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COLORADO SCHOOL OF MINES RESEARCH INSTITUTE

Study on

CHARACTERIZATION OF CRANDON MILL TAILINGS

Prepared For

Exxon Minerals Company
P.O. Box 813
Rhinelander, Wisconsin 545

by

Colorado School of Mines Research
P.O. Box 112
Golden, Colorado 80402

CSMRI Project J10523

October 12, 1982

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INTRODUCTION

Exxon Minerals Company is conducting a comprehensive environmental evaluation program for a proposed project to mine and process ore from a deposit located in Forest County, Wisconsin. The proposed operation is known as the Crandon Project. As a part of the environmental evaluation program, Exxon Minerals Company contracted with the Colorado School of Mines Research Institute to conduct a study on the physical, chemical, and mineralogical characteristics of samples representing typical tailings that would result from processing the ore

The study was conducted on tailings samples furnished by Exxon Minerals Company. These samples were produced from Crandon ore in pilot-scale investigations on the concentrating process.

Elements of the study included: chemical analysis, radiological analysis, mineralogical examinations, extraction procedure testing according to U.S. Environmental Protection Agency methods, acid production potential testing, leach testing, and rheological testing. The results of the rheological testing were furnished in a previous report dated September 17, 1981 and thus are not included here.

The study focused on the characteristics of the solid mineral material component of the tailings and did not directly address the characteristics of tailings water; this aspect is being determined by Exxon Minerals in separate studies on water management and treatment.

The reported work was authorized by a contractural agreement (Contract No. 21569) between Exxon Minerals Company and the Colorado School of Mines Research Institute, dated May 5, 1981.

SUMMARY

The physical, chemical, and mineralogical characteristics of Crandon mill tailings as determined in this study on sample material furnished by Exxon Minerals Company are summarized below.

Six tailings samples were studied. These samples were generated in pilot-scale investigations on the milling process and were identified as follows:

Sample No.	Sample Designation
1	Zinc Tailings Fines (massive ore)
2	Zinc Tailings Fines (mixture massive/stringer ore)
3	Combined Mine Backfill
4	Unclassified Zinc Tailings (massive ore)
5	Pyrite Concentrate Fines (mixture massive/stringer ore)
6	Pyrite Tailings Fines (mixture massive/stringer ore)

CHEMICAL PROPERTIES

With the exception of the pyrite tailings fines, the material studied can be characterized as high sulfide tailings. The results of chemical analysis indicate that the tailings samples are composed primarily of iron sulfide (FeS_2), silica (SiO_2), and silicates (Al_2O_3, MgO, K_2O, Na_2O, SiO_2); along with minor amounts of copper, lead, and zinc sulfides (CuS, PbS, and ZnS), calcium/magnesium carbonate ($CaCO_3$, $MgCO_3$), and iron-arsenic sulfide ($FeAsS$); with trace amounts of chromium, gold, manganese, mercury, nickel, selenium, silver, titanium, chloride, fluoride, and phosphorus present in crystal lattices or as compounds. Also, indicated is the presence of cyanide adhering to tailings particles in trace amounts.

The sulfur content of tailings samples was in the range of 22% to 41%, except for the pyrite tailings fines which had a sulfur content of less than 2%.

Comparison of the carbonate content of the samples (less than 2% for all six) with the sulfur contents indicates a deficiency of basic material for neutralization of potential acid.

Heavy metal concentrations in the tailings samples are indicated to be higher than average or typical values reported in the literature for igneous rocks and soils.

RADIOLOGICAL PROPERTIES

Radiological analysis results on radium and thorium isotopes indicate a low level of radioactivity in the tailings samples. Comparison of the analysis results with information given in the literature indicates that the radioactivity level of the tailings samples is within the range usually found for surface soils.

MINERALOGICAL PROPERTIES

X-ray diffraction analysis and optical microscopy examinations indicate that the mineral composition of the tailings samples is primarily pyrite-FeS_2 and quartz-SiO_2; with minor amounts of chlorite - Mg, Fe, Al $(S_4O_{10})(OH)_8$, Mica-$KAl_2(AlSi_3O_{10})(OH)_2$, and spalerite - ZnS; and trace amounts of dolomite - $CaMg(CO_3)_2$, chalcopyrite - $CuFeS_2$, galena - PbS, and siderite - $FeCO_3$.

Pyrrhotite - FeS, a form of iron sulfide that is usually more chemically reactive than the pyrite form, was not detected.

EPA EXTRACTION PROCEDURE RESULTS

For all six tailings samples, concentrations of specified heavy metals (arsenic, barium, cadmium, chromium, lead, mercury, selenium, and silver) in EPA extraction procedure leachates were less than the threshold hazardous waste classification levels published by the U.S. Environmental Protection Agency (Title 40, CFR 261.24, Federal Register, Vol. 45. No. 98, May 19, 1980).

ACID PRODUCING POTENTIAL

All of the tailings samples with relatively high sulfur content (22 to 41%) have a positive potential for sustained generation of sulfuric acid by oxidation of the contained sulfide minerals. The one sample with relatively low sulfur content (less than 2%), designated as pyrite tailings fines, does not appear to have this positive potential.

A literature survey on the characteristics of high sulfide tailings deficient in basic material for neutralization indicates that there is very little chance that acid generation will not occur when such materials are deposited, if conditions conducive to oxidation are present.

The literature survey also provided examples of another mechanism for acid generation - oxidation of thiosulfate ($S_2O_3^{-2}$) and polythionates ($S_nO_6^{-2}$ where n = 2, 3, 4....) formed in tailings water from sulfide tailings under alkaline conditions. Information from published studies indicates that, because of the nature of this reaction sequence, alkaline water containing polythionates leaving the high sulfide tailings as decant or as seepage may retain the potential for acidity over a considerable length of time. However, biological oxidation is reported to progress at a relatively rapid rate when the water pH becomes less than about 6.5, the water is in contact with air, and the water is warm.

LEACHING POTENTIAL AND LEACHATE COMPOSITION

Results of agitated leaching tests and column seepage tests indicate that part of the ions reporting in leachates were in solution with the tailings sample cake moisture at the initiation of testing, with the balance due to dissolving and diffusion from the tailings solids. For both agitated-leach and column-seepage testing the degree of ion solubilization was primarily a function of pH conditions. Some tailings samples experienced incipient oxidation of sulfides and acid generation, which contributed to ion concentrations in their leachates, when there was significant air contact and where conditions were conducive to oxidation.

Acidic conditions (pH 3 sulfuric acid) with agitation of the sample slurries produced the highest degree of leaching. Sulfate, iron, zinc, copper, and manganese were the primary ions leached. Leachates contained lesser concentrations of arsenic, cadmium, lead, chloride, and nitrate. Trace amounts of chromium, selenium, and silver were present in these leachates. Barium, mercury, and fluoride concentrations were below the analytical detection limits.

Leachates produced under near-neutral conditions (deionized water) with agitation of the sample slurries exhibited considerably lower ion concentrations. Sulfate, zinc, and manganese were the primary ions leached from the samples. Minor to trace amounts of iron were present in all of these leachates. Trace amounts of arsenic, cadmium, lead, and selenium were present in most of the leachates. Barium, chromium, mercury, and silver concentrations were generally below the analytical detection limits. Chloride, fluoride, and nitrate were detected at trace levels in some leachates.

Leaching tests under alkaline conditions (calcium hydroxide solution) with agitation served to show the potential for formation of soluble thiosalts and their subsequent oxidation to form sulfuric acid. This circumstance has been reported for tailings at several sulfide flotation milling operations. Tests results on three of the tailings samples indicated relatively rapid formation and oxidation of thiosalts under the test conditions.

Column seepage tests on three of the tailings samples employing alkaline, acidic, and deionized water over a 31-week period showed considerably less leaching compared to the short-term agitated tests, with output seepage pH levels remaining on the alkaline side throughout the test period for all three input water types. Significantly more sulfur species (10 to 90% more) were leached under conditions where these samples were alternately submerged under deionized water and exposed to the air compared to being constantly submerged. This indicates that oxidation of sulfides was suppressed by the water cover. Calcium, magnesium, and sulfate were the primary ions present in the slightly alkaline seepage waters. Due to the alkalinity, only minor to trace amounts of zinc, copper, iron, and lead were present.

DISCUSSION

DESCRIPTION OF STUDY

This study was designed jointly with Exxon Minerals Company to provide information on the characteristics of tailings materials that will require disposal as a result of milling ore from the Crandon deposit.

Information was obtained on six tailings samples produced in pilot-scale investigations on the milling process. These tailings samples were produced from two ore types - one described as massive ore which contains sphalerite, chalcopyrite, and galena in a pyrite matrix; and the other described as stringer ore, which is a quartz-matrix ore containing veinlets or stringers of pyrite with small amounts of calcopyrite and sphalerite. The tailings samples studied were designated as follows:

Zinc Tailings Fines (massive ore)
Zinc Tailings Fines (mixture massive/stringer ore)
Combined Mine Backfill
Unclassified Zinc Tailings (massive ore)
Pyrite Concentrate Fines (mixture massive/stringer ore)
Pyrite Tailings Fines (mixture massive/stringer ore)

The study included the following elements:

Chemical analyses
Radiological analyses
Mineralogical examinations
Extraction procedure testing according to the U.S. Environmental Protection Agency method
Acid production potential testing according to the B.C. Research method
Agitated leach testing with deionized water according to an American Society of Testing materials (ASTM) method
Literature survey on characteristics of similar tailings
Agitated leach testing using alkaline and acid solutions in parallel with the testing conducted using deionized water
Column leach testing over a period of several months with alkaline, acid, and deionized water
Rheological testing on tailings slurries to provide data for design of tailings transport facilities

SAMPLE MATERIAL

The sample materials used in this study are typical of the tailings to be produced from the proposed Crandon concentrator and stored on the surface. Ore for the concentrator would be mined from a massive sulfide deposit located in Forest County, Wisconsin. Copper and zinc sulfide concentrates along with minor amounts of lead concentrate would be produced from the ore.

Six samples of tailings produced during pilot-scale testing were furnished by Exxon Minerals Company. Shipping records and chain-of-custody forms indicate that the samples originated from the pilot-plant facilities of Lakefield Research of Canada Ltd., Lakefield, Ontario. The records show that the sample materials were shipped from Lakefield, Ontario on April 22, 1981 and received at CSMRI on May 15, 1981.

The sample material represents tailings to be produced from the two ore types in the Crandon concentrator. These two types have been designated as massive ore and stringer ore.

Sample identifications provided by Exxon Minerals Company were as follows:

Sample No.	Sample Designation
1	Zinc Tailings Fines (massive ore)
2	Zinc Tailings Fines (mixture massive/stringer ore)
3	Combined Mine Backfill
4	Unclassified Zinc Tailings (massive ore)
5	Pyrite Concentrate Fines (mixture massive/stringer ore)
6	Pyrite Tailings Fines (mixture massive/stringer ore)

Detailed information on the origin of the sample material is given in Appendix A, Chain-of-Custody Records; and sample descriptions are given in Appendix B.

Data on particle size characteristics were provided by Exxon Minerals Company and are shown in Appendix C. These data indicate a fine grind (in the range of -75 to -30 microns) except for the combined mine backfill sample which exhibited a coarser particle size range (a nominal -150 microns).

SAMPLE PREPARATION AND HANDLING

Primary samples were taken from shipping drums by first decanting the free water and then removing vertical sections of settled cake for all except the combined mine backfill sample. That sample slurry was packaged in plastic bags inside drums. The primary sample of combined mine backfill was obtained by draining water from the bags, placing the material into a pile for further draining, and then coning and quartering for a representative portion.

The primary samples were then blended and divided as required for various tasks of the study as shown in Figure 1.

The large sub-samples for rheological testing and column seepage studies were placed in sealed plastic drums while the smaller sub-samples were packaged in plastic bags and stored under refrigeration to minimize air contact and consequent oxidation of pyritic components.

Water decanted from the shipping drums when the primary samples were removed was replaced, and these drums containing unused sample material were stored at CSMRI pending return shipment to Exxon Minerals Company.

Duplicate water samples collected during the various extraction and leaching tests were preserved with appropriate reagents and stored for reference.

Data on moisture content determined for the samples when removed as settled cake from the shipping drums and after displacement washing and filtering are given in Table 1.

FIGURE 1

SAMPLE PREPARATION FLOWSHEET

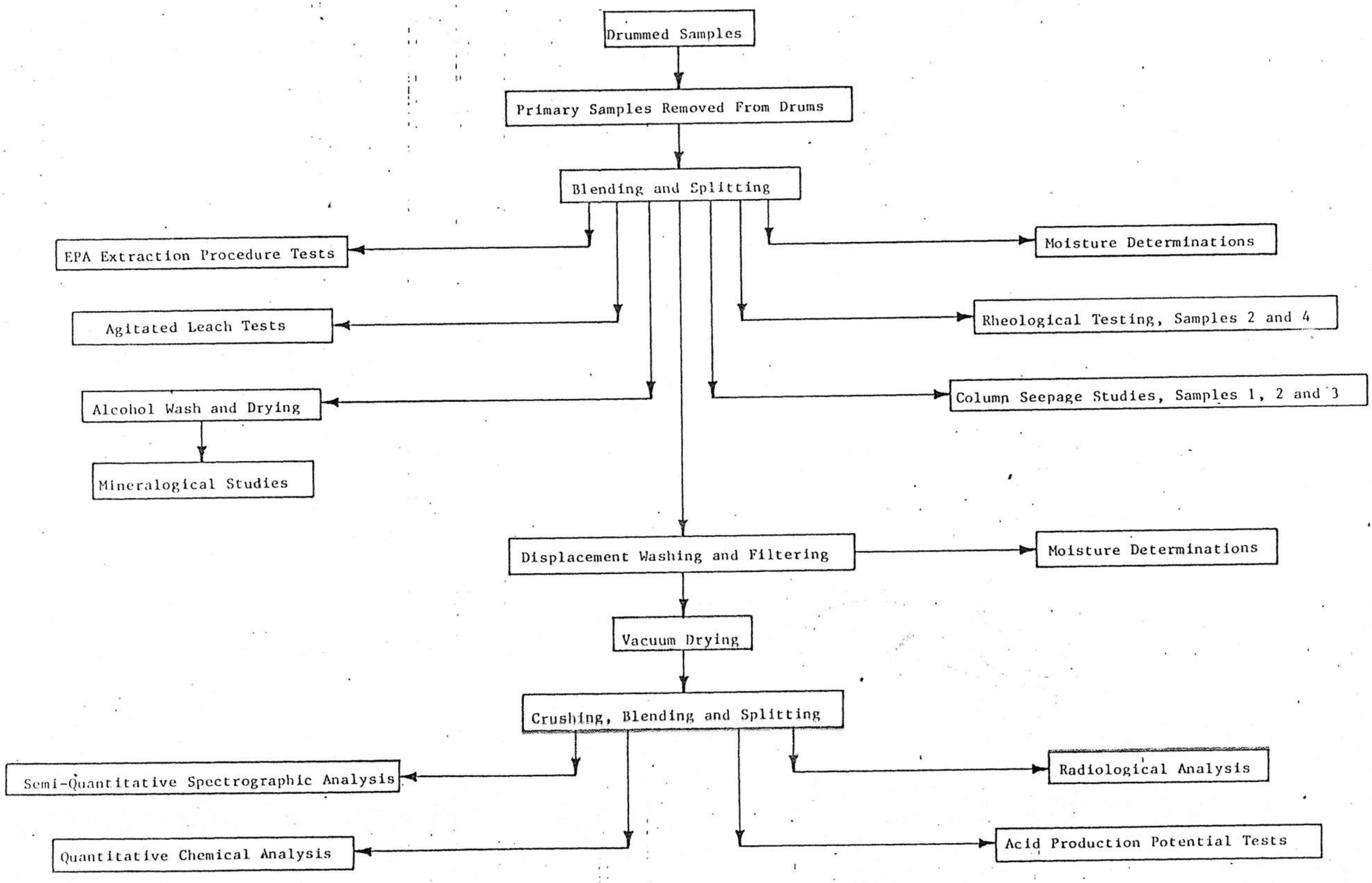

TABLE 1

MOISTURE CONTENT DATA ON TAILINGS SAMPLES

Sample No.	Description	% Moisture Settled Cake (removed from barrels)	% Moisture Filtered Cake (after washing)
1	Zinc Tailings Fines (massive ore)	11.3	18.0
2	Zinc Tailings Fines (mixture massive/stringer ore)	11.5	18.0
3	Combined Mine Backfill	16.6	9.4
4	Unclassified Zinc Tailings (massive ore)	10.8	18.6
5	Pyrite Concentrate Fines (mixture massive/stringer ore)	9.3	15.1
6	Pyrite Tailings Fines (mixture massive/stringer ore)	14.9	20.1

CHEMICAL ANALYSES

Procedure

Chemical analyses of the tailings samples were conducted in steps. First, semiquantitative emission spectrographic analyses were performed. After review of the semi-quantitative data, the list of elements and compounds for analysis given in the scope of work specified by Exxon Minerals Company was modified in consultation with the company's project representative, taking into account the spectrographic analysis data. Then quantitative analyses were performed on the six samples according to the modified list of parameters. Quantitative analytical methods were as cited in the U.S. EPA publication "Test Methods for Evaluating Solid Waste, Physical/Chemical Methods, SW-646, 1980" for those parameters included. Otherwise standard laboratory procedures were used. Data on specific gravity was included for convenience with the chemical parameters, and was obtained using air pycnometer techniques.

Results

Semiquantitative emission spectrographic analysis data on the six tailings samples are given in Appendix D. Quantitative chemical analysis data and specific gravity data are given in Appendix E.

These data indicate that the tailings samples are composed primarily of iron sulfide (FeS_2), silica (SiO_2), and silicates (Al_2O_3, MgO, K_2O, Na_2O, SiO_2); along with minor amounts of copper, lead, and zinc sulfides (CuS, PbS, ZnS), calcium-magnesium carbonate ($CaCO_3$, $MgCO_3$), and arsenic-iron sulfide ($AsFeS$); with trace amounts of chromium, gold, manganese, mercury, nickel, selenium, silver, titanium, chloride, fluoride, and phosphorus occurring either in compounds or in the crystal lattices of silicates and sulfides. The analysis data also indicates the presence of cyanide in trace amounts adhering to tailings particles even after an intensive five-stage displacement washing procedure with deionized water.

Sulfur content of the tailings samples ranged from 22 to 41%, except for the pyrite tailings fines which had a sulfur content of less than 2%.

As would be expected, the waste material from processing the massive sulfide ore of the Crandon deposit is indicated to contain several heavy metals at concentrations of potential environmental concern. Table 2 compares the range of concentrations for the metals analyzed in the tailings samples with average or typical concentration values for these in igneous rocks and soils as reported in the literature. These data indicate that, generally, the heavy metal concentrations in the tailings samples are higher than the concentrations usually encountered in igneous rock or in soils and thus provide a rationale for considering heavy metals content an item of potential environmental concern.

RADIOLOGICAL ANALYSES

Procedure

Radiological analyses for radium 226, radium 228, and thorium 230 were performed on the six tailings samples using standard counting techniques. Thorium 232 was estimated from the chemical analysis for total thorium. The samples were washed with five displacements of deionized water and vacuum dried prior to analysis.

Results

Radiological analysis data on the six tailings samples are given in Appendix F. Using data on natural radionuclide levels in soils reported in the literature as a frame of reference, the data obtained on the tailings samples indicates that radioactivity would not be an environmental concern. A comparision of literature data on radium 226 and thorium 232 levels in soils with analysis data on tailings samples is given in Table 3.

TABLE 2

CONCENTRATIONS OF HEAVY METALS IN TAILINGS SAMPLES COMPARED TO AVERAGE OR TYPICAL VALUES FOR IGNEOUS ROCK AND SOILS

	Concentration Range in Tailings Samples, ppm	Average or Typical Concentrations[1] Igneous Rock, ppm	Soils, ppm
Arsenic (As)	100 - 1,800	2	6
Cadmium (Cd)	4 - 31	0.2	0.06
Copper (Cu)	570 - 4,530	55	20
Iron (Fe)	69,600 - 390,000	56,000	38,000
Lead (Pb)	400 - 1,770	13	10
Mercury (Hg)	0.6 - 3.5	0.08	0.03
Selenium (Se)	<40 - 110	0.05	0.2
Silver (Ag)	9 - 78	0.07	0.1
Zinc (Zn)	1,260 - 10,500	70	50

TABLE 3

RADIOLOGICAL PARAMETER LEVELS IN TAILINGS SAMPLES COMPARED TO LEVELS IN SOILS

Isotope	Analysis Results on Tailings Samples, pCi/g	Levels Reported in the Literature for Soils, pCi/g[1]
Radium 226	0.2 ± 0.8 to 0.5 ± 0.9	0.3 to 2
Thorium 232	<1	0.01 to 1.4

[1] Source: Bowen, H.J.M., 1966, Trace Elements in Biochemistry, Adademic Press, New York, N.Y.

MINERALOGICAL EXAMINATIONS

Four of the six tailings samples were specified for mineralogical study. These were:

No. 1, Zinc Tailings Fines (massive ore)
No. 2, Zinc Tailings Fines (mixture massive/stringer ore)
No. 3, Combined Mine Backfill
No. 4, Unclassified Zinc Tailings (massive ore)

The objectives of the mineralogical examination were to provide an identification of the minerals present and to estimate the volume percentage of these for each sample.

