyrite, apatite, goethite, ilmenite, zircon and gold in the Tremblay tailing.

The mineralogical studies revealed that the distribution of garnet and magnetite in the Tremblay tailing pond is not uniform as these minerals an more concentrated near the creek that cuts through the pond. The magnetite grades range from 4 to 7 % and shows a degree of liberation around 80% at 200 mesh (74 μm). The overall (product) liberation of a Composite sample ranges from 61.4 to 64.5% indicating that grinding is needed to recover magnetite. As a portion of magnetite occurs as microscopic inclusions inside gangue minerals, excessive grinding will not render significant additional liberation. Garnet is consistently full of fine magnetite inclusions and it is very unlikely to generate a saleable product.

The metallurgical tests indicated that rougher magnetic separation of the Composite 3 without grinding provides a concentrate grading 56.7% magnetite with 44.6% recovery. Due to lack of liberation of magnetite particles, re-grinding of the magnetic separation rougher product followed by several cleaning stages of magnetic separation can generate a final magnetite product with characteristics close to a marketable product. In the lab the following product was obtained after three stages of cleaner magnetic separation:
% passing #325 (or 35μm) = 67.18%
Magnetite Content = 96.05%
Specific Gravity = 4.66

The estimated product yield is, however, only 1.9%.

Since a significant portion of the magnetite occurs as minuscule inclusions (around 5 μm) in the gangue minerals, it seems advisable to focus on the recovery only of the relatively coarse liberated magnetite. Pre-concentration (with no grinding) using low intensity magnetic separation would recover these coarse particles or those containing more than 60% locked magnetite. Test #1 exemplifies this process. Re-grinding the magnetic concentrate and cleaning in subsequent stages of magnetic separation should produce a high quality magnetite product.

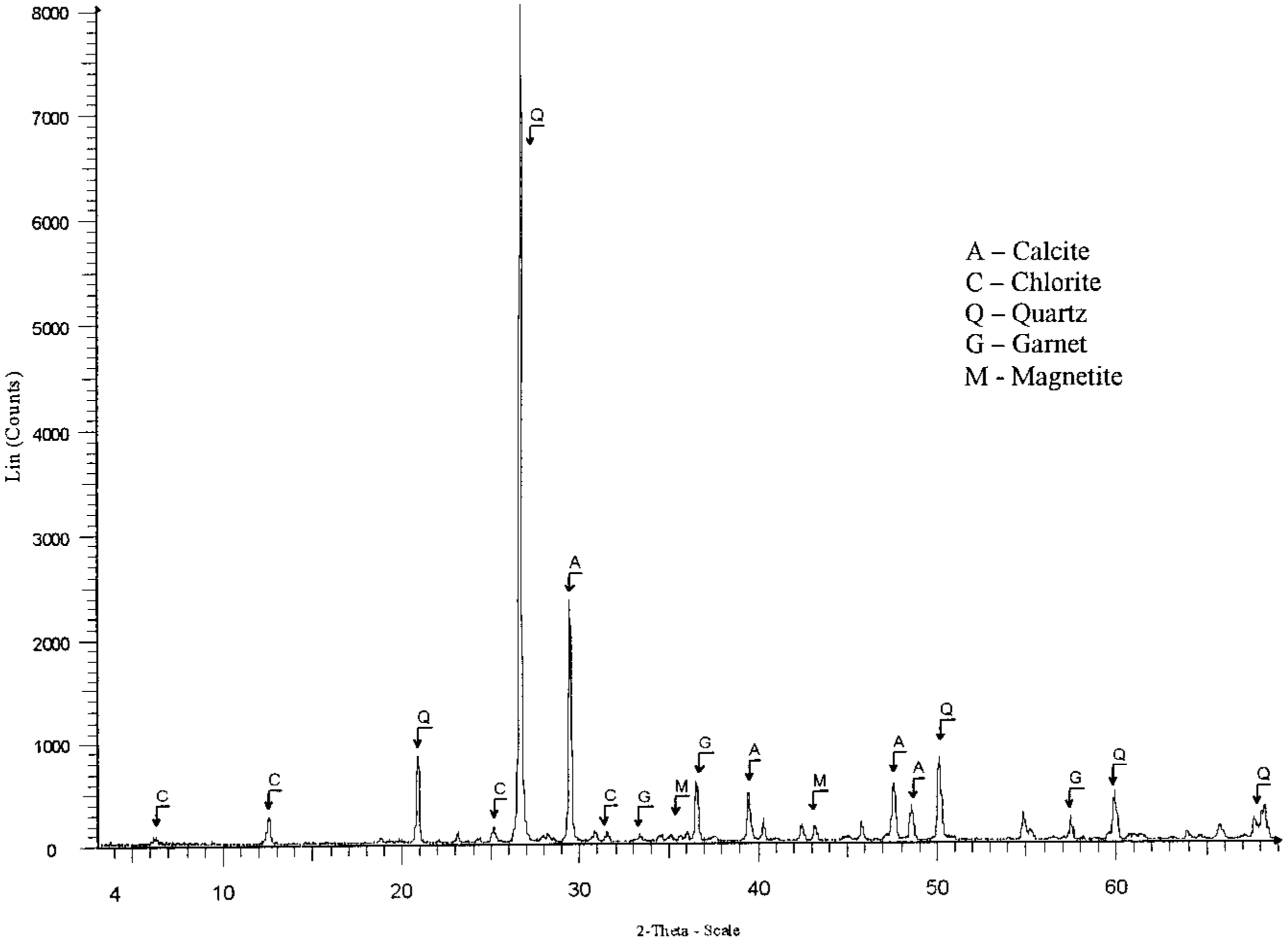
The tests to concentrate garnet resulted in a product with 20.6% garnet (yield = 15.9%), which cannot be considered saleable. The product contained grains of quartz with fine magnetite inclusions.

Copper sulfide flotation did not render a high grade product. The final product graded 1.2% Cu with very low recovery (3.1%). Gold grade reached 20 g/t in the final flotation concentrate but the recovery was also around 3%.

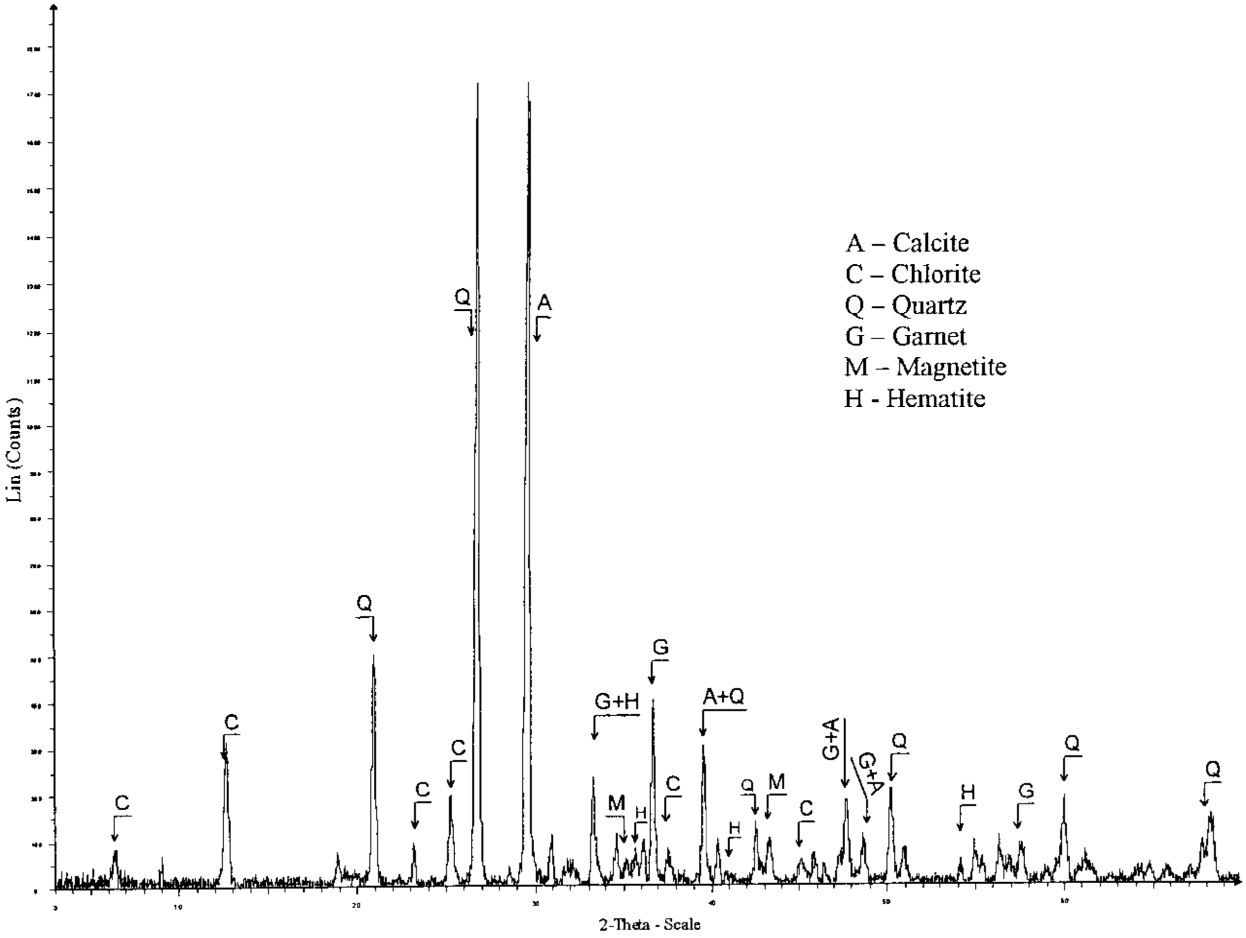
Based on the results conducted on the Composite sample, it is possible to produce a saleable magnetite concentrate. However the yield is low, likely making it uneconomic to justify processing based on only this product.

Products containing copper, gold and garnet were not of sealable quality. Product yields and recoveries were also prohibitively low indicating further upgrading of the products would not improve the project. Results obtained by International Metallurgical conducted on samples from a drilling program produced better metallurgical result, this may be due to the samples used for testing. Therefore, none tests using samples obtained below the surface may be warranted.

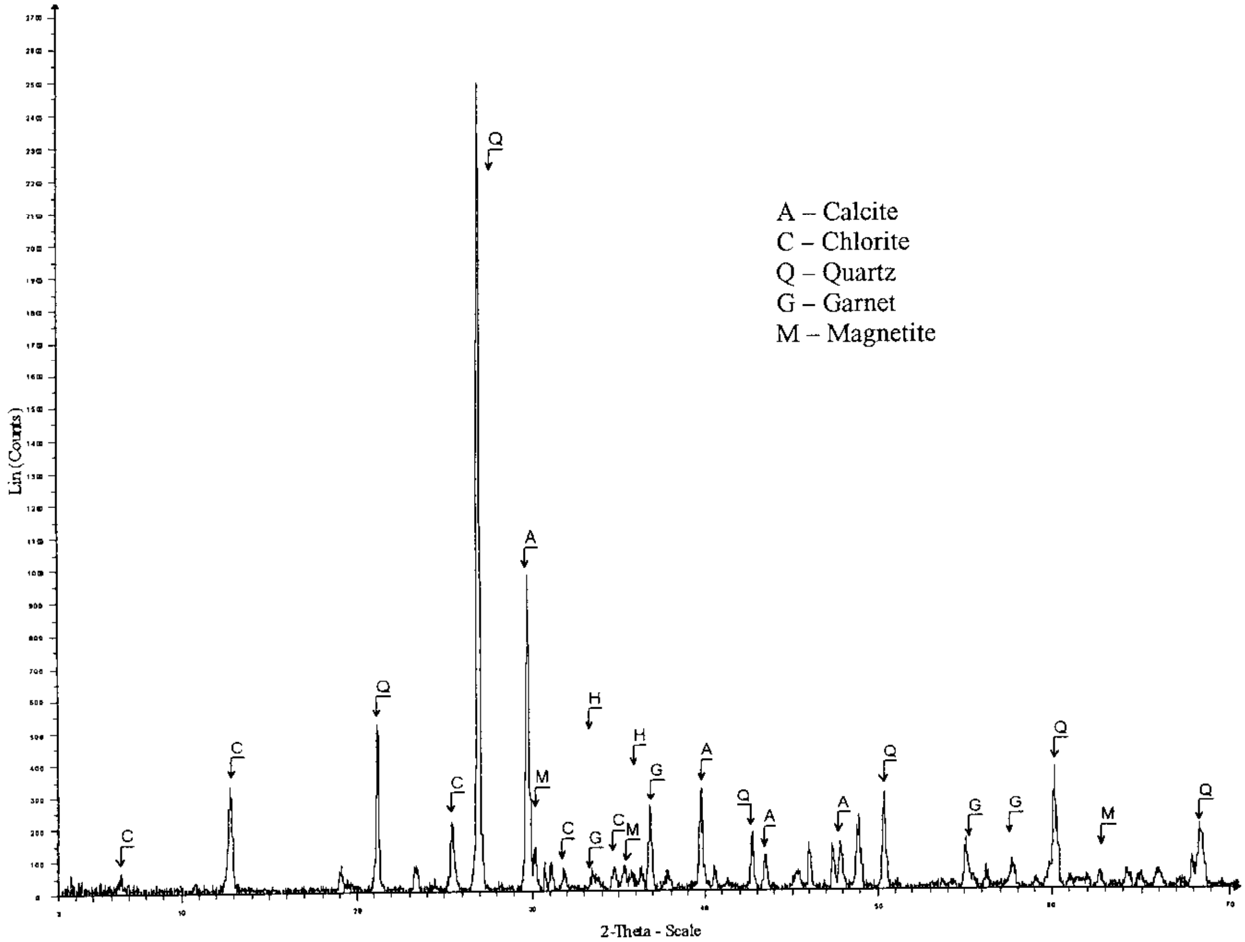
APPENDIX I-A – XRD analysis of the heavy fraction from the +150# fraction of composite 1.