Procedure

Initial Sample Preparation

Each whole sample was washed with isopropyl alcohol, decanted, and dried at low temperature (90-105°C). All solids settled from the suspensions rapidly enough to indicate that clay separations at 2μm particle size would not be necessary.

A split of each dried sample was then ground by hand in a mortar to a fine powder for x-ray diffraction analysis.

Other splits of each sample were submitted to an outside laboratory in their existing particle size conditions for preparation of polished grain mounts.

X-ray Diffraction (XRD) Analyses

The powered bulk samples were each scanned over the range $2\Theta = 3\text{-}61^\circ$ (CuKα radiation) on an x-ray diffractometer for semiquantitative estimation of their mineralogies, with the estimates of amounts being weight percentages based on peak heights. This procedure is generally able to identify minerals present in amounts greater than 1 to 5 percent by weight. Minerals less abundant then this are usually below the detection limits for the system.

Although XRD responds to weights and not volumes, if one assumes certain densities for the phases identified, it is readily possible to convert the weight percentages to volume percentages. This is only a problem for minerals for which the density is variable, depending on chemical substitution (e.g., chlorites). Another potential problem with this calculation for certain samples is that the theoretical density of a mineral may not be the same as its actual density, due to the presence of pore (void) spaces. The densities used for the minerals which were identified in significant amounts in the present samples are:

Mineral	Density (g/cm^3)
Pyrite	5.02
Quartz	2.65
Chlorite	2.90
Mica	2.84
Dolomite	2.86
Sphalerite	4.00

The quality of the XRD weight percentage estimates was controlled by comparing the peak heights on the patterns of the unknowns with the corresponding peak heights obtained from a standard mixture of known amounts of pure reference minerals. Measuring peak heights rather than peak areas was considered to be valid because all of the minerals present appear to be well-crystallized and gave sharp, symmetrical diffraction maxima. A standard sample of pure quartz is also scanned and counted daily on the diffractometer used, to detect any drift of the absolute intensities from the system. Any such drift is compensated by scaling (normalizing) all phases by the appropriate factor to give a total of one hundred percent.

Optical Microscopy

Polished grain mounts of the four samples were examined in reflected light under the ore microscope to better determine the details of the sulfide mineralogy. This was done in particular to determine the presence of sulfides other than pyrite, especially sphalerite (ZnS).

The potential for missing minor amounts of sphalerite by XRD is very real, since certain parts (peaks) of the sphalerite and pyrite x-ray patterns directly overlap each other and since the samples are tailings from known zinc-bearing material.

Results

Results of the mineralogical examinations are given in Table 4. All four samples show similar mineralogies in terms of both the species present and the approximate amounts of each species. In approximate order of decreasing abundance, the minerals present in the samples in general are pyrite, quartz, chlorite, mica, sphalerite, dolomite, chalcopyrite, siderite, and galena. Traces of other minerals may remain unidentified.

Pyrite is by far the most abundant sulfide mineral in the samples. It generally makes up >98% of the total sulfides, which in turn make up an estimated 43-55 percent by volume of the various whole samples. Minute amounts of other sulfides (sphalerite, chalcopyrite, galena) can only be observed microscopically -- they do not show up on the x-ray diffraction patterns due to very low abundance.

By microscopy, sulfides other than pyrite were seen to be very minor. Sphalerite does appear to be the most abundant nonpyrite sulfide, but it does not appear to make up more than one percent of any of the samples. No other zinc minerals appear to be present. The only other identifiable sulfides observed were traces of chalcopyrite and galena.

TABLE 4

MINERAL COMPOSITION AND ESTIMATED VOLUME PERCENTAGES OF TAILINGS SAMPLES NO. 1 THRU 4

Mineral	Formula	Sample No. 1	2	3	4
Pyrite	FeS_2	51	45	43	55
Quartz	SiO_2	35	41	51	34
Chlorite	$Mg,Fe,Al(Si_4O_{10})(OH)_8$	7	10	5	6
Mica	$K\ Al_2(AlSi_3O_{10})(OH)_2$	6	4	1	4
Sphalerite	ZnS	<1	<1	<1	<1
Dolomite	$CaMg(CO_3)_2$	1	Trace	Trace	<1
Chalcopyrite	$CuFeS_2$	Trace	Trace	Trace	Trace
Galena	PbS	Trace	Trace	Trace	Trace
Siderite	$FeCO_3$	-	-	-	Trace

Sample No.	Sample Designation
1	Zinc Tailings Fines (massive ore)
2	Zinc Tailings Fines (mixture massive/stringer ore)
3	Combined Mine Backfill
4	Unclassified Zinc Tailings (massive ore)

EPA EXTRACTION PROCEDURE TESTING

The leaching test specified by the U.S. Environmental Protection Agency (USEPA) for use in characterizing wastes under the RCRA (Resource Conservation and Recovery Act) program was conducted on each of the six samples. This test is commonly referred to as the EP (extraction procedure) toxicity test and essentially involves leaching with an acetic acid solution at a pH of 5 for a period of 24 hours. The final leachate to sample ratio is 20 to 1.

Procedure

The extraction procedure testing and leachate analyses were conducted according to the information published by the U.S. Environmental Protection Agency (Title 40, CFR 261.24 and Appendixes I, II, and III, Federal Register v. 45, no. 98, May 19, 1980). Procedural data on the tests as conducted on each of the six tailings samples are given in Appendix G. Leachates were analyzed for the eight inorganic parameters listed in the reference given above.

Results

The laboratory analysis data on the leachates from the EP toxicity testing indicated that, for all six samples, none of the eight inorganic parameters (arsenic, barium, cadmium, chromium, lead, mercury, selenium, and silver) were present in concentrations exceeding the threshold values given in Table 1 of CFR 261.24. Cadmium and lead concentrations in the leachates were generally within one order of magnitude of the threshold values, while the other parameters concentrations were generally about two orders of magnitude less than the threshold values. The threshold values and the leachate analysis data are included in Appendix G.

ACID PRODUCTION POTENTIAL TESTING

The six tailings samples were subjected to a testing procedure, specified by Exxon Minerals Company, which is designed to provide data for use in evaluating the potential of wasterock materials to produce or to consume acid. This procedure was developed at B.C. Research, Vancouver, British Columbia and was described in a paper presented by A. Bruynesteyn at the 108th annual meeting of the American Institute of Mining and Metallurgical Engineers (AIME), Feb. 19, 1979 and entitled "Determination of Acid Production Potential of Waste Materials."

Procedure

Details of the testing procedure are given in Appendix H. Two steps are involved. In the first step, the sample is analyzed for total sulfur as a measure of the theoretical maximum potential acid production, then precision titrated in a water slurry with 1-normal sulfuric acid for determination of potential acid consumption, and these results are compared on the basis of sulfuric acid equivalents. If potential acid production is indicated to exceed potential acid consumption, the second step is conducted to confirm the results of the first. In the second step, the sample is subjected in a water slurry to conditions known to be conducive to generation of acid from sulfide-bearing rock, and the extent of the acid generation is measured. The conditions of this confirmation test include an acid pH (2.5 to 2.8), innoculation with a type of bacteria (Thiobacillus ferroxidans) known to promote the oxidation of pyrite, addition of nutrient salts for the bacteria to the slurry, a temperature conducive to bacterial action (35°C), and agitation of the test slurry. The progress of the test is monitored by determining the pH of the slurry and analyzing the solution for the concentration of a dissolved metal (usually iron). The test is continued until pH no longer decreases or the dissolved metal concentration becomes constant. At that point, additional sample (½ the original weight) is added to the slurry, and after 24 hours the slurry pH is measured. If the pH is greater than 3.5 (indicating insufficient acid production and a negative potential for production of acid water) the test is ended. If the pH is 3.5 or less, another addition of ½ the original weight of sample is made, and the pH is again measured after 24 hours. If the pH is less than 3.5 or greater than 4.0 the test is ended, but if the pH falls between 3.5 and 4.0 the test is continued for an additional 48 hours and the final pH value is determined. The results are interpreted on the premise that if the pH of the test slurry remains in a range (less than 3.5) favorable to microbiological leaching action, then there is a positive potential for acid production from material represented by the sample.

Results

The results of the acid production testing are summarized in Table 5. Test data are detailed in Appendix H. These results indicate that all of the tailings samples tested have a definite potential for acid generation, except the pyrite tailings fines from a mixture of massive/stringer ore (Sample No. 6). For this sample, the initial sulfur analyses and acid consumption titration results indicated some positive potential for acid generation, but this was not confirmed by the biological test results which indicated that the material might not be a potential source for production of acid water.

TABLE 5

RESULTS OF ACID PRODUCTION POTENTIAL TESTING[1] ON TAILINGS SAMPLES

Sample No.	Description	Potential for Acid Production: Initial Chemical Test	Potential for Acid Production: Biological Confirmation Test
1	Zinc Tailings Fines (massive ore)	Positive	Positive
2	Zinc Tailings Fines (mixture massive/stringer ore)	Positive	Positive
3	Combined Mine Backfill	Positive	Positive
4	Unclassified Zinc Tailings (massive ore)	Positive	Positive
5	Pyrite Concentrate Fines (mixture massive/stringer ore)	Positive	Positive
6	Pyrite Tailings Fines (mixture massive/stringer ore)	Positive	Negative

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[1] B.C. Research procedure - see Appendix H for details.

AGITATED LEACH TESTING

Agitated leach tests on the tailings samples were conducted under three different conditions -- leaching with deionized water, leaching with sulfuric acid solution, and leaching with calcium hydroxide solution.

The objectives of this work were to study potential effects on leachate quality due to surface washing and diffusion controlled contributions, to study potential solubilization of ions and to study the formation of thiosalts in alkaline tailing slurries.

The method of agitation used was in parallel with that used in the EPA extraction procedure tests discussed in the preceding section. In this method, slurries are mixed in rotating plastic containers.

Procedure

Water Leaching Tests

All six tailings samples were subjected to leaching with deionized water. These leaching tests were conducted through three cycles. For each cycle, four parts of deionized water were slurried with one part of sample with a leaching time of 48 hours. At the end of each cycle, the sample slurries were pressure-filtered through a 0.45 micron filter, and the filtered leachates were analyzed for the parameters listed in Table 6. Also, portions of the filtered leachates from each cycle were combined into composite samples and analyzed for additional parameters as listed in Table 7. Details on the procedure used for each sample are given in Appendix I.

Leaching Tests Under Acid Conditions

All six tailings samples were leached in three cycles under acid conditions. For each cycle, one part of sample was slurried with four parts of pH 3 sulfuric acid solution. The leaching period was 48 hrs for each cycle. At the end of each cycle, the sample slurries were pressure-filtered through a 0.45 micron filter, and the filtered leachates were analyzed for the parameters listed in Table 8. Also, composite samples of leachates from the three cycles were formulated and analyzed for additional parameters as listed in Table 9. Details of the procedure used for each sample are given in Appendix I.

Leaching Tests Under Alkaline Conditions

Three tailings samples were tested under alkaline slurry conditions during one 48-hour leaching period. These samples were: No. 2(zinc tailings fines, mixture massive/stringer ore), No. 3 (combined mine backfill), and No. 5 (pyrite concentrate fines, mixture massive/stringer ore). The leach slurries were each composed of one part sample and one part solution. At the start of the leaching period, the pH levels of the slurries were adjusted to approximately 10 with hydrated lime, $Ca(OH)_2$. Then, at periods of 1, 4, 24, and 48 hours, the pH, the emf (measured with a platinum electrode: calomel reference electrode system to indicate oxidation-reduction potential), and the thiosulfate ($S_2O_3^{-2}$) concentration were determined. At the end of the leaching period, the sample slurries were pressure-filtered through a 0.45 micron filter and the filtered leachates were analyzed for the parameters listed in Table 10. Details on the testing procedure are given in Appendix I.

Results

Water Leaching Tests

Chemical analysis data on leachates are given in Tables I-1 and I-2 of Appendix I. From these data and material balance data given in Figure I-1 thru Figure I-6 of Appendix I, the extent of leaching for each chemical parameter was calculated. The results of these calculations are given in Tables 6 and 7. Table 6 shows the extent of leaching stage-by-stage, and Table 7 shows the overall leaching results.

TABLE 6

RESULTS OF 3-STAGE WATER LEACHING TESTS ON TAILINGS SAMPLES

Sample No.	Description	Leach Stage	pH	Conductivity mmhos/cm	Mg Leached/Kg Sample (dry basis)				
					Cu	Fe	Pb	Zn	SO_4
1	Zinc Tailings Fines (massive ore)	1	5.80	2.20	0.2	0.5	0.7	1,190	9,680
		2	5.65	1.14	<0.1	<0.1	0.1	60	2,840
		3	6.00	0.43	<0.1	<0.1	<0.1	20	680
2	Zinc Tailings Fines (mixture massive/stringer ore)	1	6.35	1.97	0.2	0.8	0.4	370	6,880
		2	6.00	0.58	<0.1	7	0.1	14	450
		3	6.20	0.29	<0.1	44	<0.1	<0.1	360
3	Combined Mine Backfill	1	6.55	0.81	0.2	<0.1	<0.2	70	1,990
		2	6.15	0.34	<0.1	<0.1	<0.2	14	300
		3	6.15	0.27	<0.1	<0.1	<0.2	8	370
4	Unclassified Zinc Tailings (massive ore)	1	5.90	2.03	0.1	2.4	1.0	1,130	9,410
		2	5.85	0.08	<0.1	<0.1	0.1	30	2,180
		3	6.00	0.41	<0.1	<0.1	<0.2	20	590
5	Pyrite Concentrate Fines (mixture massive/stringer ore)	1	4.90	2.02	32	240	11	2,470	9,920
		2	4.30	0.54	<0.5	190	4	100	740
		3	5.15	0.31	<0.1	210	0.1	90	600
6	Pyrite Tailings Fines (mixture massive/stringer ore)	1	6.60	1.56	0.1	<0.1	<0.2	2.3	4,600
		2	6.60	0.49	0.1	<0.1	<0.2	<0.1	310
		3	6.50	0.31	0.1	<0.1	<0.2	<0.1	290

TABLE 7

OVERALL RESULTS OF 3-STAGE WATER LEACH TESTS

Parameter	Total Mg Leached/Kg of Sample (dry basis)					
	Sample No.					
	1	2	3	4	5	6
Arsenic (As)	1.7	<0.1	0.5	2.1	8.6	<0.1
Barium (Ba)	<1.0	<1.0	<1.0	<1.0	1.2	1.2
Cadmium (Cd)	3.2	0.5	0.2	2.8	7.7	<0.1
Chromium (Cr)	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3
Copper (Cu)	<0.3	<0.3	<0.3	<0.3	32	<0.3
Iron (Fe)	0.5	52	<0.3	2.4	640	<0.3
Lead (Pb)	0.8	0.5	<0.4	1.1	15	<0.4
Manganese (Mn)	146	54	92	130	99	3.2
Mercury (Hg)	<0.007	<0.007	<0.007	<0.007	<0.007	<0.007
Selenium (Se)	2.1	1.1	1.6	2.6	1.2	3.1
Silver (As)	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Zinc (Zn)	1,270	380	92	1,180	2,660	2.4
Chloride (Cl)	<6	20	<6	<6	<6	<6
Fluoride (F)	2.2	3.7	1.3	2.1	<0.5	3.8
Nitrate (NO_3)	<0.6	2.2	<0.6	1.1	4.3	<0.6
Sulfate (SO_4)	12,970	7,560	2,580	11,960	11,140	5,200

Sample No.	Sample Designation
1	Zinc Tailings Fines (massive ore)
2	Zinc Tailings Fines (mixture massive/stringer ore)
3	Combined Mine Backfill
4	Unclassified Zinc Tailings (massive ore)
5	Pyrite Concentrate Fines (mixture massive/stringer ore)
6	Pyrite Tailings Fines (mixture massive/stringer ore)

The figures given in Table 7 on the overall leaching results indicate that sulfate, zinc, and manganese were the primary ions leached from the six samples. Minor amounts of iron were leached from the zinc tailings fines and pyrite concentrate fines produced from massive/stringer ore, but only trace amounts of iron were leached from the other four samples. Trace amounts of arsenic, cadmium, lead, and selenium were present in the leachates from most of the samples. Barium, chromium, copper, mercury, and silver concentrations in the composite leachates were generally below the analytical detection limits. Chloride concentrations in the composite leachates were below the analytical detection limit for all the samples except the zinc tailings fines produced from massive/stringer ore. Fluoride and nitrate were detected at trace levels in the composite leachates of some of the samples.

The data on the stage-leaching aspects given in Table 6 indicate that, for all six samples, most of the leaching occurred in the first stage and that the ions investigated were for the most part in solution with the sample cake moisture at the initiation of the testing. However, these data also indicate that some dissolving and diffusion of ions from the tailings continued during the second and third leach stages.

Data on leachate pH values and on iron leached may indicate that both the zinc tailings fines sample and the pyrite concentrate fines sample produced from a mixture of massive/stringer ore experienced some pyrite oxidation and incipient acid generation during the leach testing.

Leaching Tests Under Acid Conditions

Chemical analysis data on leachates from these tests are given in Tables I-3 and I-4 of Appendix I. From these data and material balance data given in Figure I-7 through Figure I-12 of Appendix I, the degree of leaching for each chemical parameter was calculated. The results of these calculations are given in Tables 8 and 9. Table 8 shows the results of leaching on a stage-by-stage basis, and Table 9 shows the overall leaching results.

The overall leaching results given in Table 9 indicate that sulfate, zinc, iron, copper and manganese were the primary ions leached from the tailings samples under acidic conditions. Arsenic, cadmium, lead, chloride, and nitrate were present in most of the leachates at much lower concentrations. Trace amounts of chromium, selenium, and silver were found in the leachate composites. Barium, mercury, and fluoride concentrations in the composite leachates were below the analytical detection limits.

The stage-by-stage leaching data given in Table 8 indicate that, for all six samples, most of the leaching occurred during the first stage. However, these data also indicate that dissolving and diffusion of ions from the tailings continued during the second and third stages.

Comparison of these leach tests conducted under acidic conditions with those conducted with deionized water indicates considerably more solubilization of arsenic, cadmium, copper, iron, lead, zinc, and sulfate. However, selenium concentrations were indicated to be lower in the acidic leachates than in those produced with deionized water.

TABLE 8

RESULTS OF 3-STAGE LEACHING TESTS ON TAILINGS SAMPLES UNDER ACID CONDITIONS

Sample No.	Description	Leach Stage	Final pH	Conductivity mmhos/cm	Mg Leached/Kg Sample (dry basis) As	Cu	Fe	Pb	Zn	SO_4[1]	Total S[1]
1	Zinc Tailings Fines (massive ore)	1	3.15	4.87	7.9	560	1,870	<0.3	4,460	20,580	7,000
		2	3.02	2.72	3.4	60	780	<0.3	130	7,780	2,620
		3	3.07	1.06	2.2	20	700	<0.3	110	1,500	520
2	Zinc Tailings Fines (mixture massive/stringer ore)	1	2.85	6.30	2.3	1,130	4,630	<0.3	4,640	28,780	9,650
		2	3.14	2.93	0.8	50	810	<0.3	10	9,180	3,040
		3	3.10	1.28	0.3	50	730	<0.3	40	1,790	620
3	Combined Mine Backfill	1	3.27	2.59	17.0	50	3,500	16	450	9,030	3,010
		2	3.09	0.80	7.5	20	35	25	30	820	290
		3	2.94	0.73	6.6	20	190	24	30	150	60
4	Unclassified Zinc Tailings (massive ore)	1	3.48	3.66	8.4	260	570	14	1,560	11,670	3,970
		2	3.31	2.21	4.3	120	600	13	410	4,950	1,630
		3	3.35	0.99	2.6	40	470	18	340	1,040	360
5	Pyrite Concentrate Fines (mixture massive/stringer ore)	1	3.33	3.28	13.5	440	1,640	14	1,770	10,900	3,660
		2	3.55	1.26	3.9	140	850	16	510	1,600	560
		3	3.45	0.89	2.1	80	560	18	330	890	320
6	Pyrite Tailings Fines (mixture massive/stringer ore)	1	3.84	3.31	0.5	190	780	13	500	10,830	3,500
		2	3.98	2.85	0.2	30	230	14	40	7,380	2,540
		3	4.07	1.28	0.1	20	100	15	20	2,050	690

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[1] Exclusive of that added in pH 3 leach solution and that added for pH adjustment

TABLE 9

OVERALL RESULTS OF 3-STAGE LEACH TESTS UNDER ACID CONDITIONS

Total Mg Leached/Kg of Sample (dry basis)

Parameter	Sample No. 1	2	3	4	5	6
Arsenic (As)	13.5	3.4	31.1	15.3	19.5	0.8
Barium (Ba)	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3
Cadmium (Cd)	13.5	12.5	1.4	7.0	7.8	1.3
Chromium (Cr)	0.9	1.1	0.4	0.4	0.7	0.3
Copper (Cu)	640	1,230	90	420	660	240
Iron (Fe)	3,350	6,170	3,725	1,640	3,050	1,110
Lead (Pb)	<0.3	<0.3	65	45	34	42
Manganese (Mn)	365	270	180	265	155	250
Mercury (Hg)	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003
Selenium (se)	0.04	0.04	0.1	0.3	0.1	0.07
Silver (As)	0.1	0.1	0.1	0.1	0.1	0.1
Zinc (Zn)	4,700	4,780	510	2,310	1,017	560
Chloride (Cl)	8	7	11	8	8	110
Fluoride (F)	<1.0	<1.0	<1.0	<1.0	<1.0	<1.0
Nitrate (NO_3)	<5	<5	17	52	30	35
Sulfate[1] (SO_4)	29,860	39,750	10,000	17,660	13,390	20,260
Total Sulfur[1] (S)	10,140	13,310	3,360	5,960	4,540	6,730

[1] Exclusive of that added in pH 3 leach solution and that added for pH adjustment

Sample No.	Sample Designation
1	Zinc Tailings Fines (massive ore)
2	Zinc Tailings Fines (mixture massive/stringer ore)
3	Combined Mine Backfill
4	Unclassified Zinc Tailings (massive ore)
5	Pyrite Concentrate Fines (mixture massive/stringer ore)
6	Pyrite Tailings Fines (mixture massive/stringer ore)

Leaching Tests Under Alkaline Conditions

The analysis results on the leachates from these tests after 48 hours of mixing are given in Table 10. The changes in thiosulfate ion concentrations in the slurry solutions during the period of leaching are plotted in Figure 2. The changes in pH are shown in Figure 3. The changes in oxidation-reduction potential, as indicated by emf measurements taken with a platinum electrode: calomel reference electrode, are shown in Figure 4.