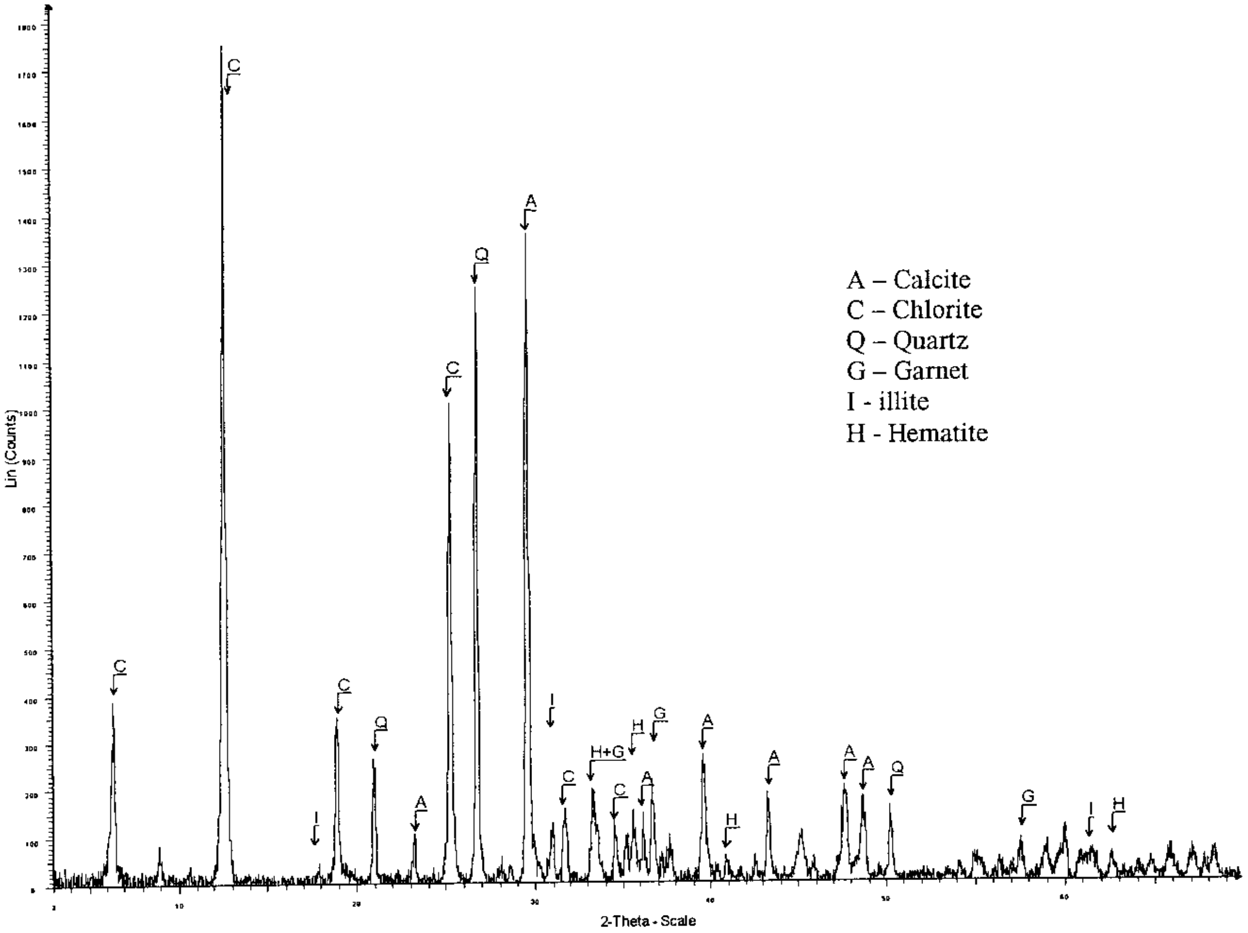
APPENDIX I-B – XRD analysis of the heavy fraction from the +270# fraction of composite 3



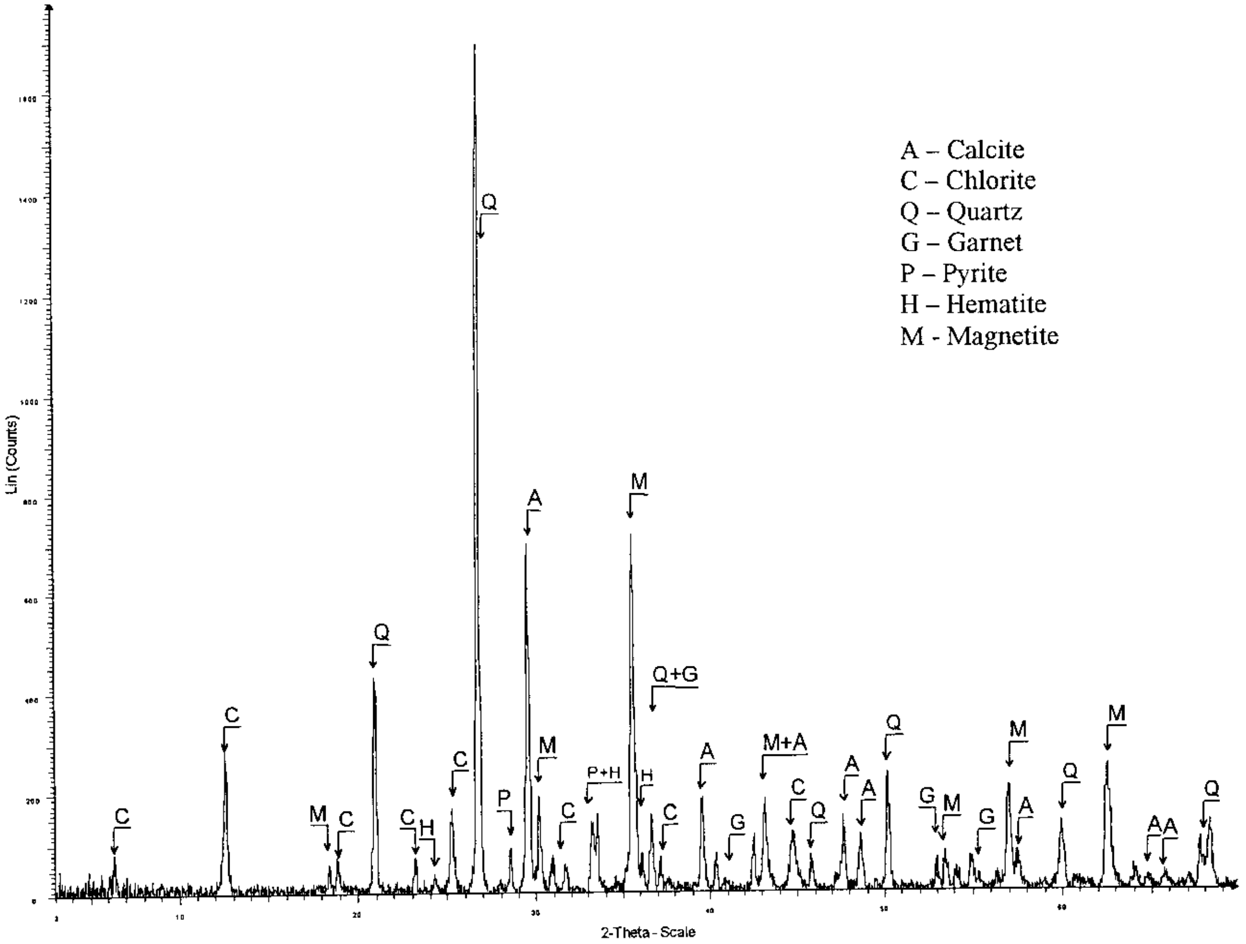
APPENDIX I-C – XRD analysis of the magnetic concentrate from the +270# fraction of composite 3.



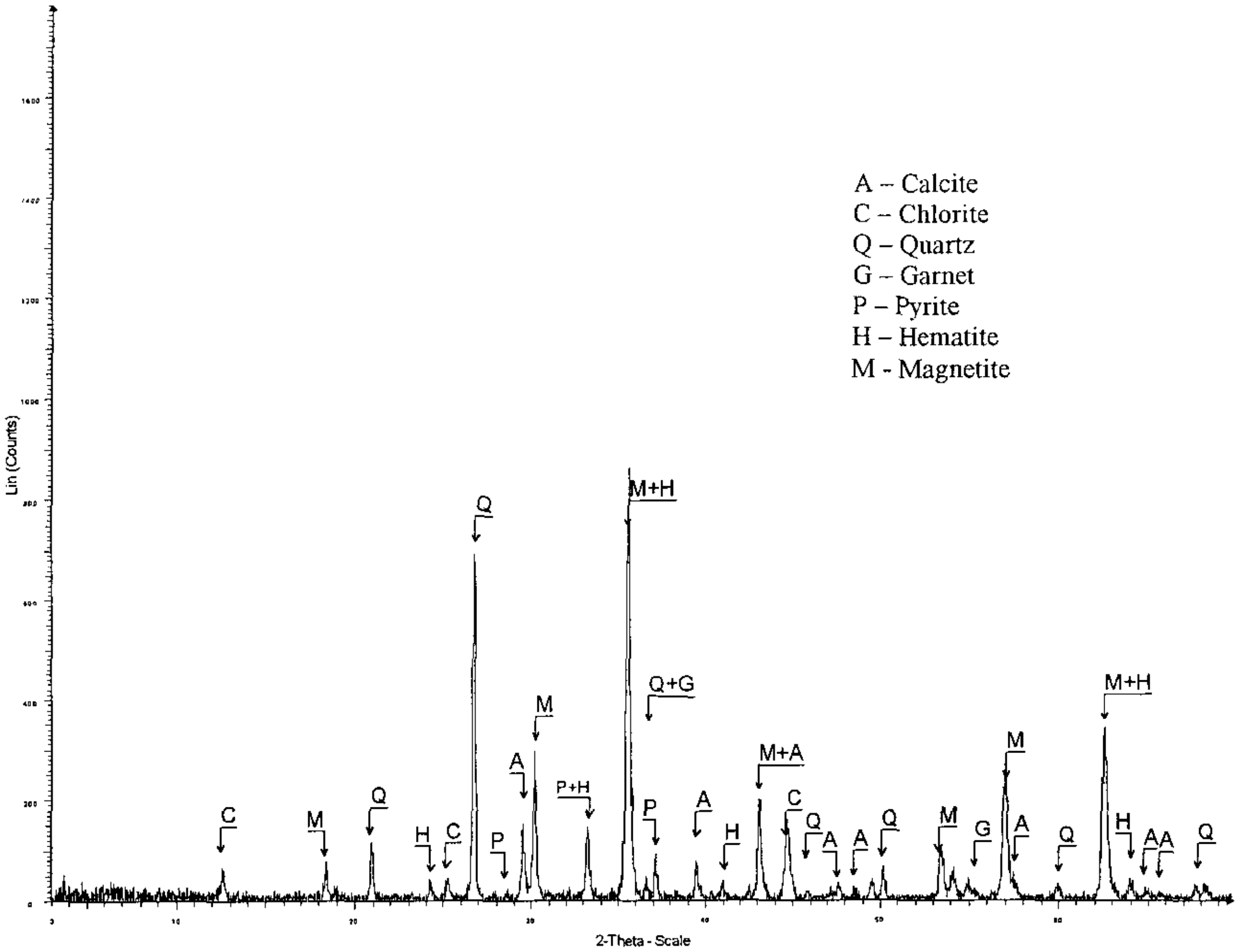
APPENDIX I-D – XRD analysis of the magnetic concentrate of the -400# fraction of composite 3



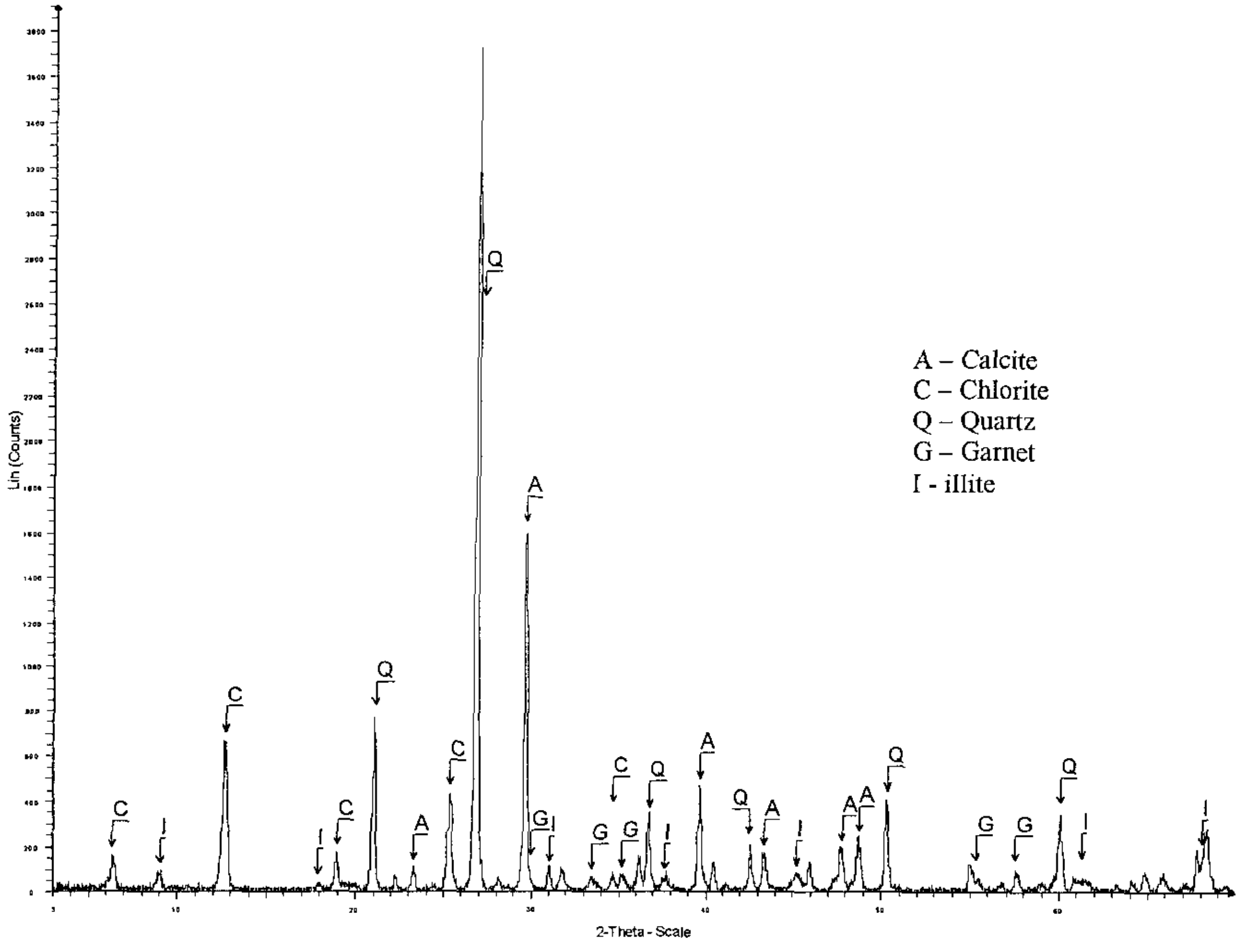
APPENDIX I-E – XRD analysis of the magnetic concentrate of the test #2 after one cleaning stage