These results indicate both formation of thiosalts and oxidation of these compounds under the alkaline test conditions. The term "thiosalts" refers collectively to thiosulfate ($S\text{-}SO_3^{-2}$), trithionate ($O_3S\text{-}S\text{-}O_3^{-2}$), tetrathionate ($O_3S\text{-}S\text{-}S\text{-}SO_3^{-2}$), and other higher polythionates ($O_3S\text{-}S_n\text{-}O_3^{-2}$). For these tests, analysis for thiosulfate was utilized as an indicator of the collective thiosalts probably present in the test slurries, and pH and cmf measurements served to corroborate the changes in thiosalt concentrations occurring in the test slurries.

The sequence of chemical reactions related to thiosalts generally accepted as taking place in sulfide tailings slurries under alkaline conditions is as follows:

- sulfide minerals + alkalinity → elemental sulfur and soluble sulfide
- elemental sulfur + soluble sulfide → polysulfides
- polysulfides + water + oxygen → polythionates and thiosulfate
- polythionates and thiosulfate + water + oxygen → sulfuric acid

The data plotted in Figures 2, 3, and 4 indicate that these reactions were occurring during the test period, with the first three reactions prevailing during the initial hours and the fourth reaction taking place through the remainder of the 48-hour period of testing.

TABLE 10

ANALYSIS RESULTS ON FINAL LEACHATES ALKALINE LEACHING OF TAILINGS

	Sample		
	2	3	5
Cu, mg/l	0.07	0.02	0.08
Fe, mg/l	<0.02	<0.02	<0.02
Pb, mg/l	<0.05	<0.05	<0.05
Zn, mg/l	89	0.19	8.3
SO_4, mg/l	2640	2320	2630
S_2O_3, mg/l	9	25	21
pH	6.48	8.02	7.46

Sample No.	Description
2	Zinc Tailings Fines (mixture massive/stringer ore)
3	Combined Mine Backfill
5	Pyrite Concentrate Fines (mixture massive/stringer ore)

FIGURE 2

THIOSULFATE CONCENTRATION IN SOLUTION
ALKALINE LEACHING OF TAILINGS

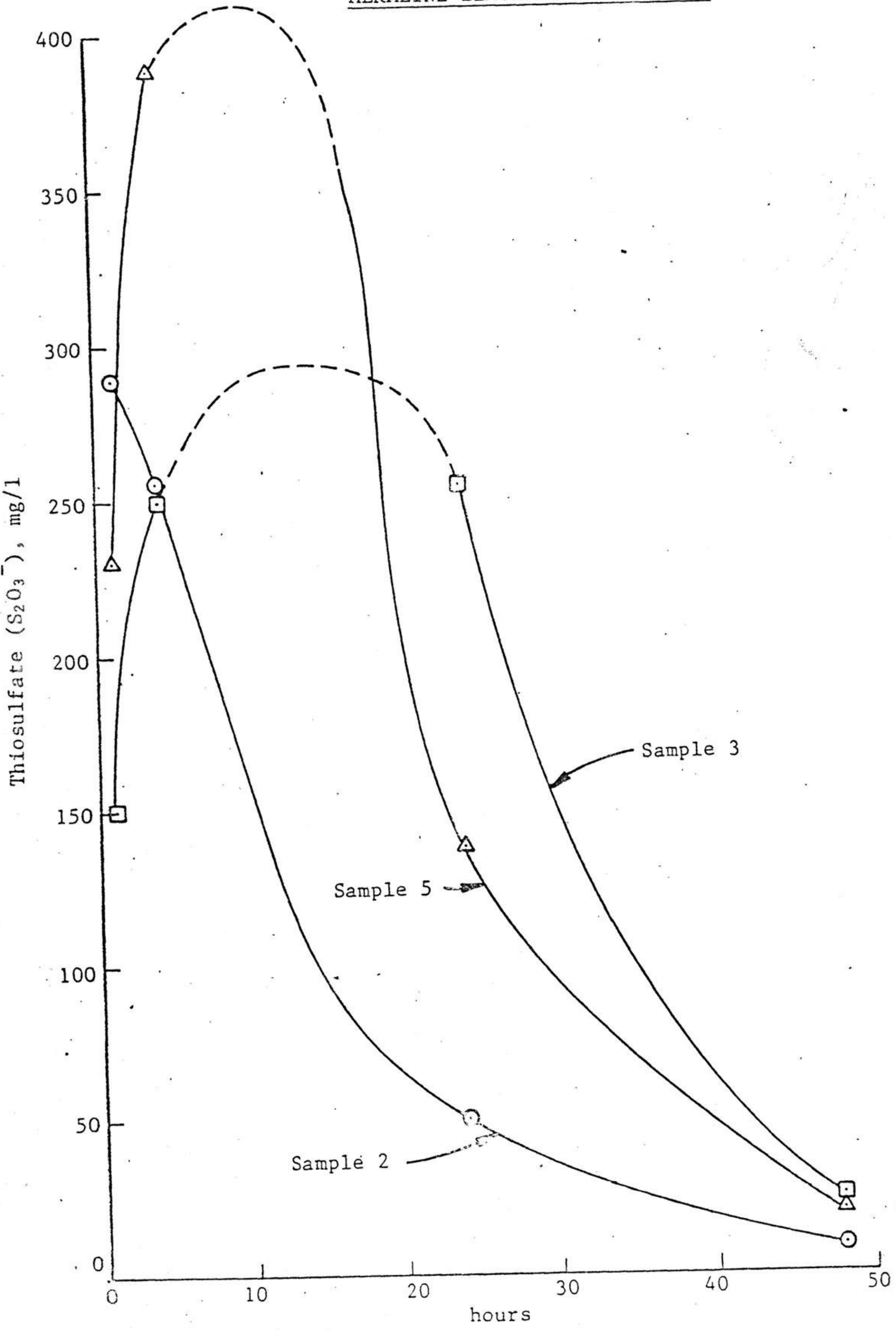


FIGURE 3

SLURRY pH
ALKALINE LEACHING OF TAILINGS

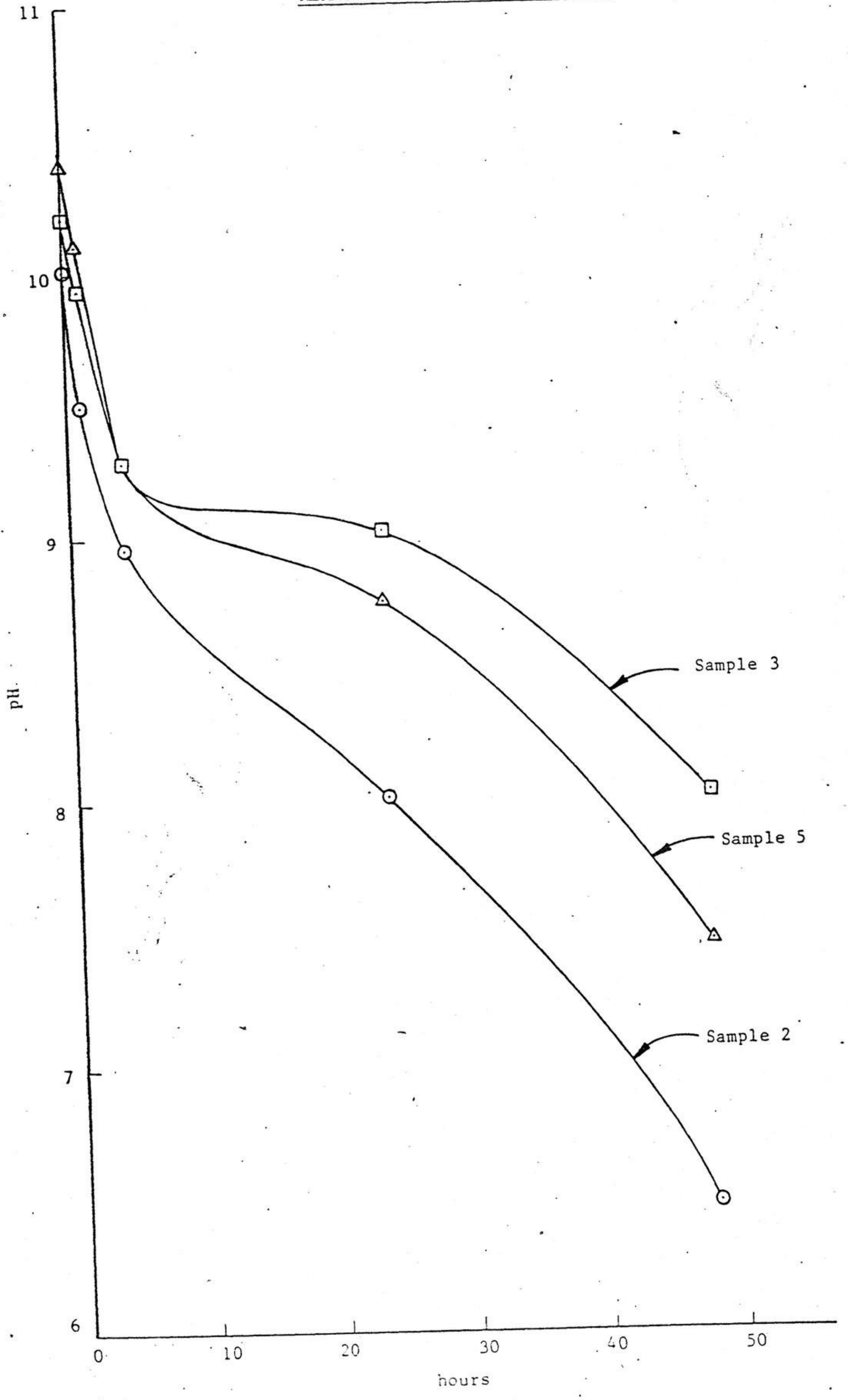
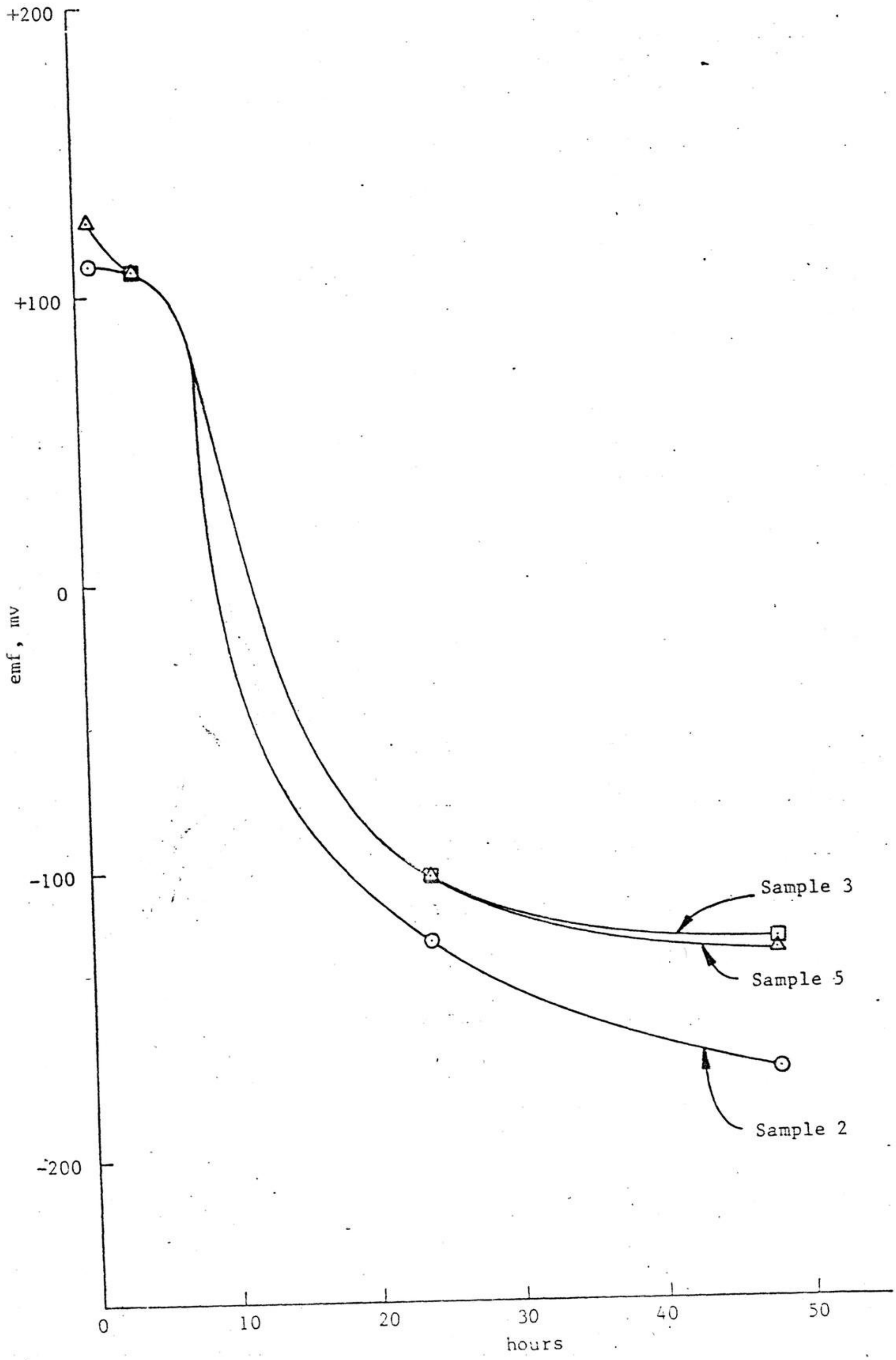


FIGURE 4

SLURRY EMF
ALKALINE LEACHING OF TAILINGS



COLUMN SEEPAGE STUDIES

Studies on seepage of water through the tailings were conducted on three samples under four different conditions. The tailings samples studies were:

- zinc tailings fines (massive ore)
- zinc tailings fines (mixture massive/stringer ore)
- combined mine backfill

The conditions studies were:

- tailings submerged under alkaline water (deionized water adjusted to pH 10 with calcium hydroxide)
- tailings submerged under deionized water (pH of 5.6)
- tailings submerged under acidic water (deionized water adjusted to pH 3 with sulfuric acid)
- tailings alternately submerged under deionized water and exposed to the air.

The objective of these studies was to provide information on seepage water characteristics and on physical and chemical changes in the tailings samples due to leaching effects.

Procedure

Tailings samples were contained in plastic columns under a fixed depth of water. Every two weeks, with the exception of one period which was three weeks, a volume of water equal to the pore volume of the tailings sample plus the pore volume of the filter at the bottom was drained from each column. An equal amount of makeup water was added after the drainage period. In the case of columns operated under submerged/exposed conditions, the amount of water drained and collected was reduced somewhat due to evaporation from sample surfaces.

Twelve columns were operated over a period of 31 weeks and 15 leachate samples were collected from each. Parameters determined on leachate samples included: pH, emf (an indicator of oxidation-reduction potential measured by a platinum electrode-calomel reference electrode system), conductivity, copper, iron, lead, zinc, calcium, potassium, magnesium, sulfate, and total sulfur. All samples were analyzed for pH and emf. The other parameters were determined on selected samplings as deemed appropriate to follow the trend of changes occurring in the columns.

Details on the configuration of the columns and on the test procedures employed in the study are given in Appendix J.

Results

Analysis data on seepage samples drained from the columns are given in Tables J-3 thru J-14 of Appendix J. From these data, and from weight and volume data given in Table J-2 of Appendix J, overall results were calculated and are presented in Table 11.

Table 11 shows the range of pH in seepage samples and the overall leaching of both total sulfur and zinc from the tailings. Due to the pH levels of the seepage, the other monitored heavy metals -- copper, iron, and lead were generally near or below the analytical detection limits. The data on pH, on total sulfur, and on zinc reporting in the seepage show generally similar seepage water characteristics for a given type of tailings irregardless of the four different study conditions. In the case of alkaline-submerged conditions, water added to the columns at pH 10 tended to exit with a lower pH, while deionized water added with a pH of 5.6 and acidic water added at a pH of 3 both exited with a higher pH. Seepage from the columns containing the two zinc tailings fines samples were probably at or near saturation with respect to sulfate ion thru the first 13-19 weeks of operation.

TABLE 11

OVERALL RESULTS OF COLUMN SEEPAGE STUDIES

	Alkaline Submerged Conditions	Deionized Water Submerged Conditions	Acidic Submerged Conditions	Deionized Water Alternate Submerged-Exposed Conditions
Zinc Tailings Fines (massive ore)				
pH Range of Seepage	7.54-8.07	7.38-8.21	7.39-8.16	7.67-8.32
Total Sulfur Leached, mg/kg of tailings	1,360	1,380	1,470[1]	1,600
Total Sulfur Leached, mg/l of seepage	220	220	230[1]	290
Zinc Leached, mg/k of tailings	23.5	20.5	26.9	18.8
Zinc Leached, mg/l of seepage	3.7	3.3	4.3	3.4
Zinc Tailings Fines (mixture massive/stringer ore)				
pH Range of Seepage	7.46-8.84	7.29-8.96	7.29-8.03	7.83-8.38
Total Sulfur Leached, mg/kg of tailings	2,190	2,180	2,310[1]	2,410
Total Sulfur Leached, mg/l of seepage	290	290	310[1]	350
Zinc Leached, mg/kg of tailings	3.5	3.1	4.9	3.5
Zinc Leached, mg/l of seepage	0.5	0.4	0.7	0.5
Mine Backfill				
pH Range of Seepage	7.38-9.43	7.44-9.59	7.27-8.86	7.02-7.98
Total Sulfur Leached, mg/kg of tailings	190	180	230[1]	340
Total Sulfur Leached, mg/l of seepage	40	35	40[1]	65
Zinc Leached, mg/kg of tailings	0.09	0.03	0.03	1.6
Zinc Leached, mg/l of seepage	0.02	<0.01	<0.01	0.3

[1] Exclusive of that added in pH 3 leach solution

The conductivity data on seepage water presented in the Appendix J tables is an indication of dissolved solids content. These data show a decreasing concentration of dissolved solids with time for all samples and conditions during the period of column operation. The chemical analysis data indicates that the dissolved solids in seepage water were principally calcium and magnesium sulfates with minor amounts of potassium and zinc sulfates. Due to the alkalinity, only trace amounts of copper, iron, and lead were present in the seepage water.

The measurements of emf on seepage water samples tabulated in Tables J-3 thru J-14 of Appendix J indicate fluctuating oxidation-reduction potentials. These fluctuations may indicate the occurrence of shifting chemical reactions involving sulfur species such as sulfide, elemental sulfur, polythionates, thiosulfate, sulfites, and sulfate. Also, these emf values were more positive than those measured on the waters added to the columns. This difference primarily reflects the presence of calcium sulfate, but also indicates that the seepage became more reducing as it passed through the tailings.

Little physical change was apparent in the tailings samples at the conclusion of the study period. The results of permeability measurements (presented in Table 15 of Appendix J) obtained on the tailings samples early in the study period and at the end showed very little change, with one exception. This was the mine backfill sample submerged under deionized water which became less permeable by a factor of approximately 10. Also, a thin layer of reddish-brown material formed on the top of the three tailings samples which were submerged under acidic water, and a thin film of this material was deposited on the walls of the columns above the level of the samples. This material was probably iron oxides formed from pyrite.

Tailings Submerged Under Alkaline Water

Data on seepage from columns operated with tailings submerged under alkaline water show that water added at pH 10 exited at markedly lower pH levels, except for the mine backfill column where the seepage eventually reached values greater than 9. The pH of the alkaline water was probably being partially reduced as it passed through the tailings by less basic material present in the tailings and partially as the result of thiosalt reactions as discussed in the preceding section. However, some part of the pH reduction appeared to be due to carbon dioxide absorption at the air-water interface of the columns. A blank column containing only deionized water adjusted to pH 10 with calcium hydroxide was monitored over a period of 3 weeks to check pH stability, and a change from pH 10 to pH 8.2 was measured in the top layer, while near the bottom of the column the pH was 8.6.