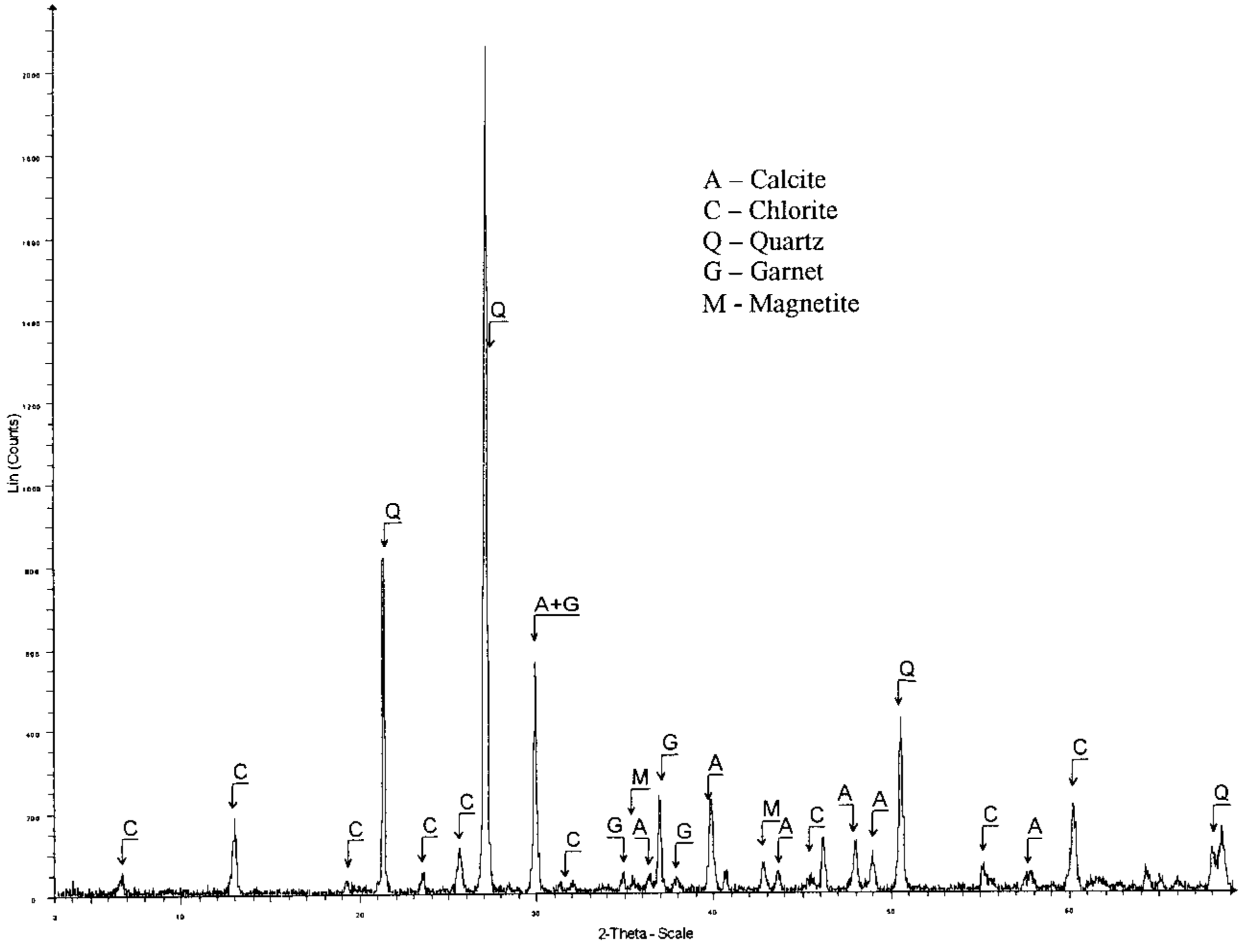
APPENDIX I-F – XRD analysis of the magnetite concentrate of the test #2 after four cleaning stages



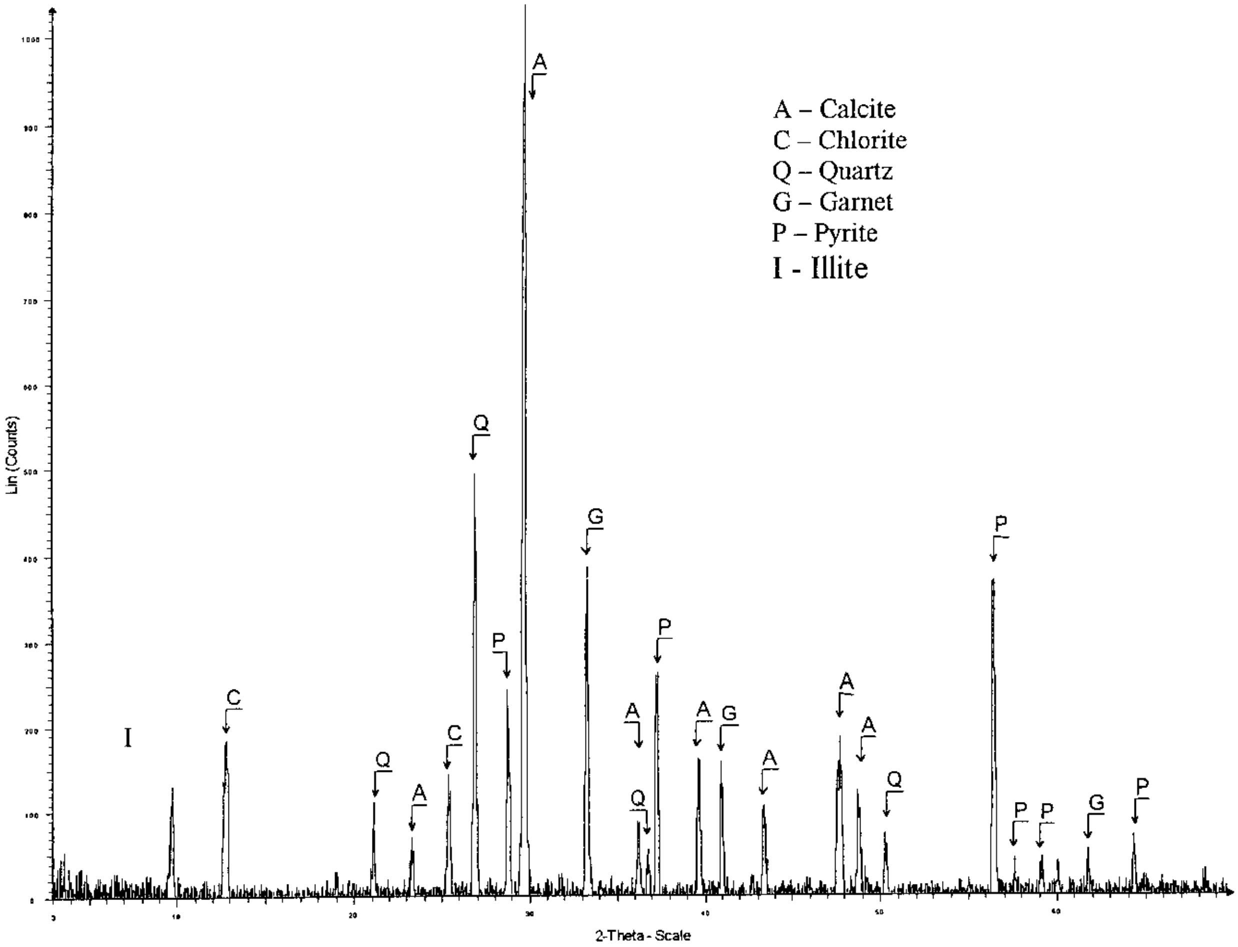
APPENDIX I-G – XRD analysis of the rougher flotation tailing from test #2



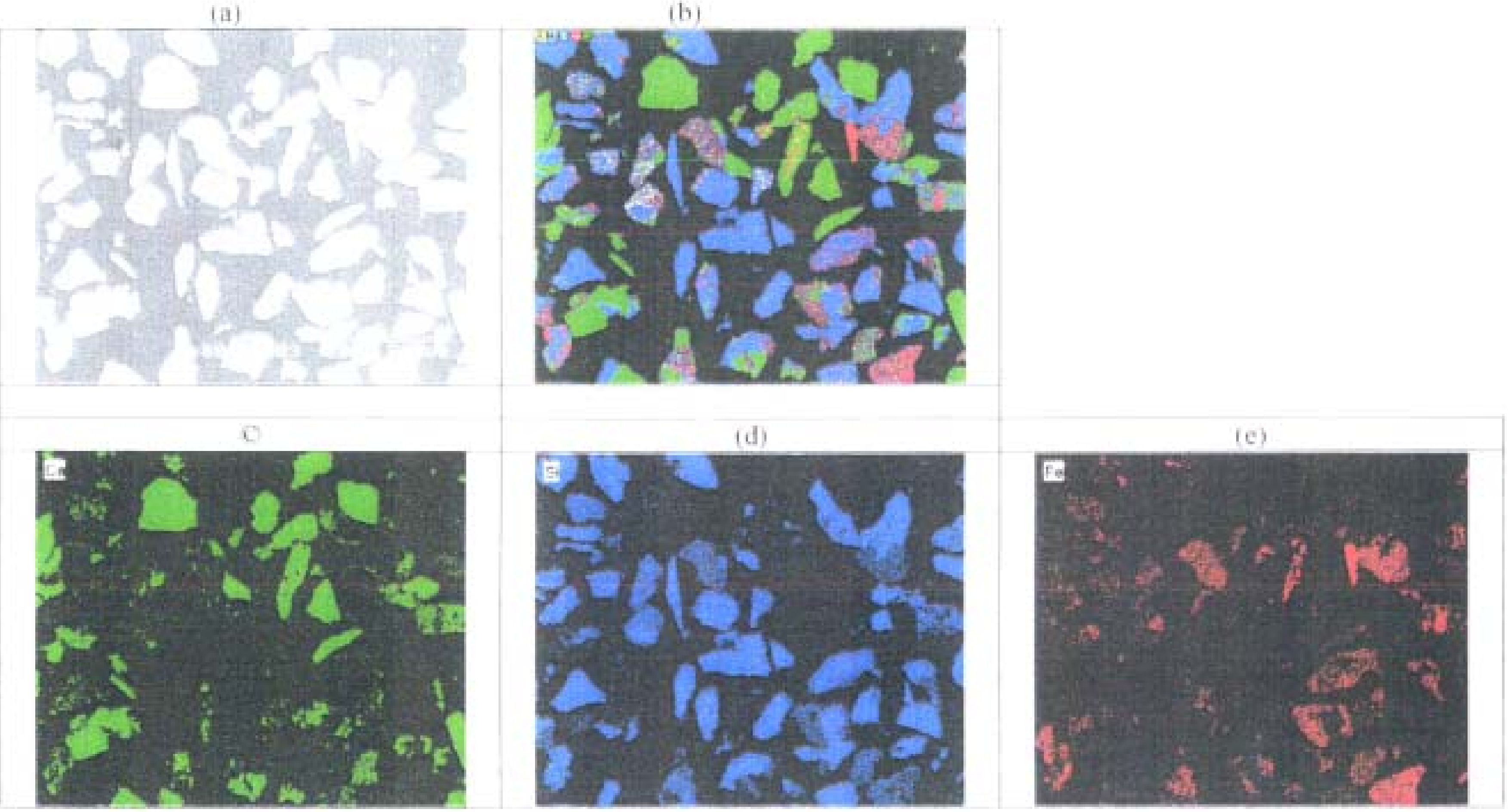
APPENDIX I-H – XRD analysis of the +100 mesh garnet concentrate from test #3 rougher flotation tailing



APPENDIX I-I – XRD analysis of the cleaner flotation concentrate from test #3 (2^{nd} cleaner)

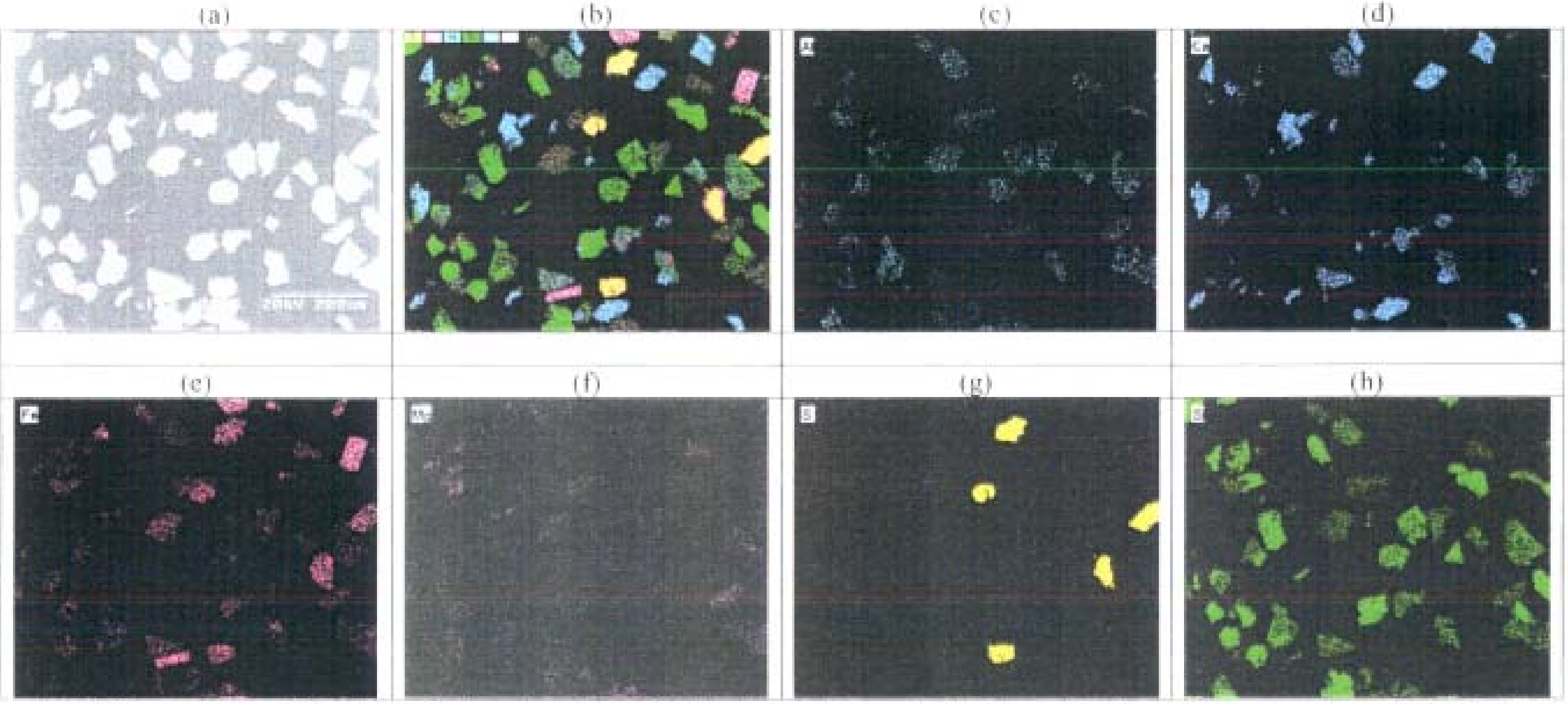


APPENDIX II-A – SEM analysis of the heavy fraction from the +150# fraction of composite 1.