The characteristics of the seepage water from the columns operated with alkaline water were very similar to that drained from columns operated with deionized water.

Tailings Submerged Under Deionized Water

Seepage from columns operated with tailings submerged under deionized water tended to increase in pH over the period of the study, indicating that leaching of basic material was taking place in the tailings and that little or no acid formation from sulfide minerals was occurring.

The seepage waters from columns containing the two zinc tailings fines were at or near saturation with respect to sulfate ion from the beginning through the 13-15 weeks and then the concentration decreased markedly.

Tailings Submerged Under Acidic Water

The characteristics of seepage from columns operated with tailings submerged under acidic water were generally similar to those of seepage from columns operated with near-neutral water. Acidic water added to the columns at pH 3 exited as seepage with markedly higher pH throughout the length of the study period. All pH measurements of seepage from the three columns were above 7. This indicates that basic material in the tailings continued to react with the sulfuric acid in the water throughout the length of the study, and that little if any acid formation occurred.

Comparison of the data from this work with that from agitated leach testing under acidic conditions presented in a previous section (Table 9) indicates that considerably more solubilization of ions from the tailings could eventually occur from acidic seepage after depletion of the contained basic material.

Tailings Alternately Submerged Under Deionized Water and Exposed to the Air

The characteristics of seepage water from columns operated with the addition of deionized water and with periodic drainage and exposure to the atmosphere were generally similar to those of seepage water from columns operated where the tailings were always submerged. However, as indicated in Table 11, for all three tailings types studied the amount of sulfur leached was greater than for any of the columns operated under continuously submerged conditions. This indicates the occurrence of some oxidation of sulfide minerals. The mine backfill sample exhibited the largest difference, probably because it drained much faster than the zinc tailings fines due to a coarser particle size, and thus more of the sample was exposed to the air for a longer time. This sample was exposed to the atmosphere for approximately 45% of the time, while the zinc tailings fines samples had exposure for only about 10% of the time.

The seepage from columns containing the two tailings fines were at or near saturation with respect to sulfate ion for the first 15-19 weeks of operation and then, as in the case of columns operated under other conditions, the concentration decreased markedly.

CHARACTERISTICS OF SIMILAR TAILINGS

A limited literature review pertaining to the characteristics of high sulfide tailings generated from base metal ore concentrators was conducted. Approximately 50 references were reviewed. These were obtained both from computer searches of DIALOG files accessing the Engineering Index, NTIS, EIC, American Chemical Society, Cambridge Science Abstracts, and American Society of Metals reference lists and from CSMRI files. Ten of the references contained specific pertinent information on high sulfide tailings and these formed the basis for the following discussion. Citations for these are given in Appendix K.

Milling of massive sulfide ores for recovery of zinc, copper, or lead is conducted at many locations in the United States and Canada as well as in other countries around the world. Tailings from these operations have the common characteristic of potential acid generation with solubilization of heavy metals. In some cases, this potential is not realized because the tailings contain sufficient basic material (usually limestone or dolomite) to effectively suppress and neutralize acid generation (Lord, 1976).

Sulfide tailings deposits without sufficient basic material retain the potential for acid generation and many have eventually exhibited oxidation of sulfide material to sulfuric acid and leaching of solubilized heavy metals into surface and/or ground waters. The literature review did not reveal any situations where on-land deposits of high sulfide tailings deficient in basic material were not subject to oxidation processes leading to acid generation, except for one case where the acid generation potential was suppressed by a natural water cover supplied by a stream and springs (Hamilton and Fraser, 1978).

Boorman and Watson (1976) conducted detailed sampling and analysis of sulfide tailings and seepage at an abandoned impoundment located in New Brunswick, Canada with the objective of studying oxidation and leaching characteristics. These tailings were reported to contain significant amounts of pyrrhotite (FeS) along with pyrite (FeS_2). Other studies (Michelutti, 1974, 1978) have shown that pyrrhotite is more reactive with oxygen than the pyrite form of iron sulfide. Boorman and Watson found three chemical-physical zones within the tailings: an oxidation zone, extending from the tailings surface to a depth of about 50 centimeters; a hard pan, up to 15 centimeters thick beneath the oxidation zone; and a reduction zone, consisting of the underlying unaltered tailings. In the oxidation zone, moisture, oxygen from the air, and bacteria combined to oxidize the sulfide minerals to sulfuric acid and leach 75% of the residual copper and 90% of the residual zinc out of the zone. However, most of the residual lead remained in the zone as an insoluble sulfate. In the hard pan zone, iron hydroxides and oxides were deposited and copper was cemented out as a sulfide. Water passing through the hard pan was found to be saturated with iron (up to 36,000 mg/l), but copper in the seepage was depleted from several hundred mg/l above the hard pan to about 1 mg/l in the reduction zone. In the reduction zone, oxygen and oxygen-dependent bacteria were depleted. Seepage water from the reduction zone was indicated to contain 18,600 mg/l iron, 1 mg/l copper, 165 mg/l zinc, 6 mg/l lead, and 28,000 mg/l sulfate. Deposition of tailings into this impoundment was initiated in 1957 and terminated in 1963. The sampling was conducted approximately ten years later.

The mineral composition of the tailings mass within the unaltered reduction zone was reported as shown in Table 12. Chemical data reported on the unaltered tailings in the reduction zone was reported as shown in Table 13.

TABLE 12

MINERAL COMPOSITION OF NEW BRUNSWICK TAILINGS

UNALTERED REDUCTION ZONE

Pyrite	∿70%
Pyrrhotite	∿ 5%
Sphalerite	∿ 3%
Chalcopyrite	∿ 2%
Galena	∿ 1%
Chlorite	∿18%
Quartz	∿18%
Magnetite	Trace
Arsenopyrite	Trace
Cassiterite	Trace
Tetrahedrite	Trace
Challcocite	Trace
Covellite	Trace
Anglesite	Trace

TABLE 13

CHEMICAL DATA ON NEW BRUNSWICK TAILINGS

UNALTERED REDUCTION ZONE

	Wt. %
Copper	0.36 to 1.26
Lead	0.55 to 1.36
Zinc	1.00 to 3.28
Iron	35 to 41
Sulfur	34 to 44

The water penetration rate through the tailings mass was estimated as 100 l/m^3/day. The tailings particle size was reported as 95% -50 microns.

James and Mrost (1965) and Mrost and Lloyd (1967) investigated the oxidation and leaching of sulfide tailings from South African milling operations. Their work showed that oxidation of sulfide and consequent acid generation is controlled by the degree of oxygen diffusion into the tailings mass, and that for finely milled tailings exposed to natural conditions of precipitation and evaporation, oxidation was relatively rapid down to a depth of 10 cm, but would occur at slower rates down to a depth of about 190 cm even during extended periods of rainfall. These tailings were composed of pyrite with small quantities of pyrrhotite, chalcopyrite, and galena, gersdorffite, and millerite.

Sidhu (1979) reported on the particle size distribution and chemical composition of deposited mill tailings from a zinc-lead concentrator located at Buchans, Newfoundland. These tailings had been drained and during some periods of the year were dry enough at the surface to allow extensive generation of fugitive dust. Bulk samples of the tailings were taken at two depth intervals, 0 to 15 cm and 15 to 22 cm. Particle size distribution of the tailings was reported as follows:

	Weight %	
	0 - 15 cm	15 - 22 cm
+ 20 microns	84	79
- 20 + 2 microns	16	21
- 2 microns	0	0

The reported chemical analysis data is shown in Table 14.

TABLE 14

CHEMICAL ANALYSIS DATA ON BUCHANS, NEWFOUNDLAND TAILINGS

	0 - 15 cm		15 - 22 cm	
Parameter, ppm	Total	Plant Available[1]	Total	Plant Available[1]
Aluminum	22,480	--	22,694	--
Calcium	9,880	1,673	11,308	1,870
Cobalt	9	--	12	--
Chromium	27	0.6	30	0.5
Copper	823	8	920	4
Iron	20,655	--	29,106	--
Lead	25	--	25	--
Magnesium	8,500	9	8,911	17
Manganese	302	18	370	23
Nickel	44	4	42	4
Phosphorus	328	0.2	246	0.3
Potassium	5,255	40	5,158	43
Sodium	436	6	339	13
Zinc	3,874	152	5,424	85
pH	6.45		6.65	

[1] Agronomy analyses involving leaching of the sample.

Based on the iron analysis data, these tailings were relatively low in residual pyrite. The pH data indicates that acid generation was not significant or that the acid had been leached out. Because of these conditions, it was reported that successful revegetation of the area was accomplished with the addition of lime and nitrogen-phosphorus-potassium fertilizer and that the vegetation was persisting after two years of growth.

Michelutti (1974) (1978) studied the characteristics of sulfide tailings produced from flotation mills located in the Sudbury Basin of Ontario, Canada. These tailings can contain up to 18% sulfur and 29% iron, with the sulfide mineralization primarily pyrrhotite (FeS). Pyrrhotite is known to be much more reactive chemically then the pyrite form of iron sulfide. In this study, the pyrrhotite was reported to have a reaction rate with oxygen almost 2 orders of magnitude higher than pyrite. This type of tailings was investigated in regard to revegetation at one tailings impoundment which had been inactive for approximately 12 years and which had not been covered with soil. Two distinct layers of the deposited tailings were sampled and analyzed - an upper highly oxidized layer and a lower layer exhibiting a lower level of oxidation. The upper layer was characterized as containing a "ferruginous cap" which had a tendency to become rock-hard due to cementing by hydrated iron oxides and sulfates which were brought toward the surface by capillary action. Considerable acid formation and solubilization of heavy metals was also noted in the upper layer. The oxidized layer was found to vary in depth from about 3 cm to 150 cm, while the "ferruginous cap" varied in thickness from about 6 cm to 60 cm. No data was reported on the depth or thickness of the lower layer. Chemical analysis data reported for the samples of oxidized and unoxidized tailings are shown in Table 15.

A revegetation test plot was established on these tailings in 1973. The revegetation cover consisted of an 8-cm layer of chipped gravel (nominal - 2 cm chips) as a capillary barrier below a layer of local soil. Both limestone and fertilizer were added to the soil. After about a year and a half, the soil pH had changed from 7.4 to 6.6. At that time about 20 acres of the impoundment were covered in the same way and vegetation planted. Apparently, three years later vegetation was still established on both the test plot and on the 20-acre area.

TABLE 15

CHEMICAL ANALYSIS DATA ON SUDBURY BASIN TAILINGS

Parameter	Concentration, ppm Upper Layer	Concentration, ppm Lower Layer
Magnesium	42,500	64,000
Sodium	1,600	16,400
Potassium	4,100	7,400
Calcium	44,400	55,300
Aluminum	23,000	32,000
Sulfur	62,300	23,100
Nickel	500	2,200
Copper	310	630
Zinc	30	128
Cobalt	410	300
Iron	270,900	120,700
pH	3.0	4.5

Hamilton and Fraser (1978) compared the deposition of high sulfide tailings under two different conditions: underwater deposition with a water cover maintained, and deposition with draining and exposure of the tailings surface to the atmosphere.

Both deposits are located in the Flin Flon area of Manitoba, Canada. In the underwater case, the majority of the tailings had been in place for 32 years. In the exposed case, deposition had ceased 22 years previously.

For the underwater case, a dike and water flow from a stream and from springs maintained a water depth over the tailings of from 30 to 107 cm. Aquatic vegetation was established on the submerged tailings. The most prevalent species were sedges (Carlx spp.), river weed (Podastemun ceratophyllum Michx.), and spike rush (Eleocharis spp.). A layer of decaying vegetation was found underlying the standing vegetation. Sampling and analysis of the deposited tailings, water over the tailings, and the inlet and outlet waters of the impoundment indicated that very little oxidation had occurred.

In the exposed case, the results of sulfide oxidation processes were very evident. Three distinctly different layers were noted in the tailings: a top 15 cm of highly oxidized material, an intermediate lateritic hard pan of 10-15 cm, with essentially unoxidized material beneath the hard pan. Sampling and analysis of the top layer confirmed the visual evidence of oxidation.

Analytical data reported on samples from the two deposits are given in the following tables.

TABLE 16

CHEMICAL ANALYSIS DATA

ON FLIN FLON TAILINGS

Parameter	Tailings Composition, mg/kg Underwater Deposit Top 15 cm	Tailings Composition, mg/kg Exposed Deposit Top 15 cm
Total Iron	174,000	245,000
Iron Oxide (Fe_2O_3)	8,000	158,000
Total Sulfur	155,000	66,000
Sulfide Sulfur	154,000	27,000
Sulfate Sulfur	300	39,000
Alumina (Al_2O_3)	84,000	29,000
Silica (SiO_2)	357,000	277,000
Copper	9,100	1,900
Zinc	47,000	1,800
Lead	1,300	800
Cadmium	100	<100
Calcium	10,400	28,700
Magnesium	17,500	5,400

TABLE 17

SOLUBLE MATERIAL IN FLIN FLON TAILINGS

	Soluble Constituents in Tailings, mg/kg of Tailings	
Parameter	Underwater Deposit Top 15 cm	Exposed Deposit Top 15 cm
Iron	0.9	8.530
Copper	0.25	797
Zinc	15.5	218
Lead	0.35	1.27
Cadmium	0.05	0.57
Calcium	145	2.360
Magnesium	18	577
Sulphate	240	35,400
pH	6.9	2.5
Conductivity (mmho/cm)	0.4	7.20

TABLE 18

ACID GENERATING POTENTIAL OF FLIN FLON TAILINGS

Parameter	Underwater Deposit Top 15 cm	Exposed Deposit Top 15 cm
Original pH	6.7	2.5
Total Sulfur (Kg/tonne)	156	65
Acid Production Potential (Kg/tonne)	475	201
Acid Consumption (Kg/tonne)	52	0
Net Acid Production Potential (Kg/tonne)	424	201

TABLE 19

WATER ANALYSIS DATA FOR SUBMERGED DEPOSIT FLIN FLON TAILINGS

Parameter	Water Analysis at Submerged Deposit, mg/l		
	Impoundment Inlet	Impoundment Outlet	Above Tailings
Copper	0.01	0.02	0.01
Zinc	0.14	0.13	0.20
Lead	0.01	0.01	0.01
Cadmium	0.01	0.01	0.01
Iron	0.11	0.23	0.05
Calcium	22.0	22.5	21.0
Magnesium	5.3	5.5	5.6
Sulphate	28.8	21.7	-
pH	7.6	7.7	7.7
Conductivity (mmho/cm)	0.20	0.39	0.26

Ogram and Fraser (1978) conducted experimental studies on reclamation of both oxidized and freshly deposited high sulfide tailings. This work was conducted over a period of six years just preceding the study carried out by Hamilton and Fraser (1978) which is reviewed above. The same general type of tailings from the Flin Flon district of Manitoba Canada was studied. The chemical properties reported for these tailings are given in Table 20.

TABLE 20

TYPICAL CHEMICAL PROPERTIES OF FLIN FLON DISTRICT TAILINGS

pH	6.5 to 7.5 (fresh), 2.5 to 4.0 (after oxidizing)
Conductivity	9 to 15 mmhos/cm
Cation Exchange Capacity	Very low

Elemental Analysis	Range, %
Iron	20 - 25
Sulfur	14 - 27
Aluminum	2 - 4.5
Calcium	1.3 - 3.0
Magnesium	1.0 - 2.8
Zinc	0.5 - 1.1
Potassium	0.4 - 0.9
Sodium	0.2 - 0.5
Copper	0.06 - 0.4
Canganese	0.05 - 0.09
Arsenic	0.04 - 0.27
Phosphorus	0.01 - 0.03

Limestone, and nitrogen - phosphorus - potassium fertilizer applications, as well as various ground covers such as straw, excelsior blankets and jute mesh, were used on test plots of the tailings. Cover such as soil or soil-rock combinations were not used. After six years of attempts, all efforts of reclamation through establishment of vegetative cover were failures due to acid generation from oxidation of the sulfides in the tailings.

In addition to the potential for acid generation due to pyrite oxidation, sulfide tailings from mill circuits operating under alkaline conditions will contain polythionates (formed according to theory from the reaction of soluble sulfide ions with elemental sulfur present in the tailings solution).

These polythionates ($S_nO_2^{-2}$ where n = 2, 3, 4....) will oxidize to sulfuric acid with the passage of time under suitable conditions of oxygen availability.

Oko (1975) studied the aspects of polythionate formation and oxidation in tailings solutions and came to the following conclusions:

> Prolonged alkaline conditions results in increased concentrations of polythionates in the tailings solution.
>
> Oxidation of polythionates is biologically catalyzed by bacteria of the Thiobacillus thiooxidans group.
>
> The rate of oxidation of polythionates can increase rapidly when the pH of the water drops below 6.5.
>
> Warm summer conditions promote the rate of polythionate oxidation.
>
> To avoid rapid oxidation and subsequent acidic conditions in water downstream or down gradient, the polythionates must first be oxidized and then neutralized at the site.
>
> The practice of adding excess lime to the tailings slurry during transport to the storage impoundment should be questioned, and deposition of mildly acidic tailings should be considered in cases where polythionates present potential environmental problems.
>
> Retention of tailings solution in bio-oxidation and neutralization-sedimentation ponds placed in series is a means of handling the polythionate problem. However, natural biological oxidation will be minimal during the winter months in northern climates. During the summer months, a retention period of about six weeks in an open pond is needed for bio-oxidation of polythionates in climates similar to that of northern Ontario.

It should be noted that this situation will not apply to situations where tailings water is recycled and/or properly treated before release.

Rivet and Oko (1971) conducted a case study on high sulfide tailings disposal and water utilization problems in a mining district of the Sudbury Basin of Ontario, Canada. The aspects of this study on acid generation from tailings deposited in a dammed-up marsh draining to a lake system is pertinent to the conclusions given by Oko in his later paper of 1975. At the time of the study, alkaline tailings deposition had been underway for a period of ten years. No data was given on the volume of tailings deposited; however, based on information in the reference, it was probably in the range of 0.1 to 0.2 x 10^6 cubic meters. Outflow water from the impoundment flowed through a downstream lake where the nominal retention time was estimated as about six weeks. Analysis of the lake outflow water indicated an acidic pH and complete oxidation of dissolved polythionate compounds during the summer months. This outflow water then entered a much larger lake (which received other streams as well) with an estimated volume of 37 x 10^6 cubic meters (30,000 acre-feet) and an estimated nominal retention time of about 3 years. At the time of the study, the pH of the larger lake had declined to a value of 3.5 due to the acid contributed by the tailings deposited in the upstream marsh. Due to requirements of Ontario regulatory agencies and to requirements for a suitable water supply for a new ore concentrator planned for construction in the vicinity, a treatment system was built and operated to neutralize the entire outflow (27,000 to 136,000 liters per minute depending on the season) from the larger lake. Water analysis data reported in the study are given in Table 21.

TABLE 21

CHEMICAL ANALYSIS DATA ON TAILINGS WATER AND LAKE WATER
SUDBURY MINING DISTRICT

Parameter, mg/l	Outflow from Tailings Dam	Outflow from Larger Lake Before Neutralization	Outflow from Larger Lake After Neutralization
Dissolved Solids	987	330	390
Suspended Solids	1.5	1.0	1.0
Sulphate before Oxidation[1]	555	195	206
Sulphate after Oxidation[1]	679	194	206
Total Fe	6	2	0.6
Total Ni	0.6	0.3	0.3
Total Cu	0.5	0.1	0.1
C.O.D.[2]	46	less than 1	less than 1
pH	6.4	3.5	7.2

[1] The difference between "sulfate after oxidation" and "sulfate before oxidation" is a measure of the amount of polythionate compounds in the water.

[2] Chemical Oxygen Demand: indicates the presence of polythionate compounds:

APPENDIX A

CHAIN OF CUSTODY RECORDS

Chain of custody records on the tailings samples used in the course of this study are reproduced as Figures A-1 through A-6. A certificate of shipment from Lakefield Research of Canada Limited for the tailings samples is reproduced as Figure A-7.

EXXON MINERALS COMPANY

FIGURE A-1

CHAIN OF CUSTODY RECORD

Crandon Project
Phase I Pilot Plant

Collector's Sample No. —

Sample Description: Massive Zinc Tailing
-20 μm slimes
Deslime Test 13, Flowsheet 3
~350 lbs

Date Produced: April 21/81

Collector's Name: Brian Glanford
(Signature)

Phone: 705-652-3341

Remarks: Sample is in two (2) drums

Shipper's Name: Lakefield Research of Canada Ltd

Address: PO Box 430
Lakefield Ontario K0L 2H0
CANADA

Sample Receiver

1. Colorado School of Mines Research Institute Golden, Colo 80401 USA
(Name and address of organization receiving sample)
2.
3.

Chain of Possession

1.	Project Manager	May 15, 1981 --
(Signature)	Title	Inclusive Dates
2.		
3.		

Note: Exxon Minerals Company Sample Designation:
Sample No. 1, Zinc Tailings Fines (massive ore)

EXXON MINERALS COMPANY

FIGURE A-2

CHAIN OF CUSTODY RECORD

Collector's Sample No. —

Sample Description: Massive /Stringer Zinc Tail
-20 micron Slimes
Deslime Test 30, Flowsheet 6
~ 350 lb. dry weight

Date Produced: April 21/81

Collector's Name: Brian Glamford
(Signature)

Phone: 705-652-3341

Remarks: Sample is in 2 drums

Shipper's Name: Lakefield Research of Canada Ltd.

Address: P.O. Box 430
Lakefield Ontario K0L-2H0
CANADA

Sample Receiver

1. Colorado School of Mines Research Institute, Golden, Colorado 80
(Name and address of organization receiving sample)

2.

3.

Chain of Possession

1.	Project Manager	May. 15, 1981 —
(Signature)	Title	Inclusive Dates
2.		
3.		

Note: Exxon Minerals Company Sample Designation
Sample No. 2, Zinc Tailings Fines (mixture massive/stringer ore)

FIGURE A-3

EXXON MINERALS COMPANY

CHAIN OF CUSTODY RECORD

Crandon Project
Phase I Pilot Plant

Collector's Sample No. ____

Sample Description: Massive/Stringer Zinc Tailing
+20 μM Sands
Deslime Test 30, Flowsheet 6
~325 lbs

Date Produced: April 21/81

Collector's Name: Brian Glanford
(Signature)

Phone: 705-652-3341

Remarks: Sample is in two (2) barrels

Shipper's Name: Lakefield Research of Canada Ltd

Address: PO Box 430
Lakefield Ontario KOL 2H0
CANADA

Sample Receiver

1. Colorado School of Mines Research Inst, Golden, Colorado 80401 USA
(Name and address of organization receiving sample)
2.
3.

Chain of Possession

1.	Project Manager	May 15, 1981 --
(Signature)	Title	Inclusive Dates
2.		
3.		

Note: Exxon Minerals Company Sample Designation:
Sample No. 3, Combined Mine Backfill

EXXON MINERALS COMPANY

FIGURE A-4

CHAIN OF CUSTODY RECORD

Crandon Project
Phase I Pilot Plant

Collector's Sample No. /

Sample Description: Massive Zinc Tailing
Undeslimed
~350 lbs

Date Produced: April 21/81

Collector's Name: Brian Glasford
(Signature)

Phone: 705-652-3341

Remarks: Sample in one drum 750 lb

Shipper's Name: Lakefield Research of Canada Ltd

Address: P O Box 430
Lakefield, Ontario KOL 2H0
CANADA

Sample Receiver

1. Colorado School of Mines Research Institute, Golden, Colo 80401 USA
(Name and address of organization receiving sample)

2.

3.

Chain of Possession

1.	Project Manager	May 15, 1981 --
(Signature)	Title	Inclusive Dates
2.		
3.		

Note: Exxon Minerals Company Sample Designation:
Sample No. 4, Unclassified Zinc Tailing (massive ore)

EXXON MINERALS COMPANY

FIGURE A-5

CHAIN OF CUSTODY RECORD

Collector's Sample No. ——

Crandon Project
Phase I Pilot Plant

Sample Description: Massive/Stringer Pyrite Conc.
-30 μM slimes
Deslime Test 36, Flowsheet 4
~250 lbs

Date Produced: April 21/81

Collector's Name: Brian Glanford
(Signature)

Phone: 705-652-3841

Remarks: Sample is in one drum 350 lb.

Shipper's Name: Lakefield Research of Canada Ltd

Address: PO Box 430
Lakefield Ontario K0L 2H0
CANADA

Sample Receiver

1. Colorado School of Mines Research Inst. Golden, Colo. 80401 USA
(Name and address of organization receiving sample)

2.

3.

Chain of Possession

1.	Project Manager	May 15, 1981 --
(Signature)	Title	Inclusive Dates
2.		
3.		

Note: Exxon Minerals Compnay Sample Designation:
Sample No. 5, Pyrite Concentrate Fines (mixture massive/stringer ore)

FIGURE A-6

EXXON MINERALS COMPANY

CHAIN OF CUSTODY RECORD

Collector's Sample No. —

Crandon Project
Phase I Pilot Plant

Sample Description: Massive/Stringer Pyrite Tailing
-30 μM Slimes
Deslime Test 17 Flowsheet 5
~300 lbs

Date Produced: April 21/81

Collector's Name: Brian Glenford (Signature) Phone: 705-652-3341

Remarks: Sample is in two (2) barrels

Shipper's Name: Lakefield Research of Canada Ltd

Address:
PO Box 430
Lakefield Ontario K0L 2H0
CANADA

Sample Receiver

1. Colorado School of Mines Research Institute Golden, Colo 80401 USA
(Name and address of organization receiving sample)
2.
3.

Chain of Possession

1.	Project Manager	May 15, 1981 --
(Signature)	Title	Inclusive Dates
2.		
3.		

Note: Exxon Minerals Company Sample Designation:
Sample No. 6, Pyrite Tailings Fines (mixture massive/stringer ore)

FIGURE A-7

LAKEFIELD RESEARCH OF CANADA LIMITED
Lakefield, Ontario

CERTIFICATE OF SHIPMENT

SHIPPED TO: COLORADO SCHOOL OF MINES
RESEARCH INSTITUTE
GOLDEN, COLORADO, U.S.A. 80401

Date: April 22/81

SHIPPED TO
ATTENTION OF: MR. LARRY TISDEL

Number of containers

Contents:

Number of Pkgs.	Weight	
~~4 PALLETS~~	~~5550 LB~~	~~Desliming Products~~
~~1 DRUM~~	~~300 LB.~~	~~"~~
5 PALLETS	6640 LB	Desliming Products

SHIPPING DATE:
SHIPPED BY:
SHIPPED VIA:
OFFICE COPY:
DEPARTMENT COPY:
CLIENT'S COPY:
SHIPPED ON:
INSTRUCTIONS FROM: Jim Wennen
CHARGES TO: 2364
SPECIAL INSTRUCTIONS:

APPENDIX B

SAMPLE DESCRIPTIONS

Summary descriptions of the tailings sampled studied are given in Table B-1. The sample material is reported by Exxon Minerals Company to be typical of waste materials that will be retained in surface storage as a result of processing ore from the Crandon deposit. The indicated origin of the sample material was tailings generated during pilot-scale process development activities at Lakefield Research of Canada Ltd., Lakefield, Ontario.

TABLE B-1

SAMPLE DESCRIPTIONS

Sample No.	Description
1	Zinc Tailings Fines (massive ore) principal mineral composition of pyrite (FeS_2) and quartz (SiO_2), nominal - 30 micron particle size[1] 29.9% Sulfur 29.1% Iron 0.62% Zinc
2	Zinc Tailings Fines (mixture massive/stringer ore) principal mineral composition of pyrite (FeS_2) and quartz (SiO_2), nominal - 30 micron particle size[1] 22.2% Sulfur 23.1% Iron 0.54% Zinc
3	Combined Mine Backfill principal mineral composition of pyrite (FeS_2) and quartz (SiO_2) nominal - 150 micron particle size[1] 24.4% Sulfur 25.7% Iron 0.24% Zinc
4	Unclassified Zinc Tailings (massive ore) principal mineral composition of pyrite and quartz nominal - 75 micron particle size[1] 31.2% Sulfur 29.4% Iron 0.68% Zinc
5	Pyrite Concentrate Fines (mixture massive/stringer ore) principal mineral composition of pyrite (FeS_2) and quartz (SiO_2) nominal - 30 micron particle size[1] 41.1% Sulfur 39.0% Iron 1.05% Zinc
6	Pyrite Tailings Fines (mixture massive/stringer ore) principal mineral composition of quartz (SiO_2), chlorite (Mg, Fe, Al, silicate, hydroxide), and mica (K, Al, silicate, hydroxide) nominal - 40 micron particle size[1] 1.86% Sulfur 6.96% Iron 0.13% Zinc

[1] Detailed information on particle size distribution of tailings provided by Exxon Minerals Company is presented in Appendix C.

APPENDIX C

PARTICLE SIZE DISTRIBUTION DATA FOR TAILINGS SAMPLES

Data provided by Exxon Minerals Company on particle size distribution for tailings samples are shown in Table C-1. It is reported that these data were obtained on pilot plant products by Lakefield Research of Canada Ltd., and that Tyler Sieve screens and a cycolosizer apparatus were used to obtain the data.

TABLE C-1

SIZE ANALYSES OF TAILING SAMPLE PRODUCTS SUBMITTED FOR WASTE CHARACTERIZATION TESTING

Sample 1 Zinc Tailing Fines (Massive Ore) DS-13, FS-6

Size, µm	Wt % Retained Each Size	Cumulative Percent Passing
34.5	10.4	89.6
26.8	9.0	80.6
18.7	16.3	64.3
12.8	15.7	48.6
9.9	10.0	38.6
- 9.9	38.6	--
	100.0	

Sample 2 Zinc Tailing Fines (Massive/Stringer Ore) DS-30, FS-6

Size, µm	Wt % Retained Each Size	Cumulative Percent Passing
37.2	3.5	96.5
28.9	5.7	90.8
20.1	15.2	75.6
13.8	17.4	58.2
10.7	11.2	47.0
-10.7	47.0	--
	100.0	

Sample 3 Combined Backfill DS-30, FS-6

Size Mesh or µm	Wt % Retained Each Size	Cumulative Percent Passing
65 mesh	0.8	99.2
100	8.0	91.2
150	11.2	80.0
200	21.2	58.8
270	12.3	46.5
38.5 µm	8.7	37.8
29.9	15.0	22.8
20.9	16.6	6.2
14.3	3.6	2.6
11.1	0.5	2.1
-11.1	2.1	--
	100.0	

TABLE C-1 (continued)

SIZE ANALYSES OF TAILING SAMPLE PRODUCTS SUBMITTED FOR WASTE CHARACTERIZATION TESTING (Continued)

Sample 4 Undeslimed Zinc Tailings (Massive Ore) Feed to DS-13

Size Mesh or μm	Wt % Retained Each Size	Cumulative Percent Passing
150 mesh	1.2	98.8
200	5.4	93.4
270	5.9	87.5
36.6 μm	0.1	87.4
28.4	2.1	85.3
19.8	9.0	76.3
13.6	11.1	65.2
10.5	8.6	56.6
-10.5	56.6	--
	100.0	

Sample 5 Pyrite Concentrate Fines (Massive/Stringer Ore) DS-36, FS-4

Size, μm	Wt % Retained Each Size	Cumulative Percent Passing
30.3	4.5	95.5
23.5	6.0	89.5
16.4	17.3	72.2
11.3	19.0	53.2
8.7	10.6	42.6
- 8.7	42.6	--
	100.0	

Sample 6 Pyrite Tailing Fines (Massive/Stringer Ore) DS-17, FS-5

Size, μm	Wt % Retained Each Size	Cumulative Percent Passing
45.7	8.1	91.9
35.5	8.3	83.6
24.8	15.9	67.7
17.0	14.9	52.8
13.1	9.6	43.2
-13.1	43.2	--
	100.0	

APPENDIX D
SEMI-QUANTITATIVE EMISSION SPECTROGRAPHIC ANALYSIS DATA[1,2] ON TAILINGS SAMPLES[3]

	Sample Number					
	1	2	3	4	5	6
Aluminum (Al)	0.6	3	0.4	1	0.2	3
Antimony (Sb)	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003
Arsenic (As)	0.3	0.2	0.2	0.2	0.3	0.001
Barium (Ba)	0.002	0.004	0.004	0.004	0.002	0.006
Beryllium (Be)	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Bismuth (Bi)	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Boron (B)	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003
Cadmium (Cd)	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Calcium (Ca)	0.1	0.15	0.07	0.1	0.07	0.2
Cesium (Cs)	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Chromium (Cr)	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003
Cobalt (Co)	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Copper (Cu)	0.1	0.2	0.06	0.08	0.6	0.06
Gallium (Ga)	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Germanium (Ge)	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Gold (Au)	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Hafnium (Hf)	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Indium (In)	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Iridium (Ir)	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Iron (Fe)	>10	>10	>10	>10	>10	7
Lead (Pb)	0.05	0.05	0.04	0.07	0.05	0.04
Lithium (Li)	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Magnesium (Mg)	0.6	0.7	0.6	0.6	0.5	3
Manganese (Mn)	0.02	0.02	0.01	0.02	0.01	0.05

Sample No.	Description
1	Zinc Tailings Fines (massive ore)
2	Zinc Tailings Fines (mixture massive/stringer ore)
3	Combined Mine Backfill
4	Unclassified Zinc Tailings (massive ore)
5	Pyrite Concentrate Fines (mixture massive/stringer ore)
6	Pyrite Tailings Fines (mixture massive/stringer ore)

[1] All values in weight percent (%)

[2] Limits of accuracy reported as ± 50%

[3] Samples washed with 5 displacements of deionized water, vacuum dryed, and pulverized as necessary prior to analysis.

APPENDIX D - (Continued)

SEMI-QUANTITATIVE EMISSION SPECTROGRAPHIC ANALYSIS DATA[1,2] ON TAILINGS SAMPLES[3]

	Sample Number					
	1	2	3	4	5	6
Molybdenum (Mo)	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Nickel (Ni)	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Niobium (Nb)	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Osmium (Os)	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Palladium (Pd)	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Platinum (Pt)	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Potassium (K)	0.4	0.4	0.3	0.3	0.2	0.4
Rhenium (Re)	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Rhodium (Rh)	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Rubidium (Rb)	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Ruthenium (Ru)	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Silicon (Si)	>10	>10	>10	>10	>10	>10
Silver (Ag)	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Sodium (Na)	0.02	0.05	0.04	0.02	0.01	0.05
Strontium (Sr)	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Tellurium (Te)	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Thallium (Tl)	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Thorium (Th)	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Tin (Sn)	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003
Titanium (Ti)	0.006	0.01	0.008	0.006	0.006	0.03
Tungsten (W)	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Vanadium (V)	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Zinc (Zn)	0.6	0.4	0.3	0.6	1	0.08
Zirconium (Zr)	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01

Sample No.	Description
1	Zinc Tailings Fines (massive ore)
2	Zinc Tailings Fines (mixture massive/stringer ore)
3	Combined Mine Backfill
4	Unclassified Zinc Tailings (massive ore)
5	Pyrite Concentrate Fines (mixture massive/stringer ore)
6	Pyrite Tailings Fines (mixture massive/stringer ore)

[1] All values in weight percent (%)

[2] Limits of accuracy reported as ± 50%.

[3] Samples washed with 5 displacements of deionized water, vacuum dryed, and pulverized as necessary prior to analysis.

APPENDIX E

QUANTITATIVE CHEMICAL ANALYSIS DATA[1,2,3] ON TAILINGS SAMPLES

	Sample No.					
	1	2	3	4	5	6
Aluminum (as Al_2O_3)	29,000	45,000	22,000	23,000	20,000	68,000
Antimony (Sb)	<100	<100	<100	<100	<100	<100
Arsenic (As)	1,600	900	1,300	1,400	1,800	100
Barium (Ba)	<40	<40	<40	<40	<40	<40
Cadmium (Cd)	17	16	8	15	31	4
Calcium (as CaO)	4,400	5,260	2,170	4,260	4,448	8,870
Carbonate (CO_3)	6,500	7,000	4,000	6,000	2,500	15,000
Chloride (Cl)	120	<100	<100	<100	<100	<100
Chromium (Cr)	26	18	19	27	26	17
Copper (Cu)	1,260	1,690	580	1,190	4,530	570
Cyanide (CN)	5	4.5	<1	4.0	4.5	<1
Fluoride (F)	730	720	490	450	260	880
Gold (Au)	0.9	0.5	0.6	0.9	1.0	0.1
Iron (Fe)	291,000	231,000	257,000	294,000	390,000	69,600
Lead (Pb)	1,540	1,000	560	1,330	1,770	400
Magnesium (as MgO)	11,100	19,700	11,800	10,400	5,390	32,200
Manganese (Mn)	450	490	390	440	260	790
Mercury (Hg)	3.4	2.2	2.5	3.4	3.5	0.6
Nickel (Ni)	11	12	18	46	12	8
Phosphorus (P)	120	100	80	90	1,040	50
Potassium (as K_2O)	3,860	4,290	1,650	3,370	9,810	8,510
Selenium (Se)	<40	110	<40	<40	<40	90
Silica (SiO_2)	279,000	380,000	338,000	274,000	89,000	671,000
Silver (Ag)	58	43	28	57	78	9
Sodium (as Na_2O)	320	580	400	300	130	1,640
Sulfur (Total S)	299,000	220,000	244,000	312,000	411,000	18,600
Thallium (Tl)	<40	<40	<40	<40	<40	<40
Thorium (Th)	<10	<10	<10	<10	<10	<10
Titanium (Ti)	360	660	460	410	650	450
Uranium (U)	<10	<10	<10	<10	<10	<10
Zinc (Zn)	6,210	5,410	2,370	6,770	10,500	1,260
Sp. Gravity	3.84	3.48	3.38	3.85	4.51	2.84

[1] Samples washed with 5 displacements of deionized water, vacuum dryed, and pulverized as necessary prior to analysis.

[2] All values in parts per million (ppm) except Sp. Gravity.

[3] Analytical methods as cited in EPA Test Methods for Evaluating Solid Waste, Physical and Chemical Methods, U.S. EPA, SW-646, 1980.

Sample No.	Description
1	Zinc Tailings Fines (massive ore)
2	Zinc Tailings Fines (mixture massive/stringer ore)
3	Combined Mine Backfill
4	Unclassified Zinc Tailings (massive ore)
5	Pyrite Concentrate Fines (mixture massive/stringer ore)
6	Pyrite Tailings Fines (mixture massive/stringer ore)

APPENDIX F

RADIOLOGICAL ANALYSIS RESULTS ON TAILINGS SAMPLES

Sample No.	Description	Radium 226 pc/g	Radium 228 pc/g	Thorium 230 pc/g	Thorium 232 pc/g
1	Zinc Tailings Fines (massive ore)	0.2 ± 0.8	0.2 ± 2.5	0.6 ± 2.2	<1
2	Zinc Tailings Fines (mixture massive/stringer ore)	0.2 ± 0.7	1.5 ± 3.1	1.3 ± 2.5	<1
3	Combined Mine Backfill	0.5 ± 0.9	1.5 ± 1.6	3.0 ± 3.1	<1
4	Unclassified Zinc Tailings (massive ore)	0.2 ± 0.8	0.1 ± 2.5	0.6 ± 2.2	<1
5	Pyrite Concentrate Fines (mixture massive/stringer ore)	0.5 ± 0.9	0.0 ± 2.0	2.6 ± 3.1	<1
6	Pyrite Tailings Fines (mixture massive stringer/ore)	0.4 ± 0.9	0.6 ± 3.0	0.0 ± 1.2	<1

APPENDIX G

EPA EXTRACTION PROCEDURE TESTS

Extraction procedure tests were conducted on the tailings samples following the procedures published by the U.S. Environmental Protection Agency (Title 40 CFR 261.24 and Appendicies I, II, and III, Federal Register v. 45, no. 98, May, 1980). Leachates were analyzed using the methods approved by the USEPA for the eight inorganic parameters listed in Table 1 under 261.24 on p. 33122 of the above referred Federal Register. These parameters and their allowable maximum concentrations in the EP extract are listed below.

	Milligrams Per Liter
Arsenic	5.0
Barium	100.0
Cadmium	1.0
Chromium	5.0
Lead	5.0
Mercury	0.2
Selenium	1.0
Silver	5.0

Analysis data on the EP extracts are given in Table G-1. Figures G-1 through G-8 provide procedural data for the tests as conducted on each of the tailings samples.

TABLE G-1
CHEMICAL ANALYSIS DATA ON EP TOXICITY PROCEDURE EXTRACTS[1,2]

	1	2[3]	2a[3]	2b[3]	3	4	5	6
Arsenic	0.057	0.039	0.024	0.024	0.016	0.050	0.072	0.003
Barium	<0.2	<0.2	<0.1	<0.1	<0.2	<0.2	<0.2	<0.2
Cadmium	0.27	0.23	0.11	0.09	0.025	0.20	0.32	0.10
Chromium	<0.02	<0.02	<0.01	<0.01	<0.02	<0.02	<0.02	<0.02
Lead	1.40	0.67	0.66	0.67	0.35	0.90	4.15	0.35
Mercury	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	0.0014
Selenium	0.011	0.016	<0.005	<0.005	0.008	0.012	0.032	0.012
Silver	<0.05	<0.05	<0.01	<0.01	<0.05	<0.05	<0.05	<0.05

Sample No.	Description
1	Zinc Tailings Fines (massive ore)
2	Zinc tailings Fines (mixture massive/stringer ore)
3	Combined Mine Backfill
4	Unclassified Zinc Tailings (massive ore)
5	Pyrite Concentrate Fines (mixture massive/stringer ore)
6	Pyrite Tailings Fines (mixture massive/stringer ore)

---

[1] All values mg/l

[2] EPA Extraction Procedure for hazardous waste identification, Federal Register, v. 45, no. 98, May 19, 1980, pp. 33127-28.