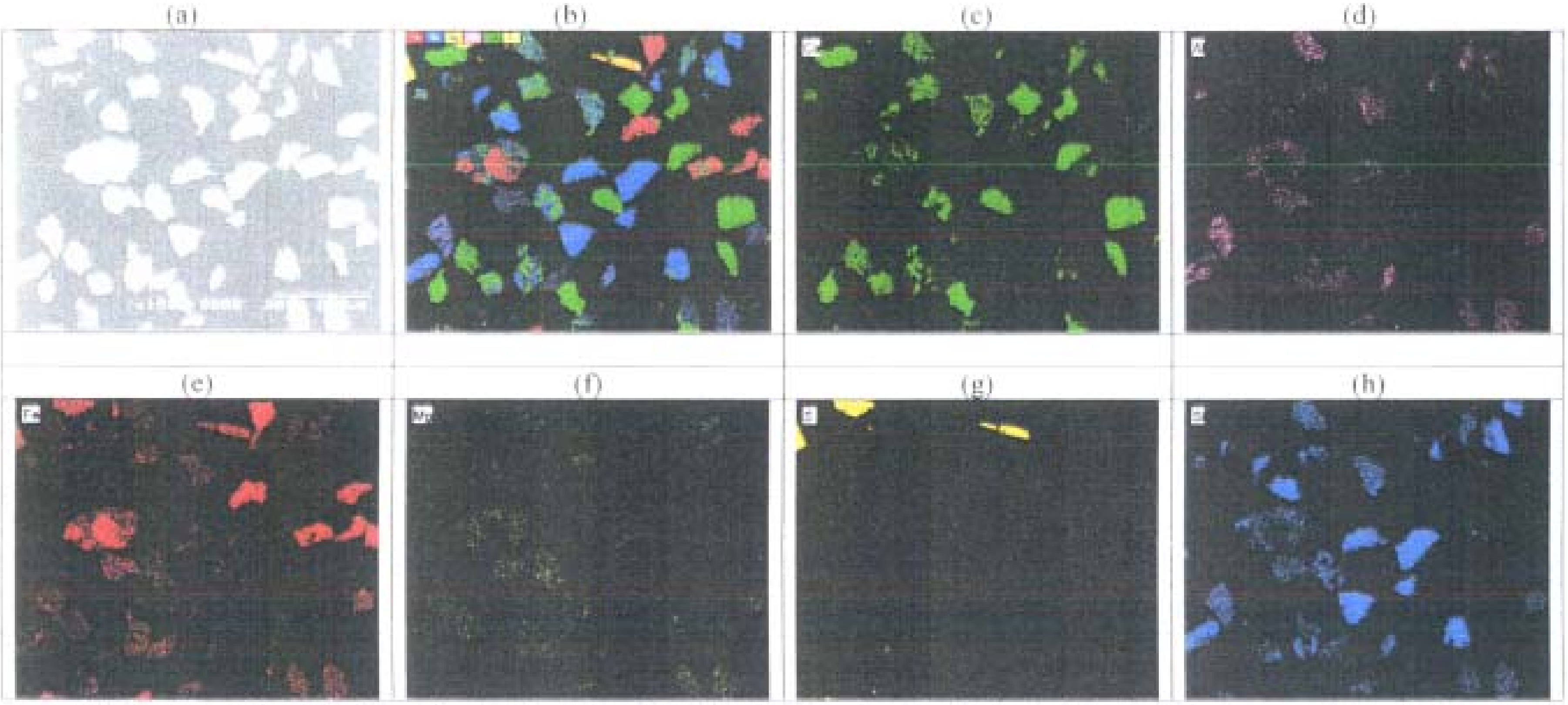
(a) SEM micrograph of the heavy fraction from the +150# fraction of composite 1; (b) combination of x-ray mappings of Ca, Si and Fe; (c) Ca x-ray mapping; (d) Si x-ray mapping and (e) Fe x-ray mapping.

APPENDIX II-B1 – SEM analysis of the the heavy fraction from the +270# fraction of composite 3



(a) SEM micrograph of the heavy fraction from the +270# fraction of composite 3; (b) combination of x-ray mappings of Al, Ca, Fe, Mg, S and Si, (c) Al x-ray mapping, (d) Ca x-ray mapping, (e) Fe x-ray mapping, (f) Mg x-ray mapping, (g) S x-ray mapping and (h) Si x-ray mapping.

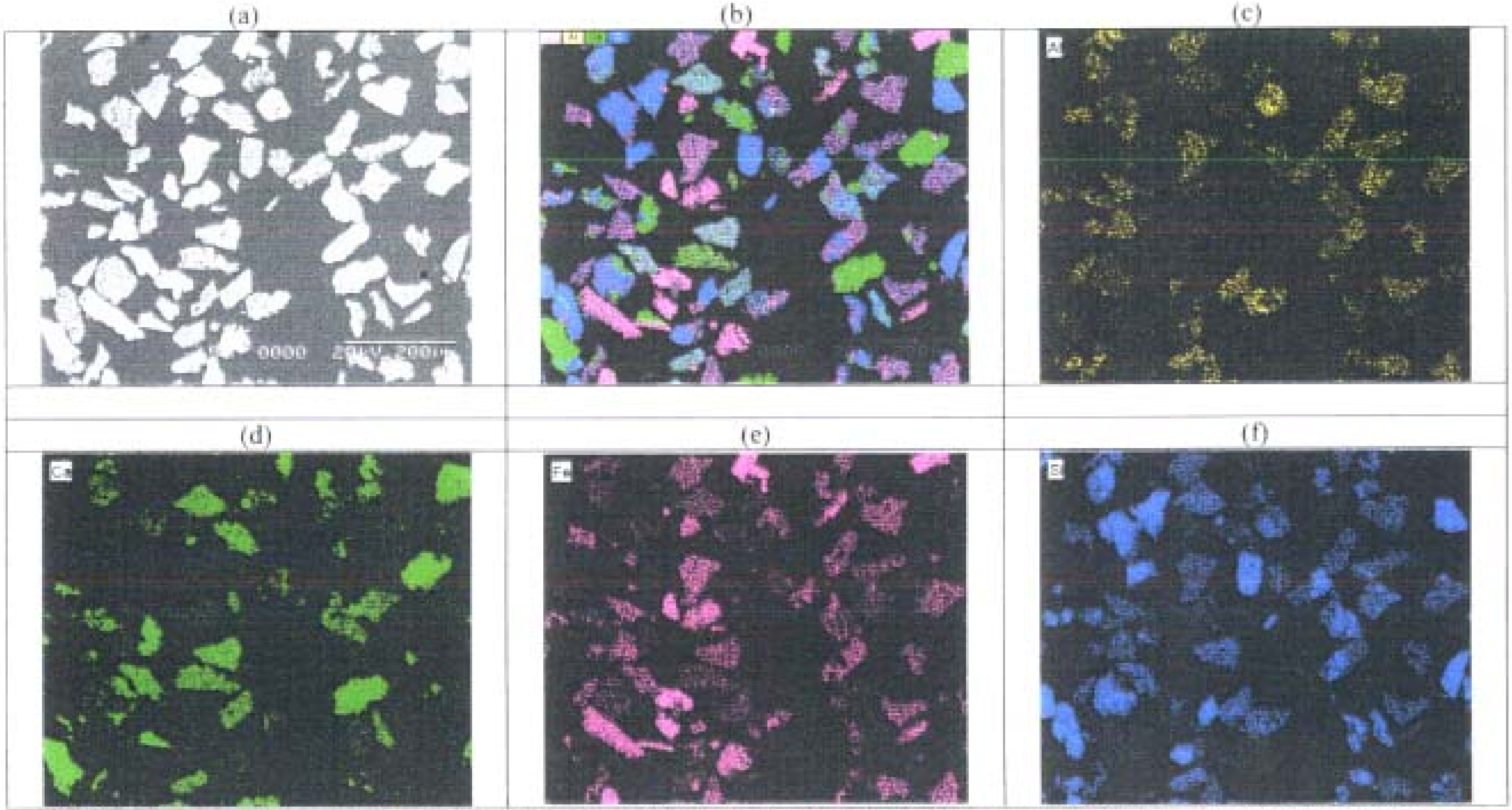
APPENDIX II-B2 – SEM analysis of the heavy fraction from the +270# fraction of composite 3


(a) (b) (c) (d)

(e) (f) (g) (h)

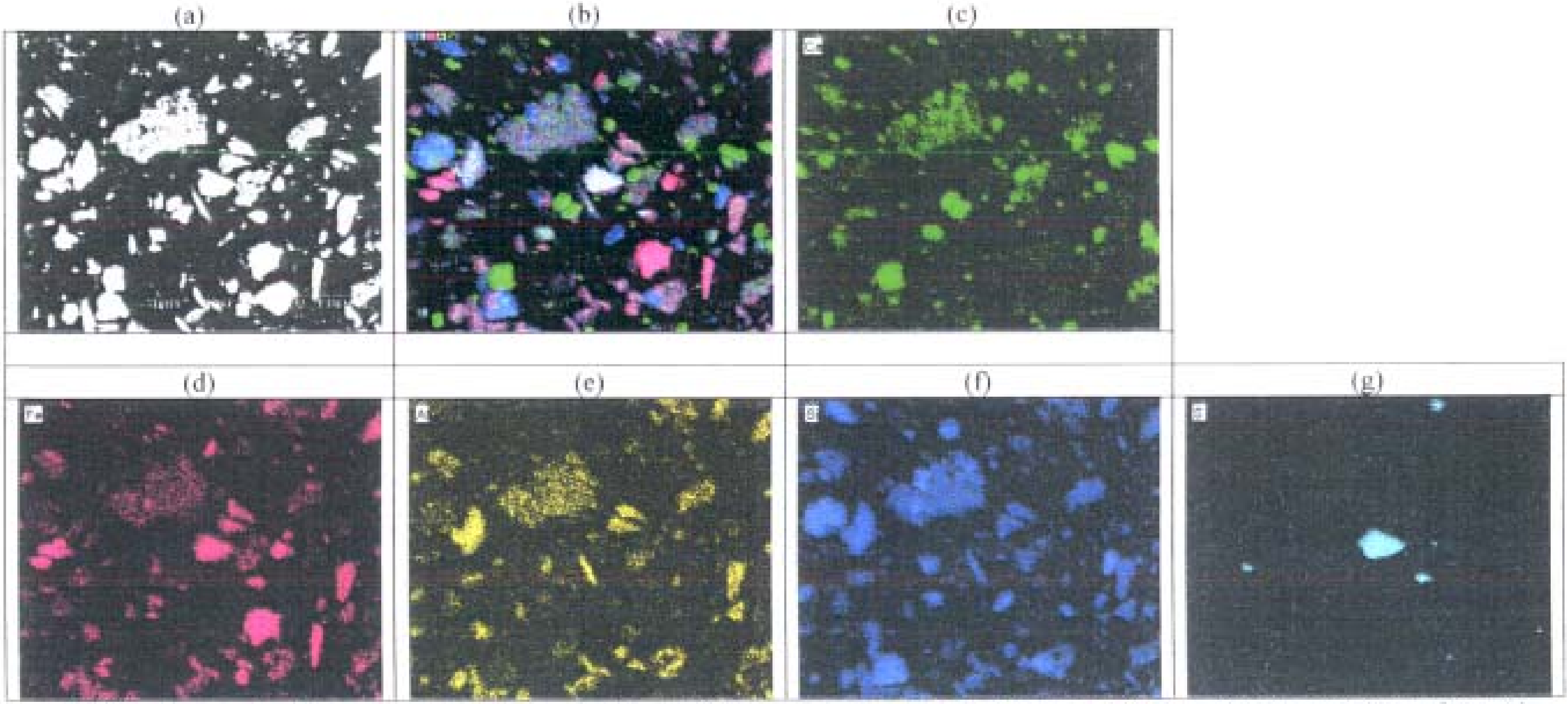
(a) SEM micrograph of the heavy fraction from the +270# fraction of composite 3; (b) combination of x-ray mappings of Al, Ca, Fe, Mg, S and Si; (c) Ca x-ray mapping; (d) Al x-ray mapping; (e) Fe x-ray mapping; (f) Mg x-ray mapping; (g) S x-ray mapping and (h) Si x-ray mapping

APPENDIX II-C – SEM analysis of the magnetic concentrate from the +270# fraction of composite 3.



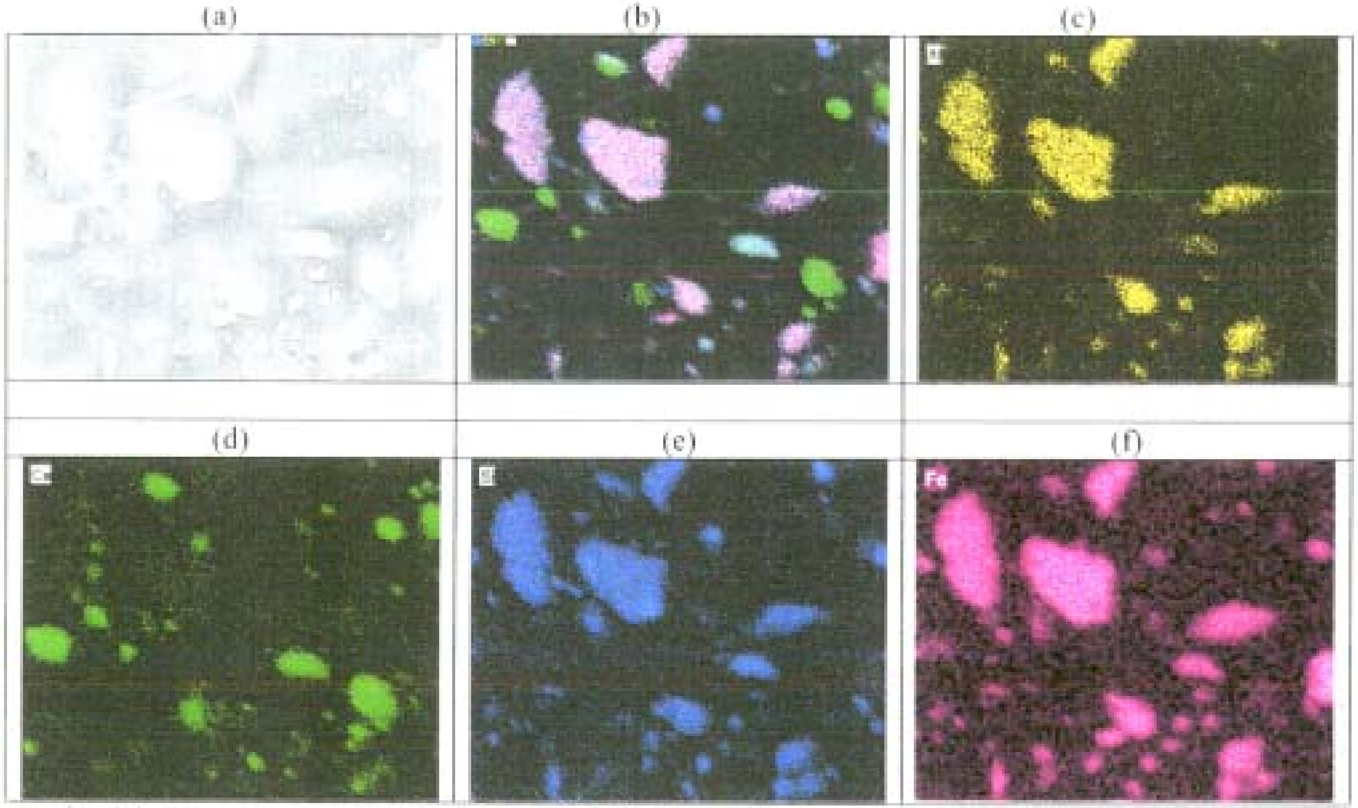
(a) SEM micrograph of magnetic concentrate from the +270# fraction of composite 3; (b) combination of x-ray mappings of Al, Ca, Fe and Si; (c) Al x-ray mapping; (d) Ca x-ray mapping; (e) Fe x-ray mapping and (f) Si x-ray mapping.