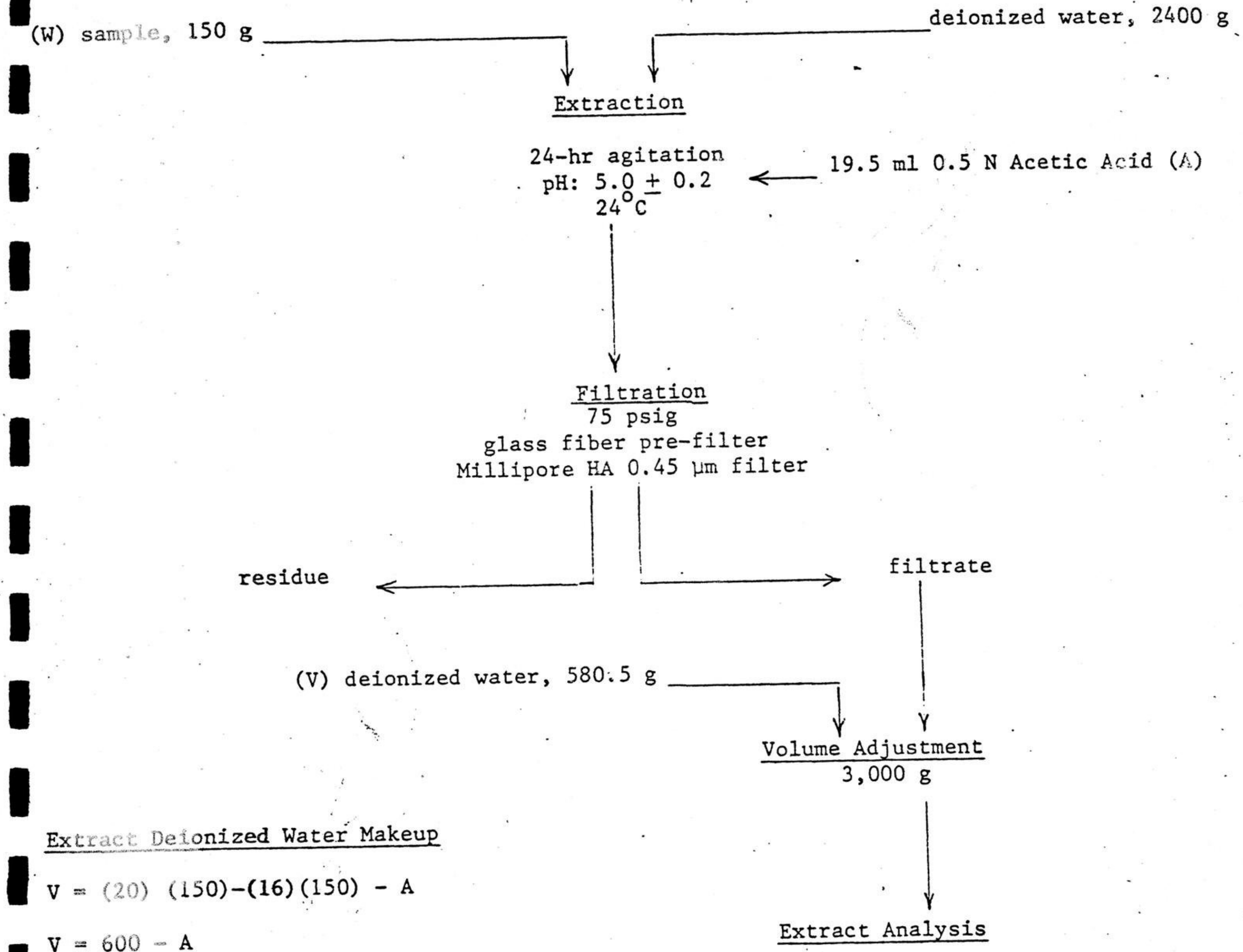
[3] Sample No. 2 was tested three times.

FIGURE G-1

Extraction Procedure, Sample No. 1

Sample Designation Zinc Tailing Fines (massive ore)

Project No. J10527 Operator M. Bishop Date 6-17-81



Extract Deionized Water Makeup

$V = (20)(150) - (16)(150) - A$

$V = 600 - A$

$V = 580.5$

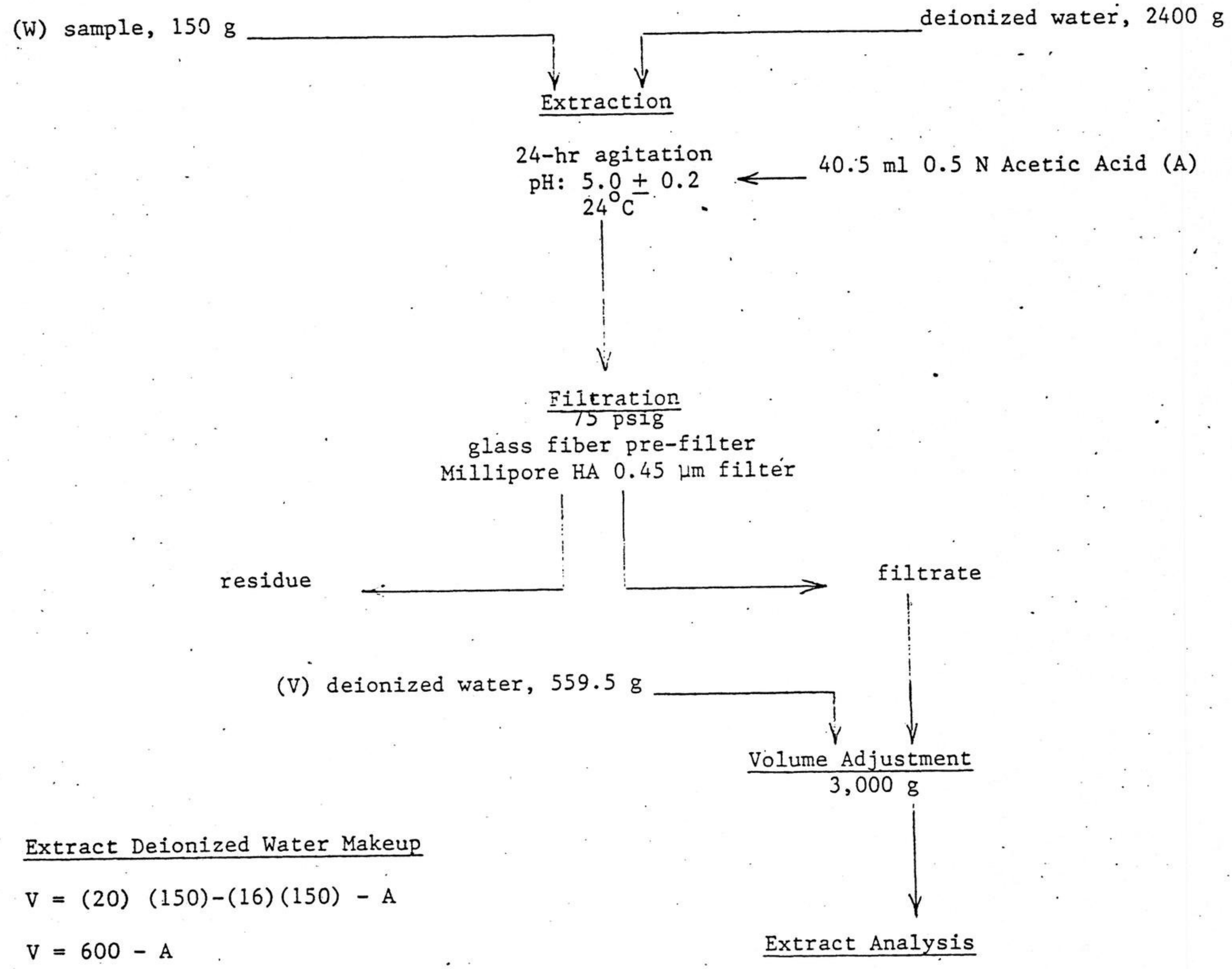
Ref: Environmental Protection Agency, 40 CFR Part 261 (FRL 1471-3) Hazardous Waste Management System: Identification and Listing of Hazardous Wastes, Appendix II, EP Toxicity Test Procedure, Federal Register, v. 45, no. 98, May 19, 1980.

FIGURE G-2

Extraction Procedure, Sample No. 2 (1st Test)

Sample Designation Zinc Tailing Fines ((mixture massive/stringer ore)

Project No. J10527 Operator M. Bishop Date 6-17-81



Extract Deionized Water Makeup

$V = (20)\ (150) - (16)(150) - A$

$V = 600 - A$

$V = 559.5$

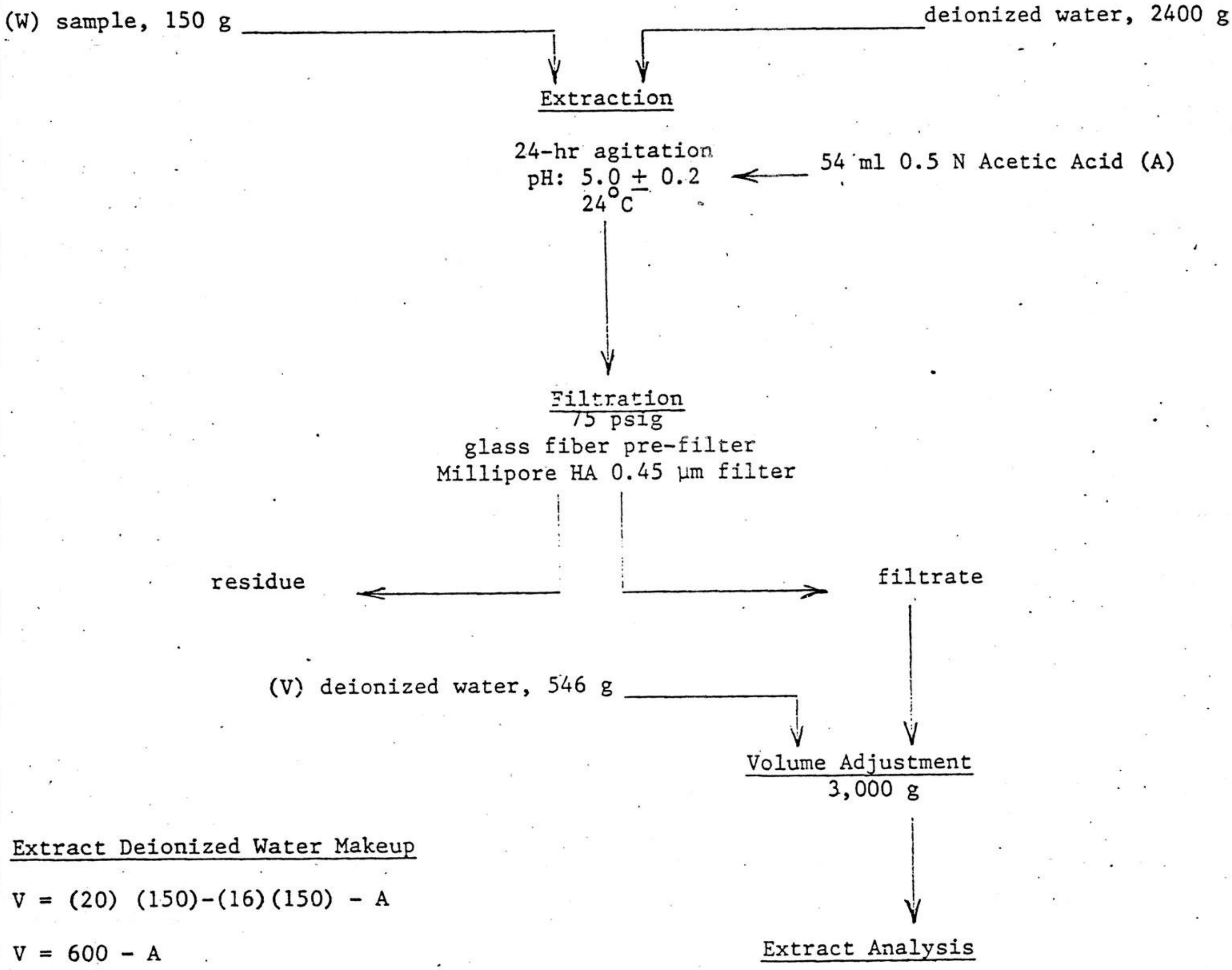
Ref: Environmental Protection Agency, 40 CFR Part 261 (FRL 1471-3) Hazardous Waste Management System: Identification and Listing of Hazardous Wastes, Appendix II, EP Toxicity Test Procedure, Federal Register, v. 45, no. 98, May 19, 1980.

FIGURE G-3

Extraction Procedure, Sample No. 2 (2nd Test)

Sample Designation Zinc Tailing Fines ((mixture massive/stringer ore)

Project No. J10527 Operator M. Bishop Date 7-20-81



Extract Deionized Water Makeup

$V = (20)(150) - (16)(150) - A$

$V = 600 - A$

$V = 546$ gms

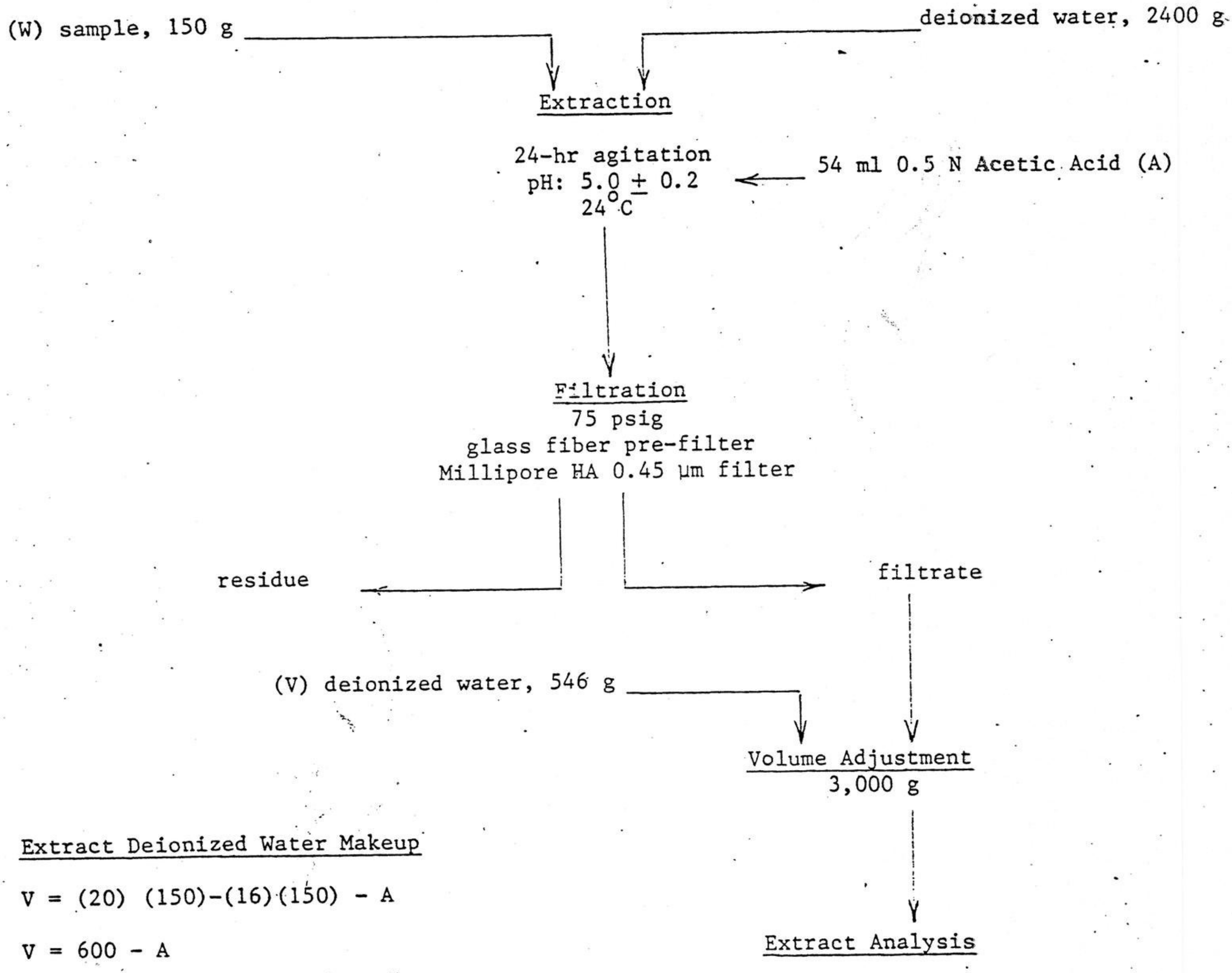
Ref: Environmental Protection Agency, 40 CFR Part 261 (FRL 1471-3) Hazardous Waste Management System: Identification and Listing of Hazardous Wastes, Appendix II, EP Toxicity Test Procedure, Federal Register, v. 45, no. 98, May 19, 1980.

FIGURE G-4

Extraction Procedure, Sample No. 2 (3rd Test)

Sample Designation Zinc Tailing Fines ((mixture massive/stringer ore)

Project No. J10527 Operator M. Bishop Date 7-20-81



Extract Deionized Water Makeup

$V = (20)\ (150) - (16)(150) - A$

$V = 600 - A$

$V = 546$ gms

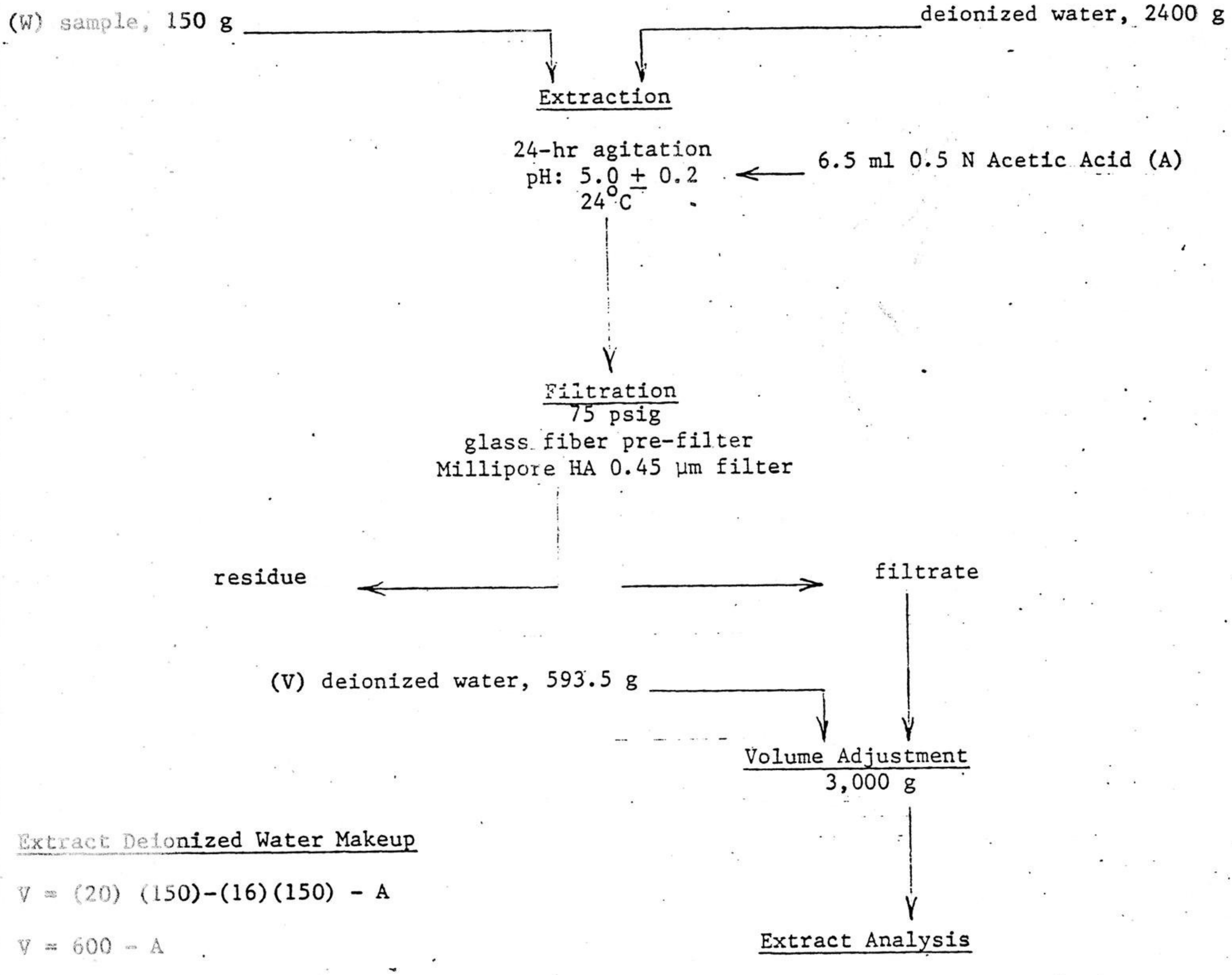
Ref: Environmental Protection Agency, 40 CFR Part 261 (FRL 1471-3) Hazardous Waste Management System: Identification and Listing of Hazardous Wastes, Appendix II, EP Toxicity Test Procedure, Federal Register, v. 45, no. 98, May 19, 1980.