APPENDIX II-D1 – SEM analysis of the magnetic concentrate of the -400# fraction of composite 3



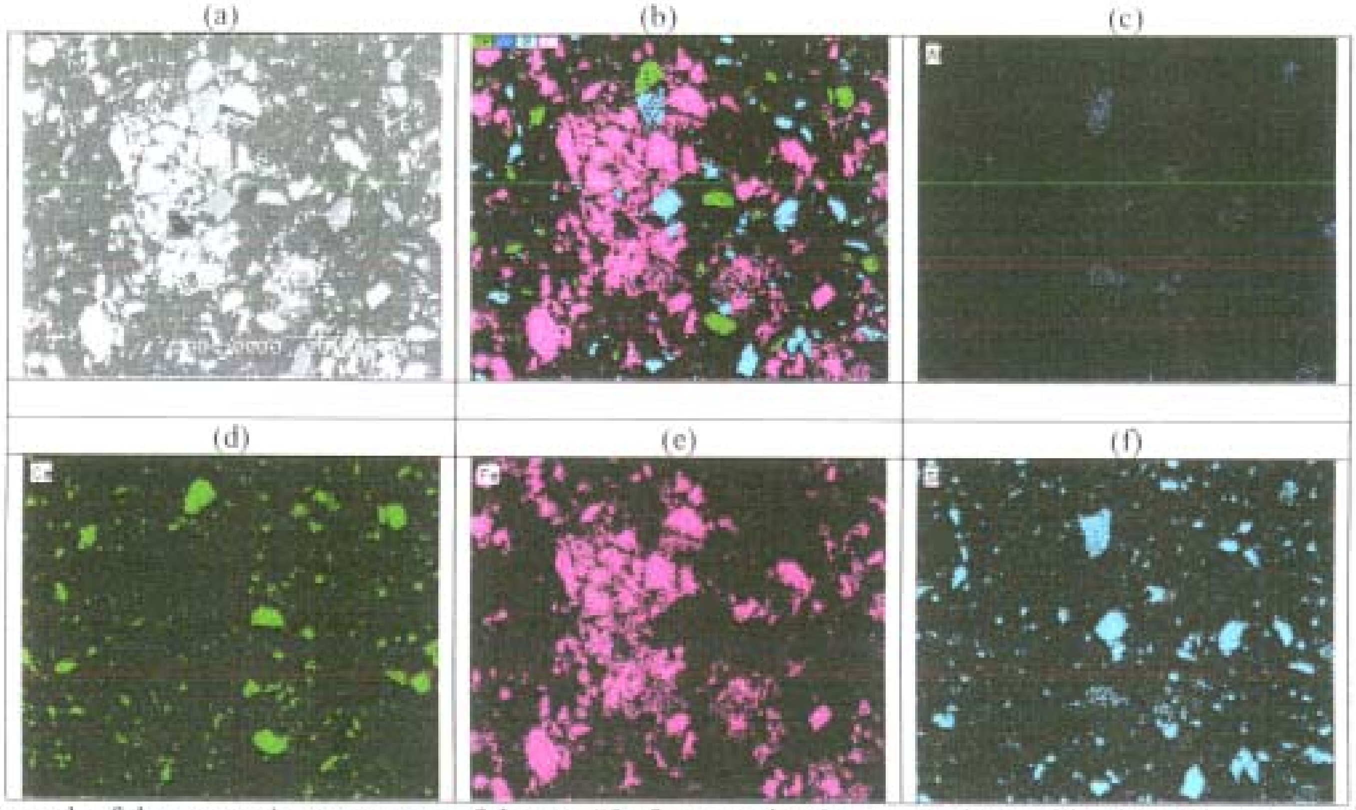
(a) SEM micrograph of the magnetic concentrate of the -400# fraction of composite 3; (b) combination of x-ray mappings of Ca, Al, Fe, Si and S; (c) Ca x-ray mapping; (d) Fe x-ray mapping; (e) Al x-ray mapping; (f) Si x-ray mapping and (g) S x-ray mapping.

APPENDIX II-D2 – SEM analysis of the magnetic concentrate of the -400# fraction of composite 3



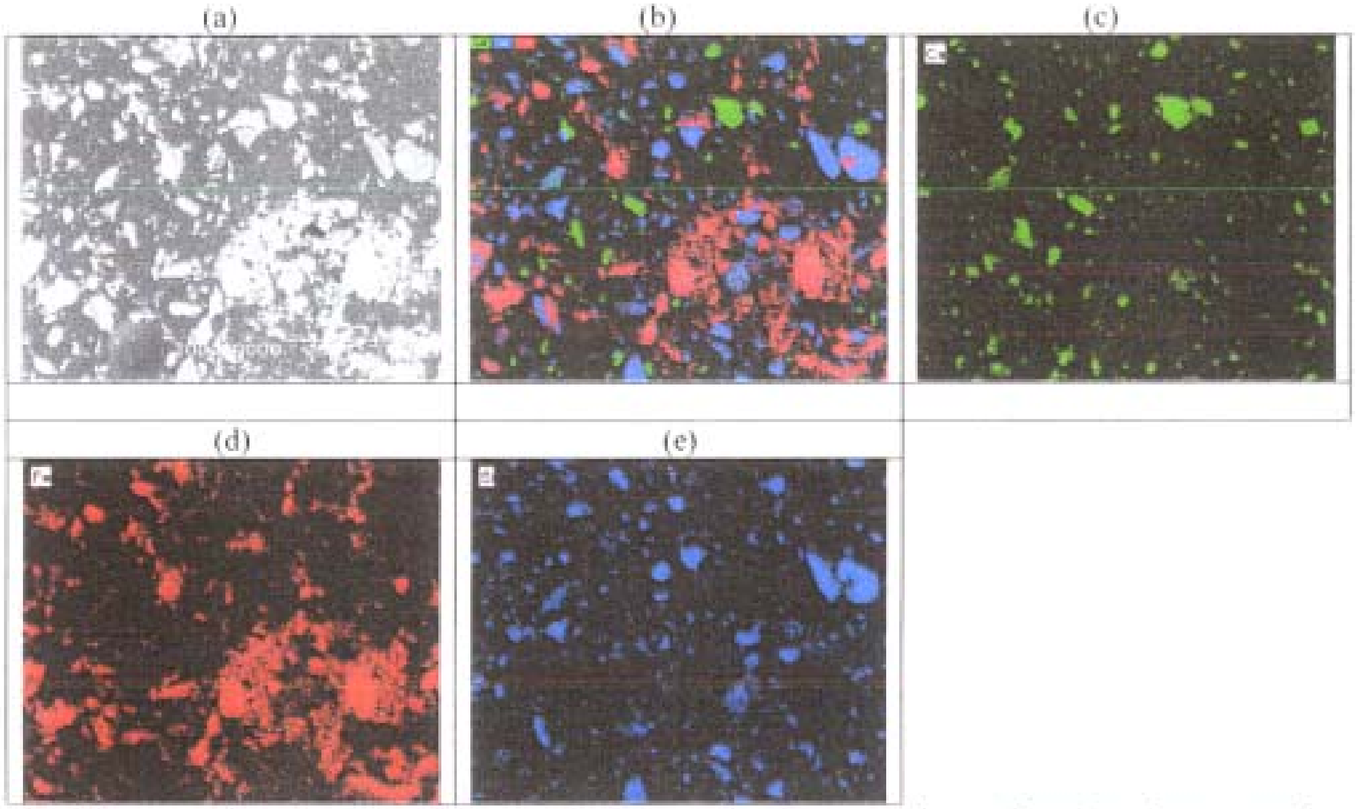
(a) SEM micrograph of the magnetic concentrate of the -400# fraction of composite 3; (b) combination of x-ray mappings of Ca, Al, , Fe, Si and S; (c) Al x-ray mapping; (d) Ca x-ray mapping; (e) Si x-ray mapping and (f) Fe x-ray mapping.

APPENDIX II-E1 – SEM analysis of the magnetic concentrate of the test #2 after one cleaning stage



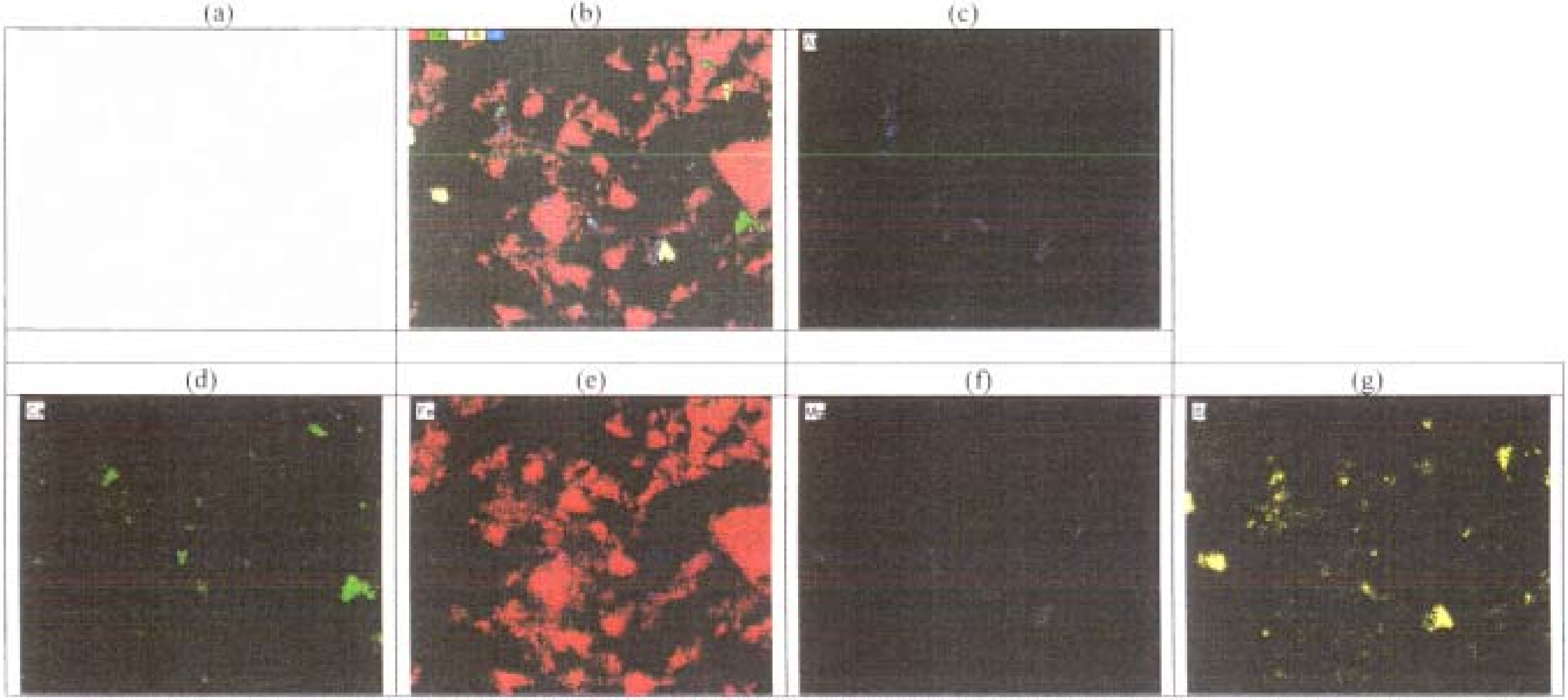
(a) SEM micrograph of the magnetic concentrate of the test #2 after one cleaning stage; (b) combination of x-ray mappings of Al, Ca, Fe and Si; (c) Al x-ray mapping; (d) Ca x-ray mapping; (e) Fe x-ray mapping and (f) Si x-ray mapping.