FIGURE G-5

Extraction Procedure, Sample No. 3

Sample Designation Combined Mine Backfill

Project No. J10527 Operator M. Bishop Date 7-20-81



Extract Deionized Water Makeup

$V = (20)(150) - (16)(150) - A$

$V = 600 - A$

$V = 593.5$

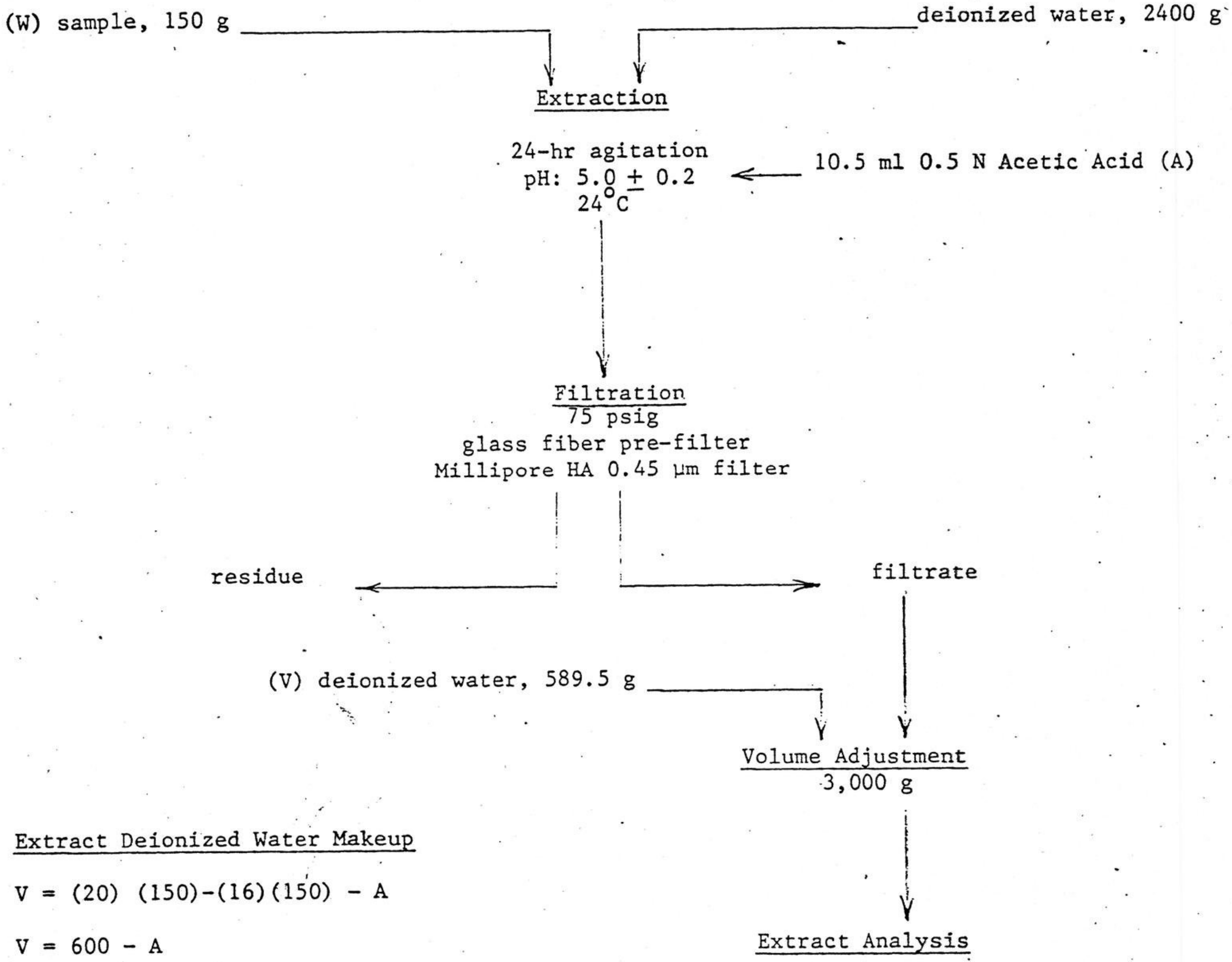
Ref: Environmental Protection Agency, 40 CFR Part 261 (FRL 1471-3) Hazardous Waste Management System: Identification and Listing of Hazardous Wastes, Appendix II, EP Toxicity Test Procedure, Federal Register, v. 45, no. 98, May 19, 1980.

FIGURE G-6

Extraction Procedure, Sample No. 4

Sample Designation Unclassified Zinc Tailings (massive ore)

Project No. J10527 Operator M. Bishop Date 7-20-81



Extract Deionized Water Makeup

$V = (20)\ (150) - (16)(150) - A$

$V = 600 - A$

$V = 589.5$

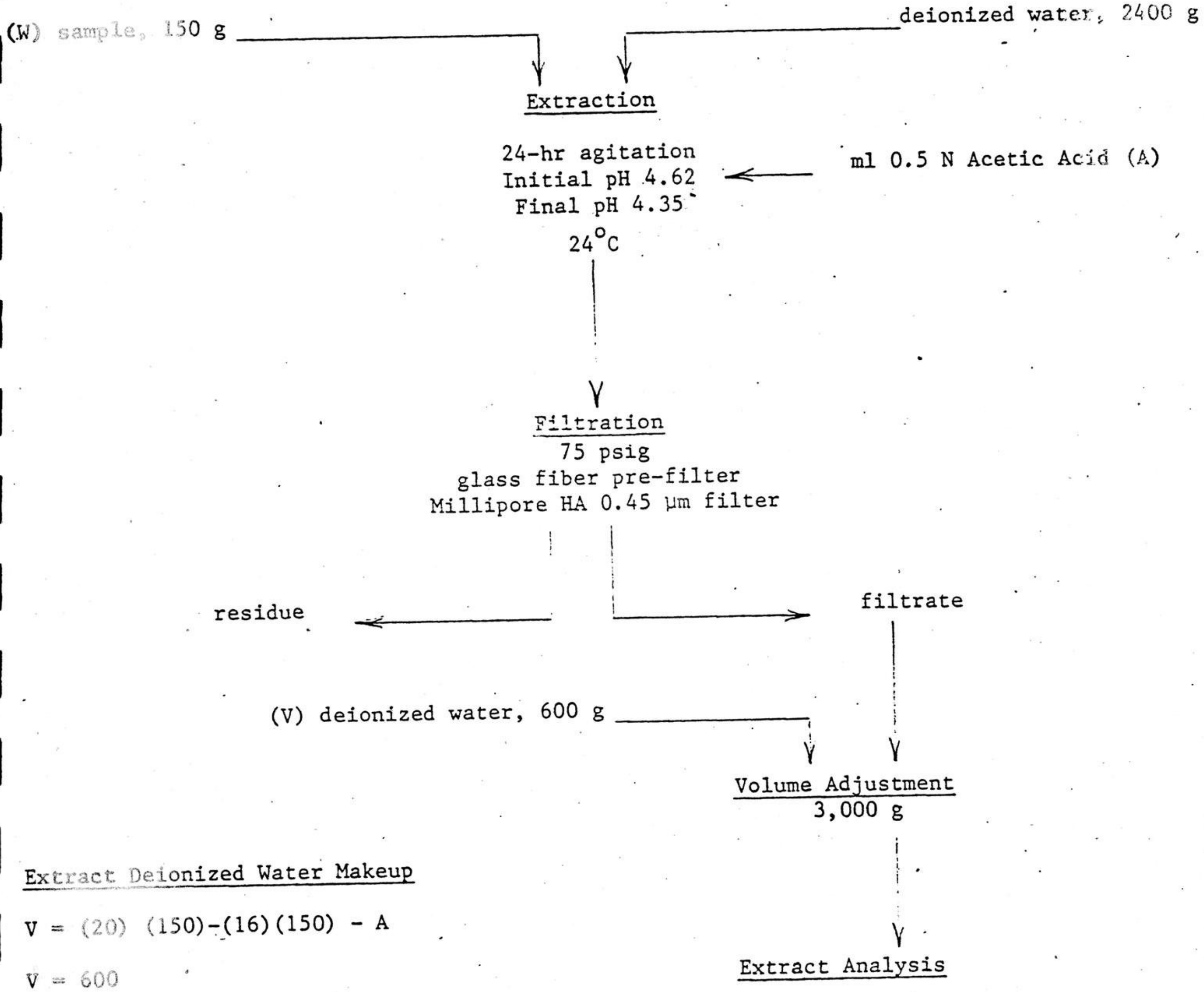
Ref: Environmental Protection Agency, 40 CFR Part 261 (FRL 1471-3) Hazardous Waste Management System: Identification and Listing of Hazardous Wastes, Appendix II, EP Toxicity Test Procedure, Federal Register, v. 45, no. 98, May 19, 1980.

FIGURE G-7

Extraction Procedure, Sample No. 5

Sample Designation Pyrite Concentrate Fines (mixture massive/stringer ore)

Project No. J10527 Operator M. Bishop Date 7-20-81



Extract Deionized Water Makeup

$V = (20)(150) - (16)(150) - A$

$V = 600$

$V = 600$

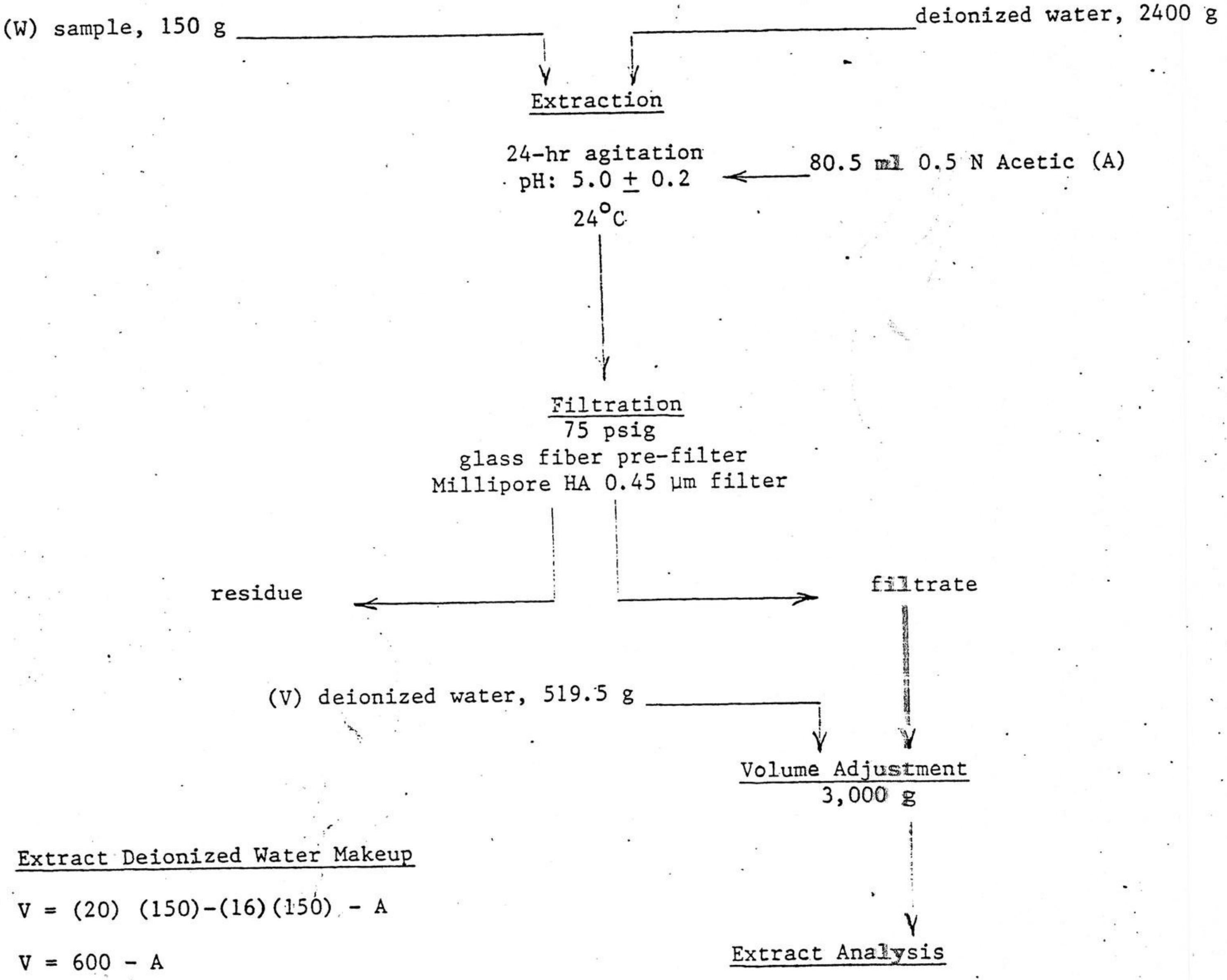
Ref: Environmental Protection Agency, 40 CFR Part 261 (FRL 1471-3) Hazardous Waste Management System: Identification and Listing of Hazardous Wastes, Appendix II, EP Toxicity Test Procedure, Federal Register, v. 45, no. 98, May 19, 1980.

FIGURE G-8

Extraction Procedure, Sample No. 6

Sample Designation Pyrite Tailings Fines (mixture massive/stringer ore)

Project No. J10527 Operator M. Bishop Date 7-20-81



Extract Deionized Water Makeup

$V = (20)(150) - (16)(150) - A$

$V = 600 - A$

$V = 519.5$

Ref: Environmental Protection Agency, 40 CFR Part 261 (FRL 1471-3) Hazardous Waste Management System: Identification and Listing of Hazardous Wastes, Appendix II, EP Toxicity Test Procedure, Federal Register, v. 45, no. 98, May 19, 1980.

APPENDIX H

ACID PRODUCTION POTENTIAL TESTS

All six tailings samples were subjected to a testing procedure, specified by Exxon Minerals Company, which has the objective of evaluating the potential of wasterock materials for producing acid or consuming acid. The testing procedure was developed at B.C. Research, Vancouver, British Columbia and was described in a paper presented by A. Bruynesteyn at the 108th annual meeting of AIME, Feb. 9, 1979 and entitled "determination of Acid Production Potential of Waste Materials."

PROCEDURE

The information on testing procedures provided by Exxon Minerals Company is given in full below. A few modifications were necessary to adopt the procedures to the tailings samples. These included: use of the standard gravimetric method for total sulfur analysis instead of the Leco furnace method due to the high sulfur content of the tailings samples; in the biological confirmation tests, wet ball-milling and oven-drying at 105°C were not employed since the sample materials were already at a suitably small particle size; and also in the biological confirmation tests, sample weights used were on the basis of approximately equal sulfur content to provide a better comparison between samples. These weights were as follows:

Sample	%s	Biological Confirmation Test Wt., g
1	30	2.0
2	22	3.0
3	24	3.0
4	31	2.0
5	41	1.5
6	1.9	30.0

The information provided by Exxon Minerals Company on test procedures follows.

TEST PROCEDURES FOR EVALUATING ACID-PRODUCING POTENTIAL OF ORE AND WASTE ROCK

INITIAL TEST (CHEMICAL)

Sample

The sample selected must be taken in such a manner that it is truly representative of the type of mineralization being examined. A composite made up of split drill core or of randomly selected grab samples should be satisfactory. The number of samples to be examined will depend on the variability of the mineralization and must be left to the discretion of the geologist taking the samples. The bulk sample is crushed to a size which can be conveniently handled, (i.e. -2 in.) and then thoroughly mixed and approxiamately a 2-lb portion split out, by coning and quartering. This sample is then pulverized to pass a 100 mesh screen and used for assay, the titration test, and the confirmation test if necessary.

Assay

The pulverized sample is assayed in duplicate for total sulfur, using a Leco furnace. The total sulfur assay value is expressed as pounds of sulfuric acid per ton of sample, assuming a 1:1 conversion factor, which is the acid-producing potential of the sample.

Titration Test

Duplicate 10-g portions of the pulverized sample are suspended in 100 ml of distilled water and stirred for approximately 15 minutes. The natural pH of the sample is then recorded and the sample titrated to pH 3.5 with 1.0 N sulfuric acid with a radiometer automatic titrator. The test is continued until less than 0.1 ml of acid is added over a 4-h period. The total volume of acid added is recorded and converted to lb per ton of sample. This is the acid-consuming ability of the sample, i.e.

$$\text{acid-consuming ability (lb/ton)} = \frac{\text{ml of } 1.0\text{ N } H_2SO_4 \times 0.049 \times 2000}{\text{wt of sample in g}}$$

$$\text{or for a 10-g sample} \quad = \text{ml of } 1.0\text{ N } H_2SO_4 \times 9.8$$

CONFIRMATION TEST (BIOLOGICAL)

Sample

The remaining portion of the pulverized sample is ball-milled (wet) for 2 to 3 h to produce a 400 mesh sample which is dried overnight at 105°C.

Shake-Flask Leaching Test

Duplicate 30-g portions (or a smaller amount if the sulfide content exceeds 2%) are placed in 250 ml Erlenmeyer flasks with 70 ml of a nutrient medium containing 3 g/l $(NH_4)_2SO_4$; 0.10 g/l KCl; 0.50 g/l K_2HPO_4; 0.50 g/l $MgSO_4 \cdot 7H_2O$; 0.1 g/l $Ca(NO_3)_2$. Add sufficient sulfuric acid (either 12 or 36 N) to bring the pH to 2.5. Shake the flasks for approximately 4hr and the pH should be between 2.5 and 2.8. If necessary add additional acid until the pH remains in that range, and then inoculate the flasks with 5 ml of an active culture of Thiobacillus ferrooxidans. Record the weight of the flasks and contents. Plug the flasks with a loose cotton plug and incubate at 35°C on a gyratory shaker.

The experimental leaching flasks are returned to their original weight before sampling by adding distilled or de-ionized water. Monitor the pH and concentration of a dissolved metal, e.g. iron, copper or zinc, for the first three days to ensure that the pH remains below 2.8. Thereafter, monitor every second day until microbiological activity has ceased, i.e. the pH no longer drops or the dissolved metal concentration remains constant.

When microbiological activity has ceased, add half the weight of feed used originally (i.e. 15 g), shake 24 h and record the pH. If it is greater than pH 3.5, terminate the test. If it is 3.5 or less, again add half the weight of feed (i.e. 15 g) and shake for 24 h. If the pH is less than 3.5 or greater than pH 4, the experiment is terminated. If the pH is between 3.5 and 4.0, the sample is shaken an additional 48 h and the final pH value recorded.

Interpretation

The object of this test is to determine if the sulfide-oxidizing bacteria can generate enough sulfuric acid from the sulfides present to satisfy the sample's acid demand. Experience has shown that not all sulfide minerals are amenable to microbiological attack nor do they all oxidize completely. So the acid-producing potential indicated by the sulfur assay may over-estimate the true acid producing potnetial. If the bacteria generate the acid, microbiological action will continue on a self-sustaining basis if it becomes established, and acidic mine water will result. In this test, the acid demand is satisfied initially by adding sulfuric acid. This permits the bacteria to generate the maximum amount of sulfuric acid from the sample concerned. Once microbiological action has ceased, half the original sample weight is added. If there has not been sufficient acid production, the pH will approach the natural pH of the sample (i.e. above pH 3.5) and the sample is reported as not being a potential source of acid mine water. If the pH remains at 3.5 or below, the remainder of the sample is added and the sample is shaken for up to 72 h before measuring the final pH. If the pH is still in the leaching range, i.e. pH 3.5 or below, there is a strong possibility that natural leaching will occur and acid mine drainage will be produced. if the pH is above 3.5, there is no possibility of acid mine drainage occurring.

If the sample produces excess acidity, there is the possibility of metal recovery by microbiological leaching. A measure of this potential can be obtained by estimating the percentage of the contained metal which has been solubilized during the leaching test. Under such circumstances, it may be desirable to promote microbiological action as a means of recovering valuable metals from a waste material. In such a system, suitable precautions must be taken to prevent the metal and acid-rich leach waters from entering the natural drainage system of the surrounding area.

DATA AND RESULTS

The data obtained and the interpretation of results for the acid production testing is given in the following six tables (H-1 through H-6).

These results indicate that of the six sample materials tested, all but the relatively low-sulfur content pyrite tailings fines from a mixture of massive/stringer ore (Sample No. 6) have a definite potential for acid generation. In the case of Sample No. 6, the initial sulfur analyses and acid consumption titration results indicated some positive potential for acid generation; however, the biological confirmation test results indicate that this material would not be a potential source for production of acid water.

TABLE H-1

ACID PRODUCING POTENTIAL TEST RESULTS

Sample: 1

Initial Chemical Test (In Duplicate)	A	B
Initial Slurry pH:	7.20	7.13
Acid Potential		
Total Sulfur Analysis, %:	29.8	30.0
Equivalent H_2SO_4, lb/ton:	1,823	1,835
Acid Consumption		
In H_2SO_4 titration, ml:	2.348	2.457
Equivalent H_2SO_4, lb/ton:	23	24

Interpretation: Positive acid producing potential is indicated.

Biological Confirmation Test (In Duplicate)

	pH		Fe, mg/l		
Day	A	B	A	B	
0	2.73	2.62	-	-	Inoculation with Thiobacillus ferroxidons
1	2.71	2.63	64	70	
2	2.76	2.65	63	66	
3	2.71	2.62	64	64	
5	2.54	2.42	-	-	
7	2.40	2.29	68	83	
9	2.34	2.25	124	114	
12	2.23	2.18	227	215	
14	2.19	2.16	272	276	
16	2.12	2.11	368	308	
19	2.07	2.10	456	317	
21	2.04	2.12	519	347	
23	2.05	2.15	583	382	
26	2.04	2.12	-	-	Addition of ½ sample weight
27	2.13	2.31	-	-	Addition of ½ sample weight
28	2.26	2.54	-	-	

Interpretation: Final pH is less than 3.5 indicating a strong possibility for production of acid waters through a microbiological action.