APPENDIX II-E2 – SEM analysis of the magnetic concentrate of the test #2 after one cleaning stage



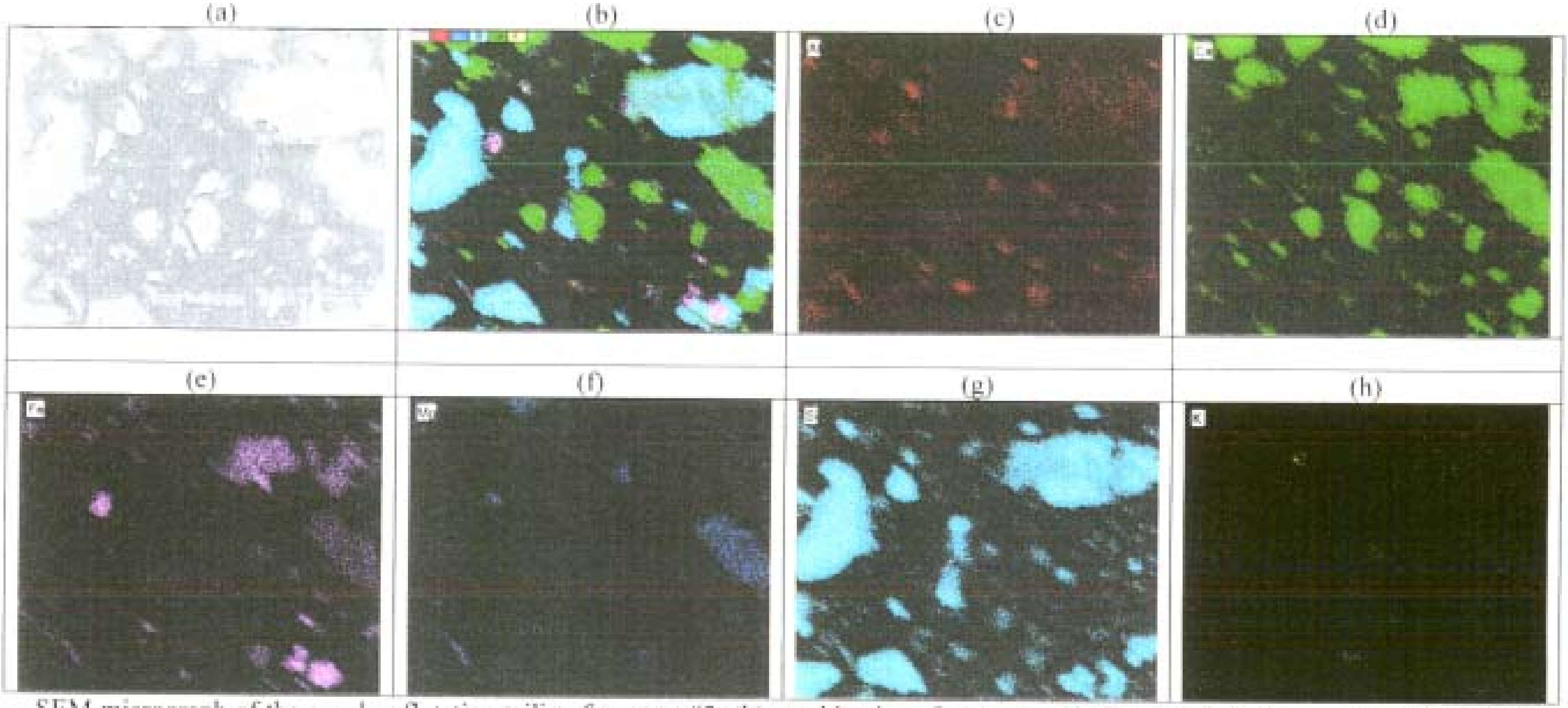
SEM micrograph of the magnetic concentrate of the test #2 after one cleaning stage; (b) combination of x-ray mappings of Al, Ca, Fe and Si; (c) Ca x-ray mapping; (d) Fe x-ray mapping and (e) Si x-ray mapping.

APPENDIX II-F – SEM analysis of the magnetite concentrate of the test #2 after four cleaning stages



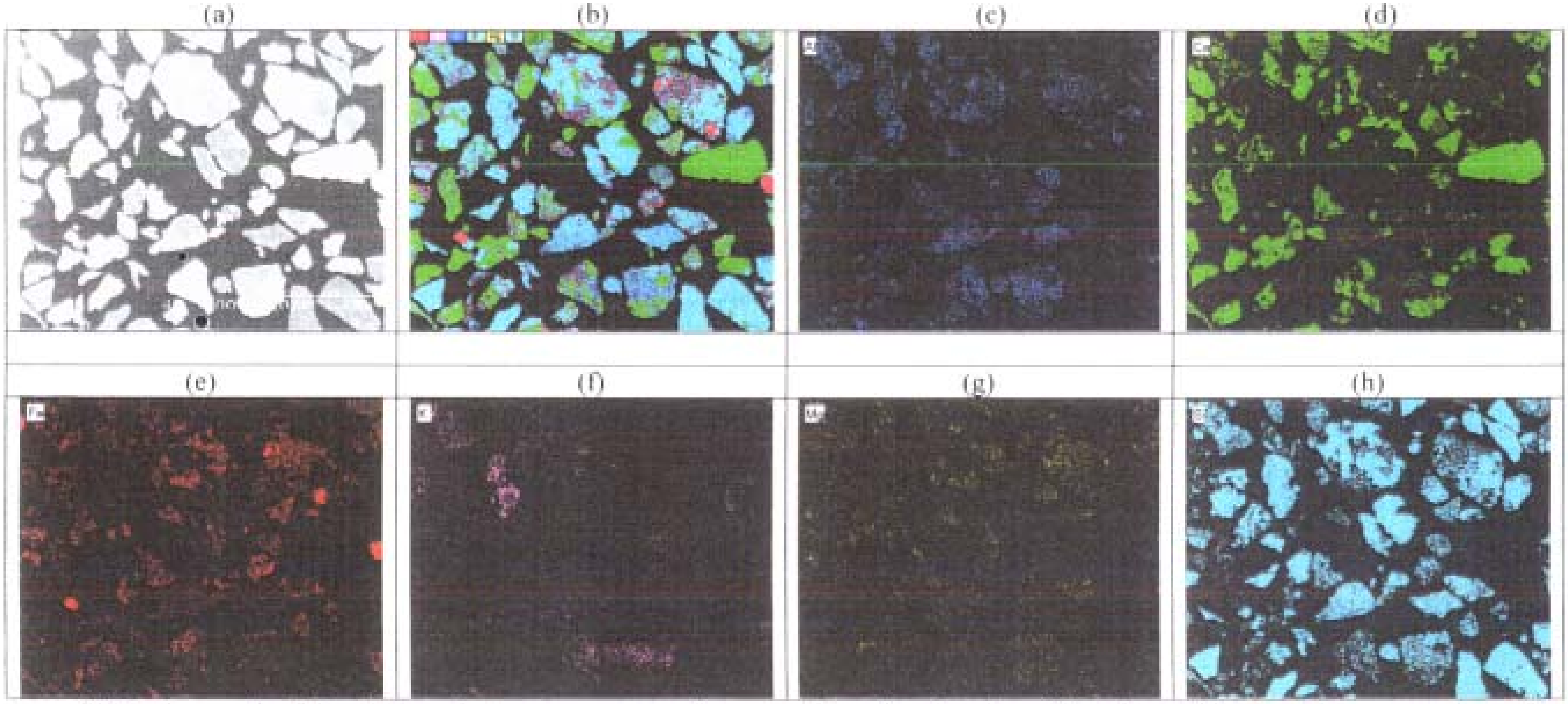
(a) SEM micrograph of the magnetite concentrate of the test #2 after four cleaning stages; (b) combination of x-ray mappings of Al, Ca, Fe, Mg and Si; (c) Al x-ray mapping; (d) Ca x-ray mapping; (e) Fe x-ray mapping; (f) Mg x-ray mapping and (g) Si x-ray mapping.

APPENDIX II-G – SEM analysis of the rougher flotation tailing from test #2



SEM micrograph of the rougher flotation tailing from test #2; (b) combination of x-ray mappings of Al, Ca, Fe, Mg, Si and K; (c) Al x-ray mapping; (d) Ca x-ray mapping; (e) Fe x-ray mapping; (f) Mg x-ray mapping; (g) Si x-ray mapping and (h) K x-ray mapping.

APPENDIX II-H – SEM analysis of the +100 mesh garnet concentrate from test #3 rougher flotation tailing



SEM micrograph of the +100 mesh garnet concentrate from test #3 rougher flotation tailing; (b) combination of x-ray mappings of Al, Ca, Fe, Mg, S and Si; (c) Ca x-ray mapping; (d) Al x-ray mapping; (e) Fe x-ray mapping; (f) Mg x-ray mapping; (g) S x-ray mapping and (h) Si x-ray mapping.

APPENDIX II-11 – SEM analysis of the cleaner flotation concentrate from test #3 (2^{nd} cleaner)

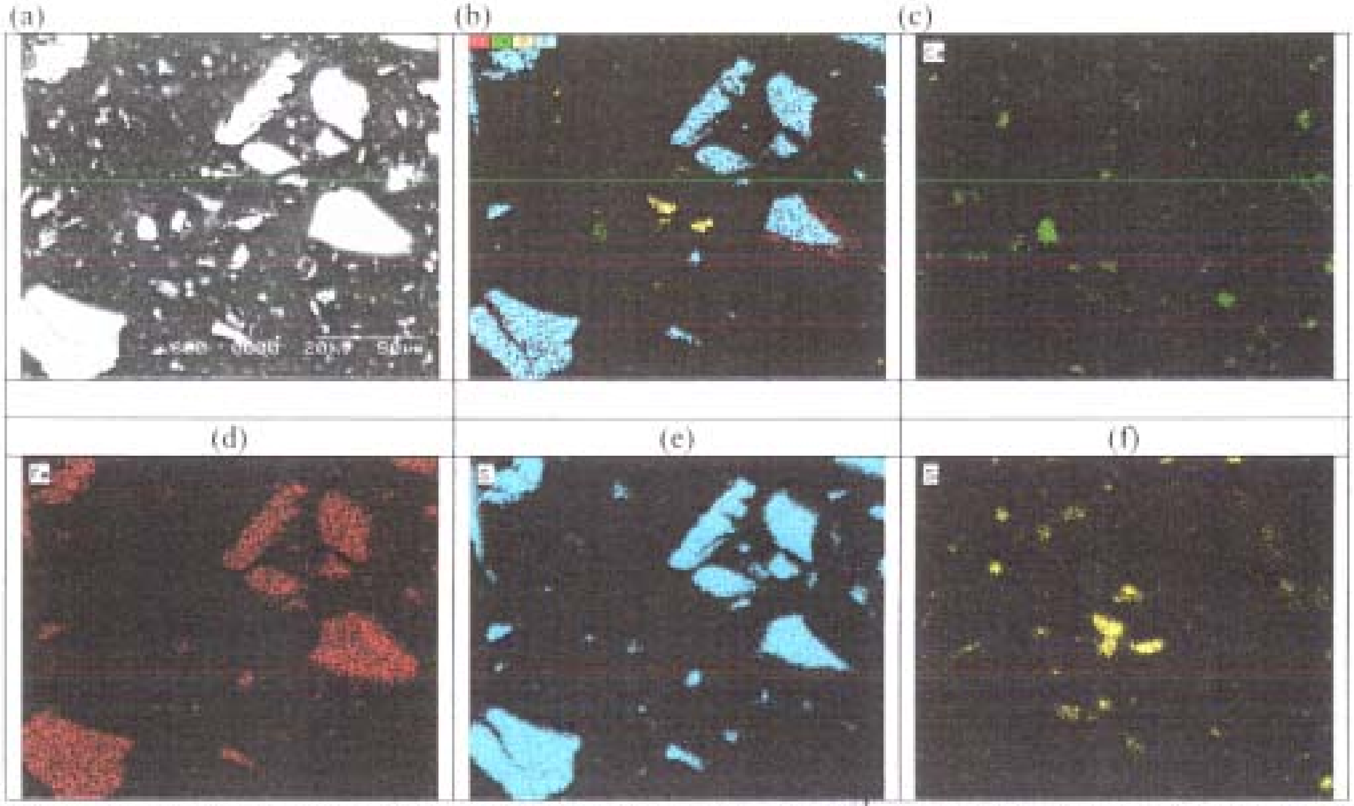
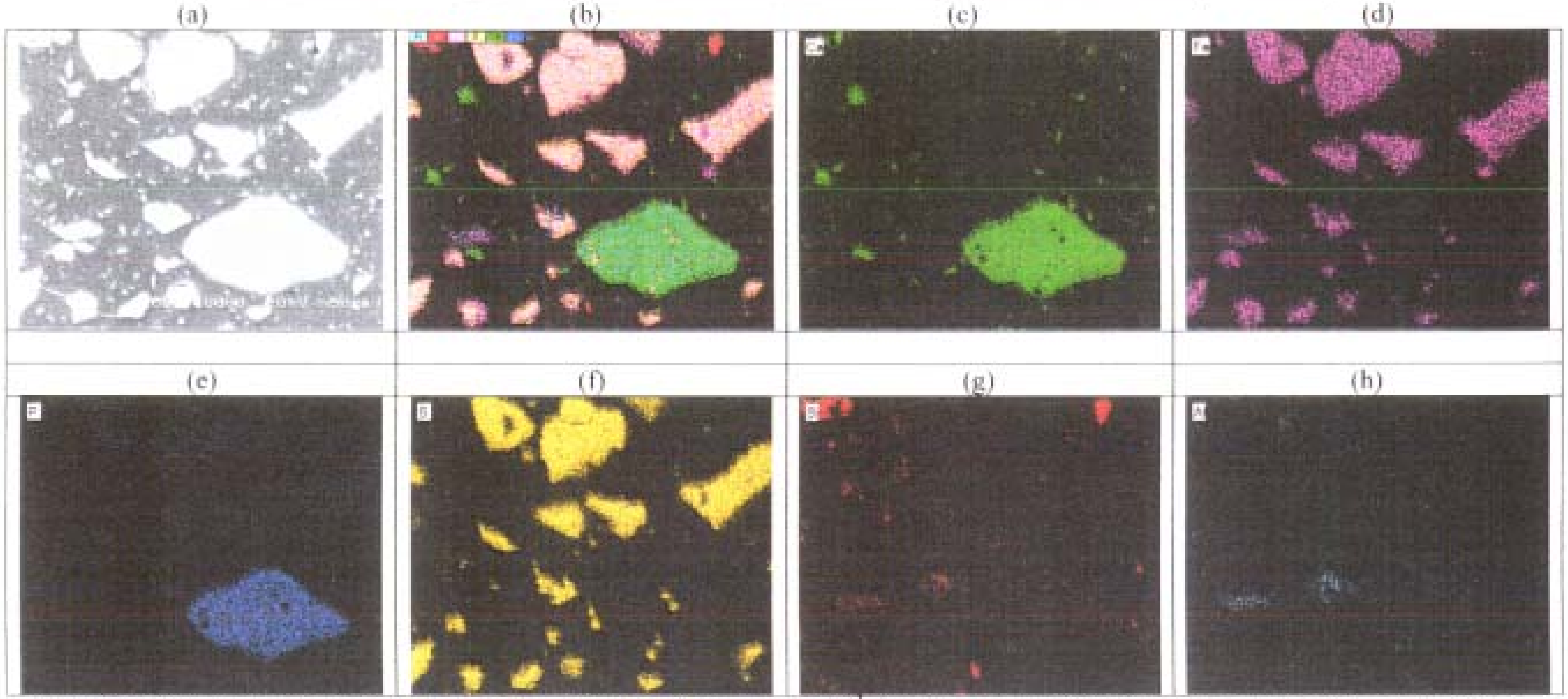


Figure : (a) SEM micrograph of the cleaner flotation concentrate from test #3 (2^{nd} cleaner); (b) combination of x-ray mappings of Ca, Fe, S and Si; (c) Ca x-ray mapping; (d) Fe x-ray mapping; (e) S x-ray mapping and (g) Si x-ray mapping.

APPENDIX II-12 – SEM analysis of the cleaner flotation concentrate from test #3 (2^{nd} cleaner)



(a) SEM micrograph of the cleaner flotation concentrate from test #3 (2^{nd} cleaner); (b) combination of x-ray mappings of Al, Ca, Fe, P, S and Si; (c) Ca x-ray mapping; (d) Fe x-ray mapping; (e) P x-ray mapping,; (f) S x-ray mapping; (g) Si x-ray mapping and (h) Al x-ray mapping.

APPENDIX III – Design Criteria

1. Rougher Ball Mill

The purpose of the rougher ball mill is to activate the sulfide surfaces of the feed and improve magnetite liberation. This was modeled on a bench-scale by using a three minute grind, which gave us a product size P80 of 139 microns. In sizing a ball mill, it is key to determine the power necessary to grind the feed to the size required.

$$W = 13.4 \cdot W_i \left(\frac{1}{\sqrt{P_{80}}} - \frac{1}{\sqrt{F_{80}}} \right)$$

Wi = 14.05 kWh/tonne, P80 = 139 microns, F80 = 235 microns.

W = 3.69 hp/tph * 300 tph = 1110 hp = Power (P)

The density and the dimensions also can determine power, so one can back-calculate the mill size through the power needed (Nordberg Method)[3].

$$P = \left(\frac{\rho}{315} \right) \cdot A \cdot B \cdot C \cdot L$$

ρ = 3.1 g/cm3,
A = $D^{2.5}$/5.6642, D = mill diameter,

Mill type and loading factor for wet overflow mill at 36% loading, B = 4.84
Speed factor at 75% of critical speed, C = 0.1838
Length = L

Using a length to diameter ratio of 1.5, the mill length is calculated as 6.1 m and the mill diameter is 4.0 m, leading to a power draw of 1180 hp.

2. Rougher Wet Drum Magnetic Separator

WDMS was selected on the basis of the flow rate per magnet width, so the tonnage rate has to be converted into flow rate. With the low grade of magnetite in the feed, the load per foot of magnet is not an issue, as it is with dense media separation in coal operations.

The highest pulp density recommended is 30% solids, so the flow can be determined as follows.

300 tonnes/h, at 3.1 s.g., and 30% solids
300tph x 70%Water / 30%Solids + 300tph / 3.1s.g. = 799 m^3/h
799 m^3/h = 3508 USGPM

Using a 36-inch diameter, 750 Gauss Double Drum Eriez WDMS, the flow per magnet width at 30% solids is 50 USGPM/ft, as determined from Eriez manual. The manual recommends a double drum WDMS if the feed has not been deslimed and the solids content is high.

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[3] Weiss,N. (Ed) *SME Mineral Processing Handbook*, SME-AIME, Littleton, CO, 1986

3508 USGPM / 50 USGPM/ft = 70.15 ft of magnet. Using the largest width WDMS available, at 144 inches width, this leads to:
70.15 ft/ 12 ft = 5.85 WDMS, or 6

Thus we recommend six 91.4 cm diameter, 750 Gauss Double drum separators.

3. Rougher Flotation Cells

Flotation cells can be sized using the equation NV = Q*T*E*X/24
where N = number of cells, V = effective cell volume,
Q = dry ore throughput, tons/24 hr, T = retention time,
E = pulp expansion factor due to aeration, and X = pulp flow rate.

Using Test 3 results, where 91.8% of the feed goes to flotation, thus:

Q = 300 tonnes/h * 24 hr * 91.8% = 7303 tons/day (imperial)

Scaling up from the bench test, T = 9 min * 2.1 = 18.9 min

E = 1/0.85 = 1.18, with 15% air.

X can be determined through the following equation, where P% is the pulp density at 35% and s.g. is 3.1 g/cm^3.

$$X = 0.5338 \cdot \left(\frac{1}{\text{s.g.}} + \frac{100}{P\%} - 1\right), \quad X = 1.16$$

Thus, NV = 7303tpd * 18.9min *1.18 * 1.16/24hr = 7873 cubic ft.

Most rougher cell banks have from 14 to 17 cells, so the size ranges could be as shown in Table 18:

Table 18: Size ranges of the rougher cell banks

Number of Cells	14	15	16	17
Volume of Cells (cubic meter)	15.9	14.86	13.9	13.11
Volume of Cells (cubic meter)	15.1	14.87	14	13.14

Fourteen cells take up the least amount of space, thus it is recommended a rougher bank of fourteen 15.1 cubic meter mechanical flotation cells.

Note that the pulp density for the rougher flotation is 35%, where the WDMS was at it's maximum of 30%. This would need either running the flotation cells at a slightly lower pulp density, or having a simple dewatering stage before the flotation cells.

Personnel

Dr. Sonia Veiga (metallurgical engineer), Research Associate, Dr. Bern Klein (mineral processing engineer), Dr. Marcello Veiga (metallurgical engineer) and Dr. Scott Dunbar (geophysical engineer), Assistant Professors conducted the work. Dr. S. Veiga has the expertise in materials engineering, Dr. B. Klein in mineral processing, Dr. M. Veiga in process mineralogy and Dr. S. Dunbar, P. Eng., in mineral economic, valuation.

COST STATEMENT

Recovery of Magnetite from Phoenix Tailings

Professional Services		**$4,000.00**
Site visit, sample collection and shipping		**$500.00**
Analysis (SEM, XRD, ICP)		**$1,000.00**
Final report and office expenses		**$500.00**
	TOTAL	**$6,000.00**

The University of British Columbia
CIMI - Center for Industrial Minerals Innovations
www.cimi.mining.ubc E-mail: cimi@mining.ubc.ca
6350 Stores Rd., Vancouver, BC, Canada, V6T 1Z4
ph: (604) 8223986, fax: (604) 8225599

INVOICE **October 2, 2002**

TO:

Mr. George Stewart
Kettle River Resources
Box 130 – 298 Greenwood St
Greenwood, B.C. V0H 1J0
Fax: 250 - 4452259

Re: Recovery of Magnetite from Phoenix Tailings

Professional Services		$4,000.00
Site visit, sample collection and shipping		$500.00
Analysis (SEM, XRD, ICP)		$1000.00
Final report + office expenses		$500.00
	Total	$6,000.00
G.S.T. (7%)		$420.00
	Total	**$6,420.00**
Advance Payment (Sept 11, 2001)		**$3,210.00**
	Balance Due	**$3,210.00**

Bern Kle

Dr. Bern Klein
Department of Mining & Mineral Process Engineering
517D - 6350 Stores Road, Vancouver, BC, V6T 1Z4

Make cheque payable to: Department of Mining & Mineral Process Engineering

The University of British Columbia
CIMI – Center for Industrial Mineral Innovations
6350 Stores Rd., Vancouver, BC, Canada, V6T 1Z4
ph: (604) 8223986, fax: (604) 8225599

August 7, 2001

Kettle River Resources
Box 130 - 298 Greenwood Street
Greenwood, B.C., V0H 1J0
Fax: (250) 445-2259

Attention: Mr. George Stewart, President

Dear Mr. Stewart:

Re: Recovery of Magnetite from Phoenix Tailings

As discussed during our telephone conversation on August 1, the following is a proposal for a study to assess the recovery and usage of magnetite from the Phoenix tailings for dense media separation. The study would be conducted at the University of British Columbia's Center for Industrial Mineral Innovations. The study will be divided into the following sections:

1. Review existing reports about the Phoenix tailings.
2. Survey site and collect sample for testwork.
3. Determine suitability of magnetite for dense media separation.
4. Conduct a market study of BC, Alberta and Washington coal operations.
5. Conduct metallurgical testing to develop a magnetite recovery process
6. Produce magnetite dense media product for characterization.
7. Complete order of magnitude capital and operating cost estimate for processing facility and cash flow analysis

The study will be conducted by Mr Alireza Entesari, who is a MASc student in the Department of Mining and Mineral Process Engineering. Professor Bern Klein will supervise the work and assist with the preparation of the final report. The study will take three months to complete, plus one month to prepare a final report. The work will be conducted on a fee for service basis with a breakdown as follows:

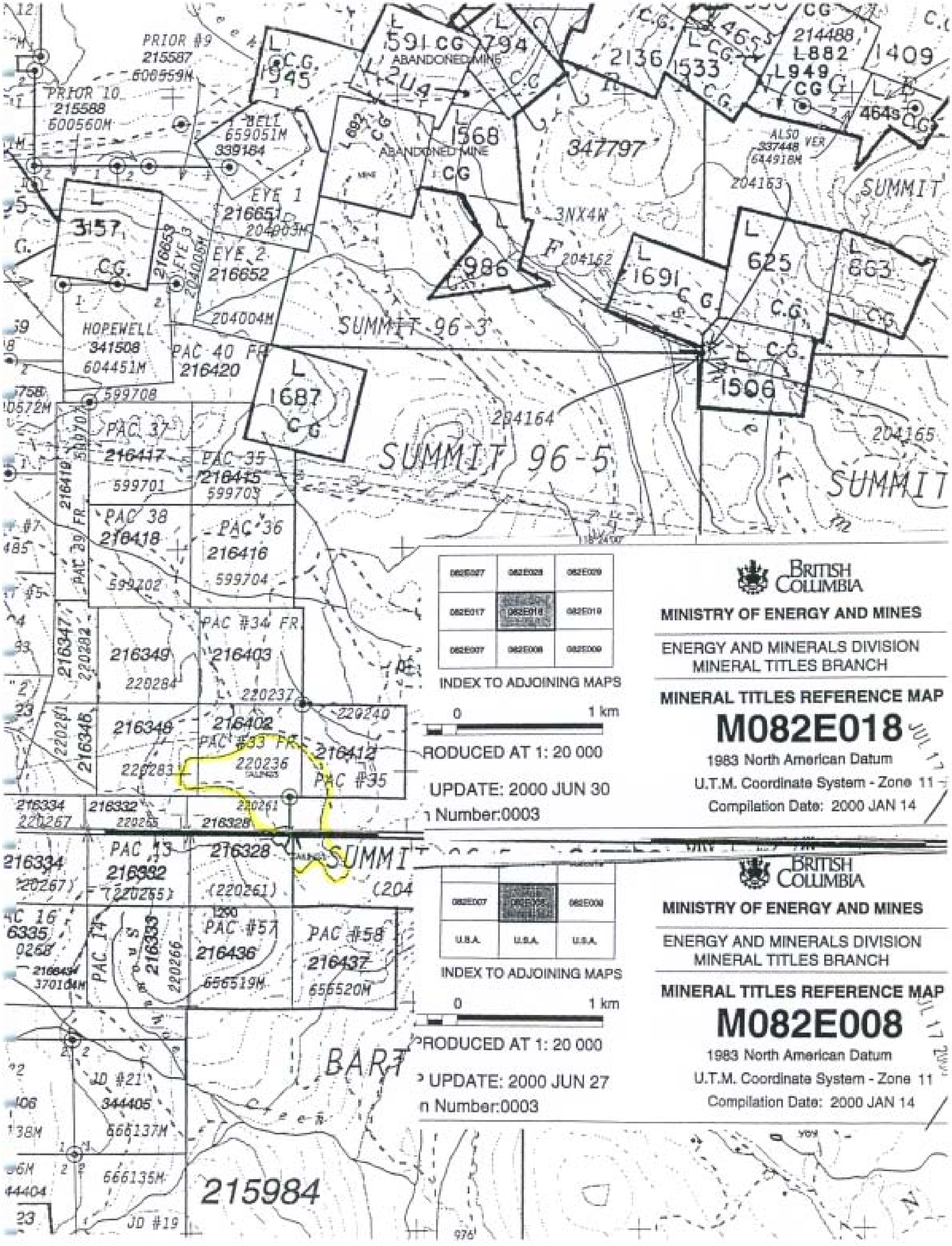
1. Student time - 4 months @ $1,000.00 per month	$4,000.00
2. Site visit, sample collection and shipping	$500.00
3. Miscellaneous analyses (assays, SEM time, XRD)	$1,000.00
4. Preparation of final report	$500.00
Total	$6,000.00

Please feel free to contact me at any time to discuss the details of this proposal.

Yours very truly,

Bern Klein

Bern Klein, Ph.D.
Assistant Professor



BRITISH COLUMBIA

MINISTRY OF ENERGY AND MINES

ENERGY AND MINERALS DIVISION
MINERAL TITLES BRANCH

MINERAL TITLES REFERENCE MAP
M082E018

1983 North American Datum
U.T.M. Coordinate System - Zone 11
Compilation Date: 2000 JAN 14

INDEX TO ADJOINING MAPS

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082E017	082E018	082E019
082E007	082E008	082E009

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BRITISH COLUMBIA

MINISTRY OF ENERGY AND MINES

ENERGY AND MINERALS DIVISION
MINERAL TITLES BRANCH

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M082E008

1983 North American Datum
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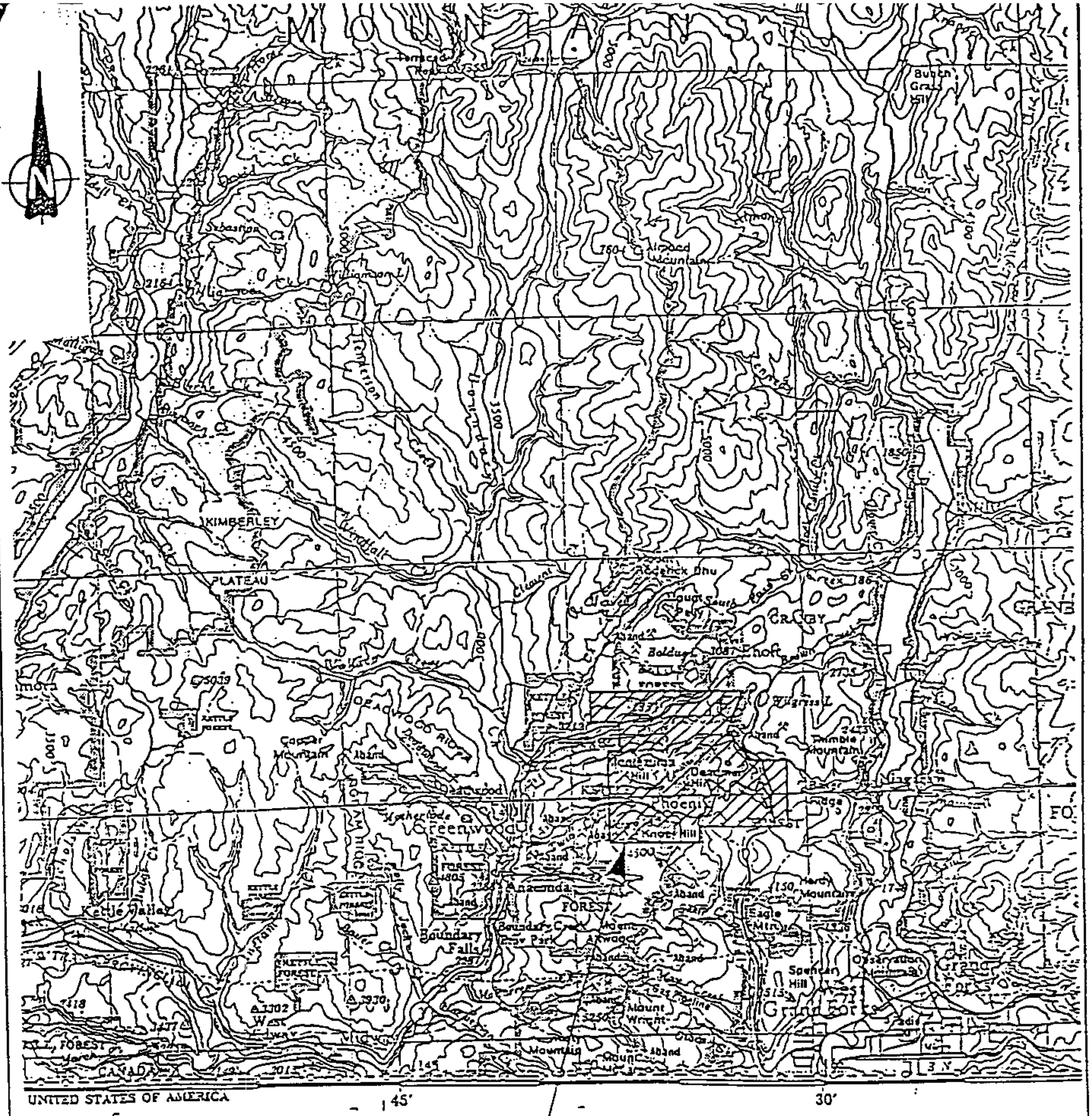


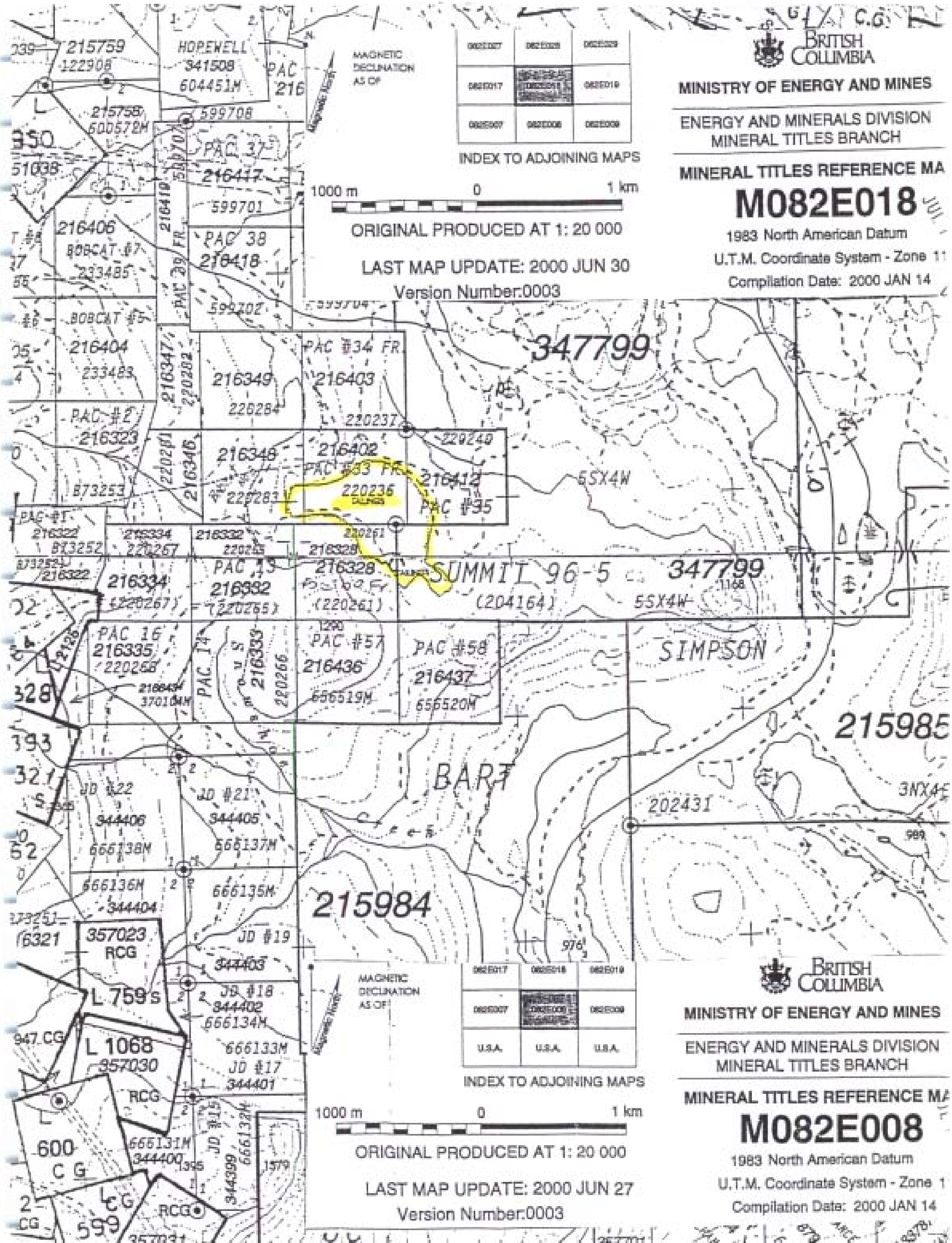
PHOENIX

KETTLE RIVER RESOURCES LTD.

PHOENIX PROJECT

LOCATION MAP

SCALE 1:250,000	Date:April,1991
NTS-82E/2	FIG. 1



INDEX TO ADJOINING MAPS

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MINISTRY OF ENERGY AND MINES

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1983 North American Datum

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Compilation Date: 2000 JAN 14

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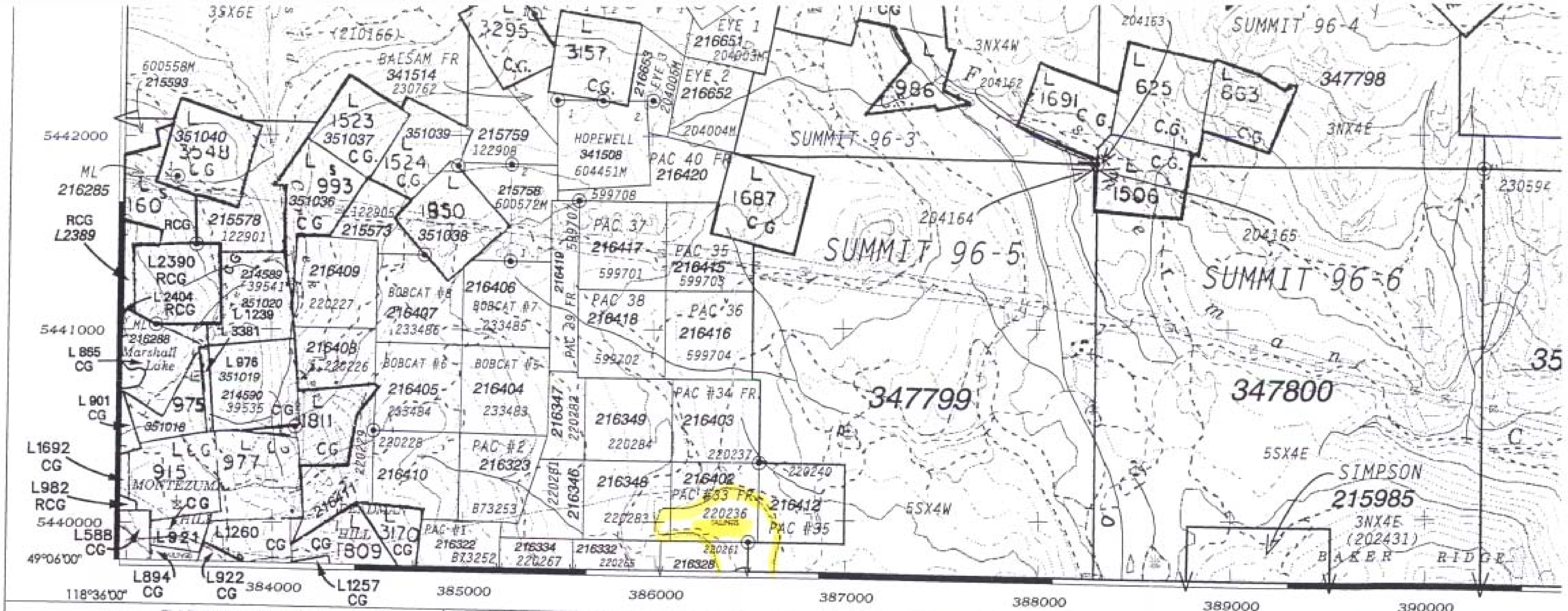

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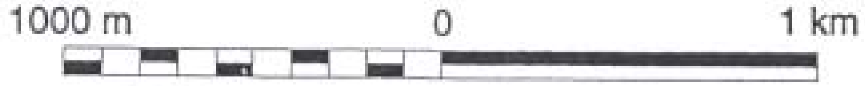

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MINERAL TITLES REFERENCE MAP

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1983 North American Datum

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JUL 17 2000

MMISSIONER OFFICES

AST / LIARD
365 Hornby Street
ouver BC V6Z 2G3
c Query: (604) 660-2672
(604) 660-2653
g Divisions: Atlin, Clinton,
Lillooet, Skeena, New
minster, and Vancouver

ILOOPS / OKANAGAN
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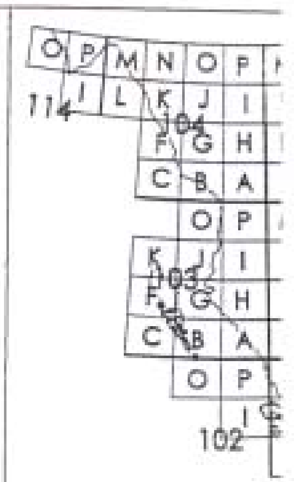
oops BC V2C 6K4
Query: (250) 828-4540
250) 828-4233
Divisions: Kamloops,
, Osoyoos, Revelstoke,
ameen, and Vernon

EAST KOOTENAY
100 Cranbrook Street North
Cranbrook BC V1C 3P9
Public Query: (250) 426 1211
FAX: (250) 426-1253
Mining Divisions: Fort Steele
and Golden

KOOTENAY
310 Ward Street
Nelson BC V1L 5S4
Public Query: (250) 354-6103
FAX: (250) 354-6407
Mining Divisions: Greenwood,
Nelson, Slocan, and Trail
Creek



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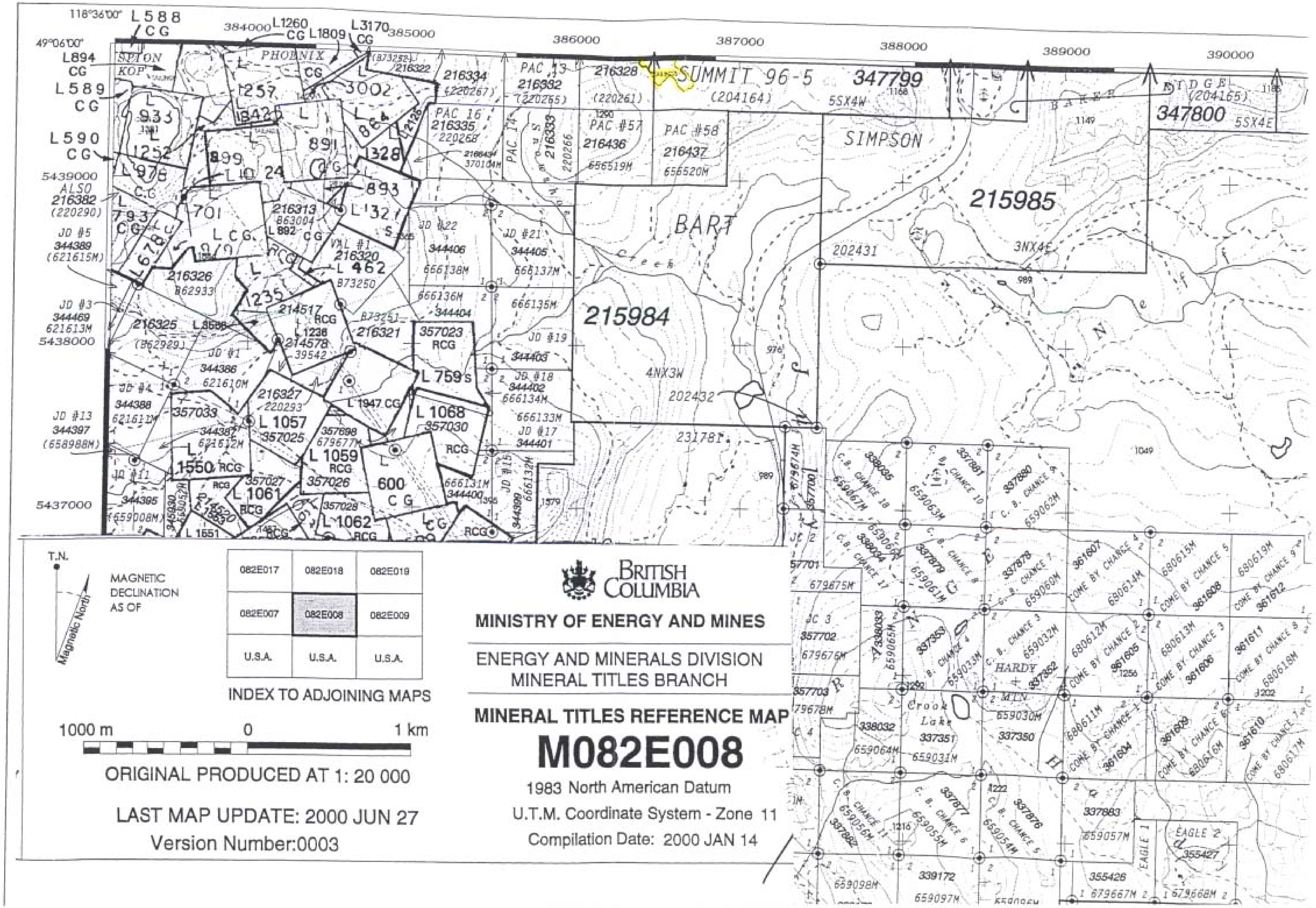

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