TABLE H-2

ACID PRODUCING POTENTIAL TEST RESULTS
Zinc Tailing Fines (mixture massive/stringer ore)

Sample: 2

Initial Chemical Test (In Duplicate)	A	B
Initial Slurry pH:	7.68	7.73
Acid Potential		
Total Sulfur Analysis, %:	21.8	22.2
Equivalent H_2SO_4, lb/ton:	1,334	1,358
Acid Consumption		
1N H_2SO_4 titration, ml:	3.304	3.140
Equivalent H_2SO_4, lb/ton:	32	31

Interpretation: Positive acid producing potential is indicated.

Biological Confirmation Test (In Duplicate)

	pH		Fe, mg/l		
Day	A	B	A	B	
0	2.66	2.84	-	-	Inoculation with Thiobacillus ferrooxidans
1	2.77	2.90	123	103	
2	2.86	3.00	128	112	
3	2.86	3.03	135	126	
5	2.80	2.98	126	132	
7	2.60	2.84	136	141	
9	2.50	2.64	210	156	
12	2.31	2.43	345	290	
14	2.27	2.37	400	366	
16	2.23	2.31	446	402	
19	2.20	2.27	482	438	
21	2.22	2.29	512	468	
23	2.24	2.23	547	462	
26	2.20	2.29	-	-	Addition of ½ sample weight
27	2.74	2.91	-	-	Addition of ½ sample weight
28	3.20	3.27	-	-	

Interpretation: Final pH is less than 3.5 indicating a strong possibility for production of acid waters through microbiological action.

TABLE H-3

ACID PRODUCING POTENTIAL TEST RESULTS
Combined Mine Backfill

Sample: 3

Initial Chemical Test (In Duplicate)	A	B
Initial Slurry pH:	4.77	4.63
Acid Potential		
Total Sulfur Analysis, %:	24.3	24.5
Equivalent H_2SO_4, lb/ton:	1,486	1,499
Acid Consumption		
1N H_2SO_4 titration, ml:	1.188	1.368
Equivalent H_2SO_4, lb/ton:	12	13

Interpretation: Positive acid producing potential is indicated.

Biological Confirmation Test (In Duplicate)

	pH		Fe, mg/l		
Day	A	B	A	B	
0	2.68	2.58	-	-	Inoculation with Thiobacillus ferrooxidans
1	2.66	2.50	-	79	
2	2.69	2.56	62	71	
3	2.69	2.55	63	75	
5	2.63	2.53	-	-	
7	2.47	2.36	-	-	
9	2.43	2.32	-	-	
12	2.38	2.29	93	103	
14	2.38	2.29	117	130	
16	2.37	2.29	133	148	
19	2.34	2.31	138	151	
21	2.37	2.34	160	169	
23	2.40	2.40	178	182	
26	2.38	2.38	-	-	Addition of ½ sample weight
27	2.81	2.77	-	-	Addition of ½ sample weight
28	3.29	3.22	-	-	

Interpretation: Final pH is less than 3.5 indicating a strong possibility for production of acid waters through microbiological action.

TABLE H-4

ACID PRODUCING POTENTIAL TEST RESULTS
Unclassified Zinc Tailings (massive ore)

Sample: 4

Initial Chemical Test (In Duplicate)	A	B
Initial Slurry pH:	5.77	5.96
Acid Potential		
Total Sulfur Analysis, %:	31.2	31.1
Equivalent H_2SO_4, lb/ton:	1,909	1,902
Acid Consumption		
1N H_2SO_4 titration, ml:	2.558	2.309
Equivalent H_2SO_4, lb/ton:	25	23

Interpretation: Positive acid producing potential is indicated.

Biological Confirmation Test (In Duplicate)

	pH		Fe, mg/l		
Day	A	B	A	B	
0	2.61	2.71	-	-	Inoculation with Thiobacillus ferrooxidans
1	2.58	2.70	76	65	
2	2.62	2.72	71	65	
3	2.61	2.73	79	63	
5	2.50	2.68	-	65	
7	2.34	2.43	-	-	
9	2.27	2.36	105	-	
12	2.20	2.25	193	162	
14	2.18	2.21	231	214	
16	2.13	2.16	263	246	
19	2.10	2.13	285	276	
21	2.11	2.16	337	299	
23	2.14	2.18	358	322	
26	2.11	2.16	-	-	Addition of ½ sample weight
27	2.30	2.36	-	-	Addition of ½ sample weight
28	2.49	2.65	-	-	

Interpretation: Final pH is less than 3.5 indicating a strong possibility for production of acid waters through microbiological action.

TABLE H-5

ACID PRODUCING POTENTIAL TEST RESULTS
Pyrite Concentrate Fines (mixture massive/stringer ore)

Sample: 5

Initial Chemical Test (In Duplicate)	A	B
Initial Slurry pH:	6.05	5.98
Acid Potential		
Total Sulfur Analysis, %:	41.3	40.9
Equivalent H_2SO_4, lb/ton:	2,526	2,502
Acid Consumption		
1N H_2SO_4 titration, ml:	1.495	1.494
Equivalent H_2SO_4, lb/ton:	15	15

Interpretation: Positive acid producing potential is indicated.

Biological Confirmation Test (In Duplicate)

	pH		Fe, mg/l		
Day	A	B	A	B	
0	2.57	2.54	-	-	Inoculation with Thiobacillus ferrooxidans
1	2.55	2.50	108	118	
2	2.62	2.55	109	118	
3	2.59	2.55	112	120	
5	2.50	2.44	-	-	
7	2.42	2.36	-	-	
9	2.38	2.32	-	-	
12	2.30	2.25	122	145	
14	2.28	2.23	152	168	
16	2.24	2.19	159	191	
19	2.20	2.16	177	205	
21	2.20	2.18	201	217	
23	2.20	2.13	231	243	
26	2.14	2.16	-	-	Addition of ½ sample weight
27	2.27	2.32	-	-	Addition of ½ sample weight
28	2.41	2.47	-	-	

Interpretation: Final pH is less than 3.5 indicating a strong possibility for production of acid waters through microbiological action.

TABLE H-6

ACID PRODUCING POTENTIAL TEST RESULTS
Pyrite Tailings Fines (mixture massive/stringer ore)

Sample: 6

Initial Chemical Test (In Duplicate)	A	B
Initial Slurry pH:	8.82	9.07
Acid Potential		
Total Sulfur Analysis, %:	1.83	1.88
Equivalent H_2SO_4, lb/ton:	112	115
Acid Consumption		
1N H_2SO_4 titration, ml:	5.225	5.509
Equivalent H_2SO_4, lb/ton:	51	54

Interpretation: A much lower potential is indicated for producing acid compared to the other five materials.

Biological Confirmation Test (In Duplicate)

	pH		Fe, mg/l		
Day	A	B	A	B	
0	2.68	2.67	-	-	Inoculation with Thiobacillus ferrooxidans
1	2.89	2.57	1,360	982	
3	2.47	2.43	1,220	1,330	
6	2.38	2.36	1,380	1,550	
8	2.38	2.35	1,580	1,730	
10	2.39	2.34	1,740	1,920	
13	2.36	2.28	2,070	2,310	
15	2.38	2.28	1,990	2,270	
17	2.38	2.25	1,870	2,260	
20	2.34	2.27	1,950	2,270	
22	2.39	2.21	-	-	Addition of ½ sample weight
23	3.63	3.20	-	-	Addition of ½ sample weight
24	4.29	3.90	-	-	

Interpretation: The initial chemical test is not confirmed. Final pH is greater than 3.5 indicating that acid waters would not be produced from this material.

APPENDIX I

AGITATED LEACH TESTS ON TAILINGS SAMPLES

Agitated leaching tests were conducted on the tailings samples following a method developed by Committee D-19 on Water of the American Society of Testing Materials (ASTM) and published as Method A-Water-Extraction Procedure under "Proposed Methods for Leaching of Waste Materials". One modification made in the method was the use of the USEPA extraction procedure "rotary extractor" apparatus for agitation. Thus, agitation for these leaching tests was in parallel with the EPA extraction procedure tests detailed in Appendix F.

Three sets of tests were conducted. In one set, the leach solution was deionized water; in the second set, a sulfuric acid solution was used; and in the third set, the samples were contacted with an alkaline calcium hydroxide solution. Both the deionized water leaching tests and the acidic-solution leach tests were conducted thru three cycles on all six samples with each cycle of leaching covering 48 hours. The alkaline-solution leach tests were conducted on three samples -- No. 2 (Zinc Tailings Fines, mixture massive/stringer ore), No. 3 (Combined Mine Backfill), and No. 5 (Pyrite Concentrate Fines, mixture massive/stringer ore) with one 48-hour period of leaching.

Analysis data for the deionized water leach tests are given in Tables I-1 and I-2. Figures I-1 thru I-6 provide procedural information on this set of tests.

Analysis data for the sulfuric acid solution leach tests are given in Tables I-3 and I-4. Procedural information for this set of tests is shown in Figures I-7 thru I-12. Table I-5 provides data on sulfuric acid input requirements for adjustment of leach slurries to a pH of 3.0.

Test conditions and leachate analysis data for the tests conducted under alkaline conditions are given in Table I-6.

Analytical methods used and limits of detection for the parameters determined on the leachates from these three sets of tests are given in Table I-7.

TABLE I-1

LEACHATE ANALYSIS RESULTS
MULTI-CYCLE AGITATED WATER-LEACHING
OF TAILINGS SAMPLES

Parameter	Cycle	SAMPLE NO. 1	2	3	4	5	6
pH	1	5.80	6.35	6.55	5.90	4.90	6.60
	2	5.65	6.00	6.15	5.85	4.30	6.60
	3	6.00	6.20	6.15	6.00	5.15	6.50
Conductivity mmhos/cm	1	2.20	1.97	0.81	2.03	2.02	1.56
	2	1.14	0.58	0.34	0.98	0.54	0.49
	3	0.43	0.29	0.27	0.41	0.31	0.31
Cu mg/1	1	0.04	0.05	0.05	0.02	7.5	0.03
	2	<0.03	<0.03	<0.03	<0.03	0.10	<0.03
	3	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Fe mg/1	1	0.12	0.19	<0.02	0.56	56	<0.02
	2	<0.02	1.59	<0.02	<0.02	44	<0.02
	3	<0.02	10.5	<0.02	<0.02	47	<0.02
Pb mg/1	1	0.24	0.10	<0.03	0.23	2.66	<0.03
	2	0.06	0.03	<0.03	0.03	0.96	<0.03
	3	<0.05	<0.03	<0.05	<0.05	0.05	<0.05
Zn mg/1	1	271	85	15	258	575	0.52
	2	25	8.9	2.5	20	50	0.04
	3	6.7	0.44	1.7	5.2	21	0.02
$\overset{=}{\mathrm{SO}^{4}}$ mg/1	1	2,199	1,603	425	2,156	2,311	1,025
	2	708	209	84	573	270	150
	3	186	91	77	159	144	69

Sample No.	Description
1	Zinc Tailings Fines (massive ore)
2	Zinc Tailings Fines (mixture massive/stringer ore)
3	Combined Mine Backfill
4	Unclassified Zinc Tailings (massive ore)
5	Pyrite Concentrate Fines (mixture massive/stringer ore)
6	Pyrite Tailings Fines (mixture massive/stringer ore)

TABLE I-2

ANALYSIS RESULTS
LEACHATE COMPOSITES FROM 3 CYCLES OF WATER-LEACHING OF TAILINGS SAMPLES

	SAMPLE NO.					
Parameter[1]	1	2	3	4	5	6
As	0.14	<0.01	0.04	0.17	0.70	<0.01
Ba	<0.1	<0.1	<0.1	<0.1	0.1	0.1
Cd	0.26	0.044	0.018	0.22	0.63	<0.005
Cr, total	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
[2]Cu	<0.02	<0.02	<0.02	<0.02	2.5	<0.02
[2]Fe	0.04	4.1	<0.02	0.19	49	<0.02
[2]Pb	0.10	0.04	<0.03	0.08	1.2	<0.03
Mn	11.7	4.4	7.0	10.4	8.1	0.26
Hg	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	0.0006
Se	0.17	0.09	0.12	0.21	0.10	0.25
Ag	<0.01	<0.01	<0.01	<0.01	<0.01	0.01
[2]Zn	101	31	6	94	215	0.20
Cl	<0.5	1.6	<0.5	<0.5	<0.5	<0.5
F	0.18	0.30	0.10	0.17	<0.04	0.31
NO_3	<0.05	0.18	<0.05	0.09	0.35	<0.05
[2]SO_4	1030	630	195	960	910	410
[2]pH	5.8	6.3	6.3	5.9	4.6	6.6
[2]Conductivity, mmhos/cm	1.3	0.9	0.5	1.1	1.0	0.8
Color, Units	1	1	1	3	2	1

[1] All values mg/l except conductivity and color.

[2] Values calculated from analysis results on cycle leachates.

FIGURE I-1

AGITATED LEACHING PROCEDURE

Sample Designation Zinc Tailings Fines (massive ore), Sample No. 1

Project No. J10529 Operator M. Bishop Date 7-4-81

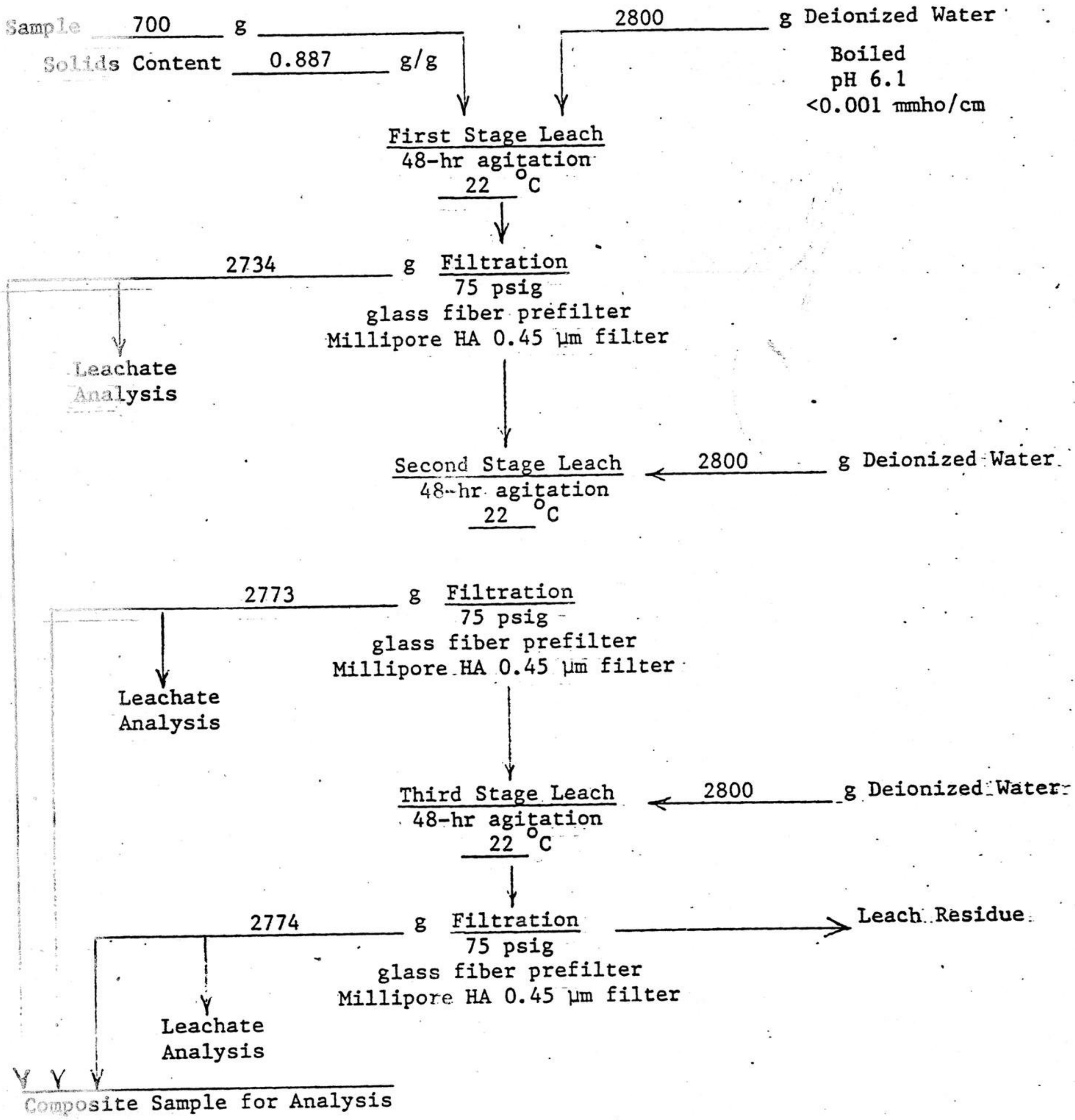


FIGURE I-2

AGITATED LEACHING PROCEDURE

Sample Designation Zinc Tailings Fines (mixture massive/stringer ore), Sample No. 2

Project No. J10529 Operator M. Bishop Date 7-4-81

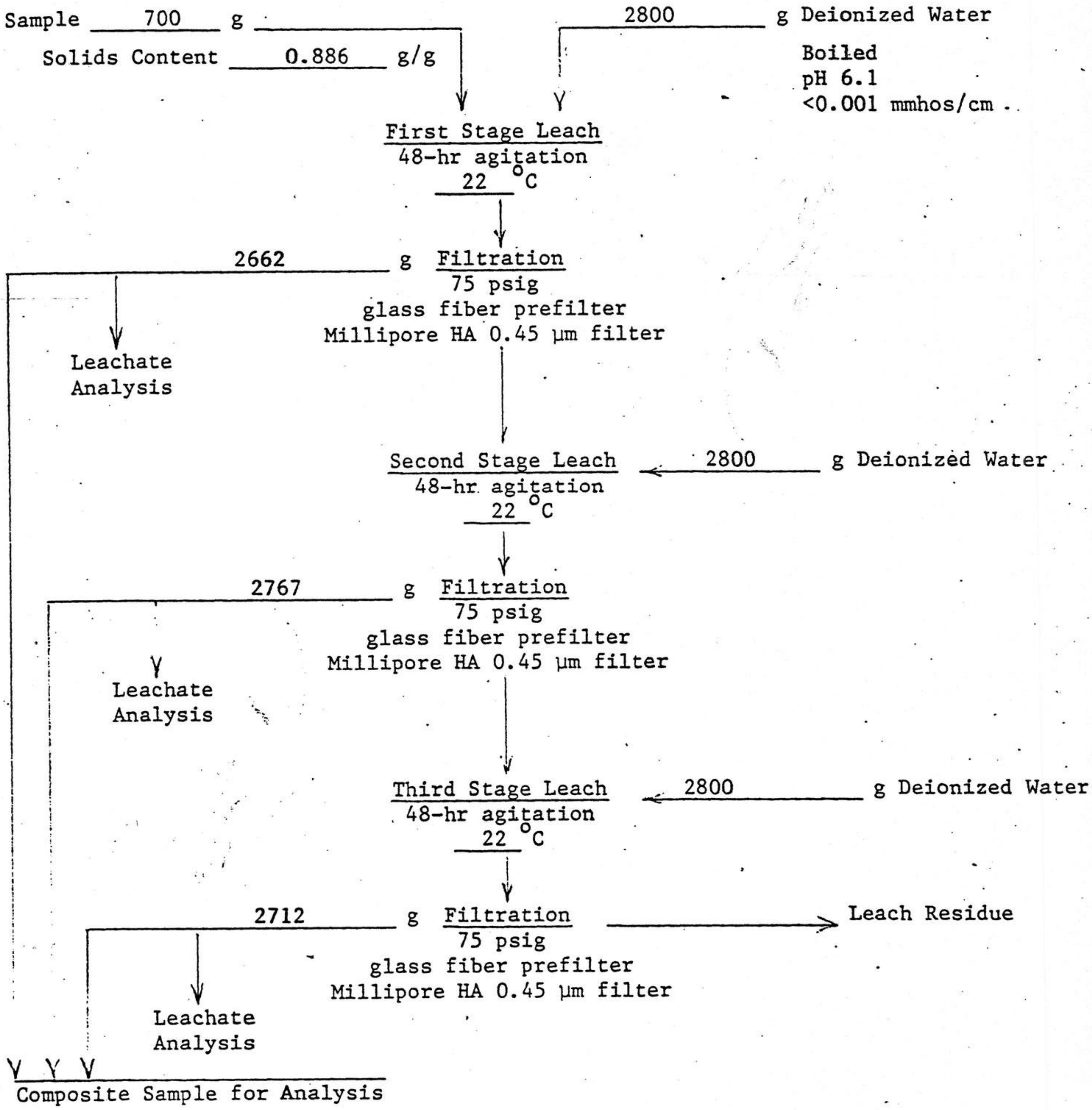


FIGURE I-3

AGITATED LEACHING PROCEDURE

Sample Designation Combined Mine Backfill, Sample No. 3

Project No. J10529 Operator M. Bishop Date 7-4-81

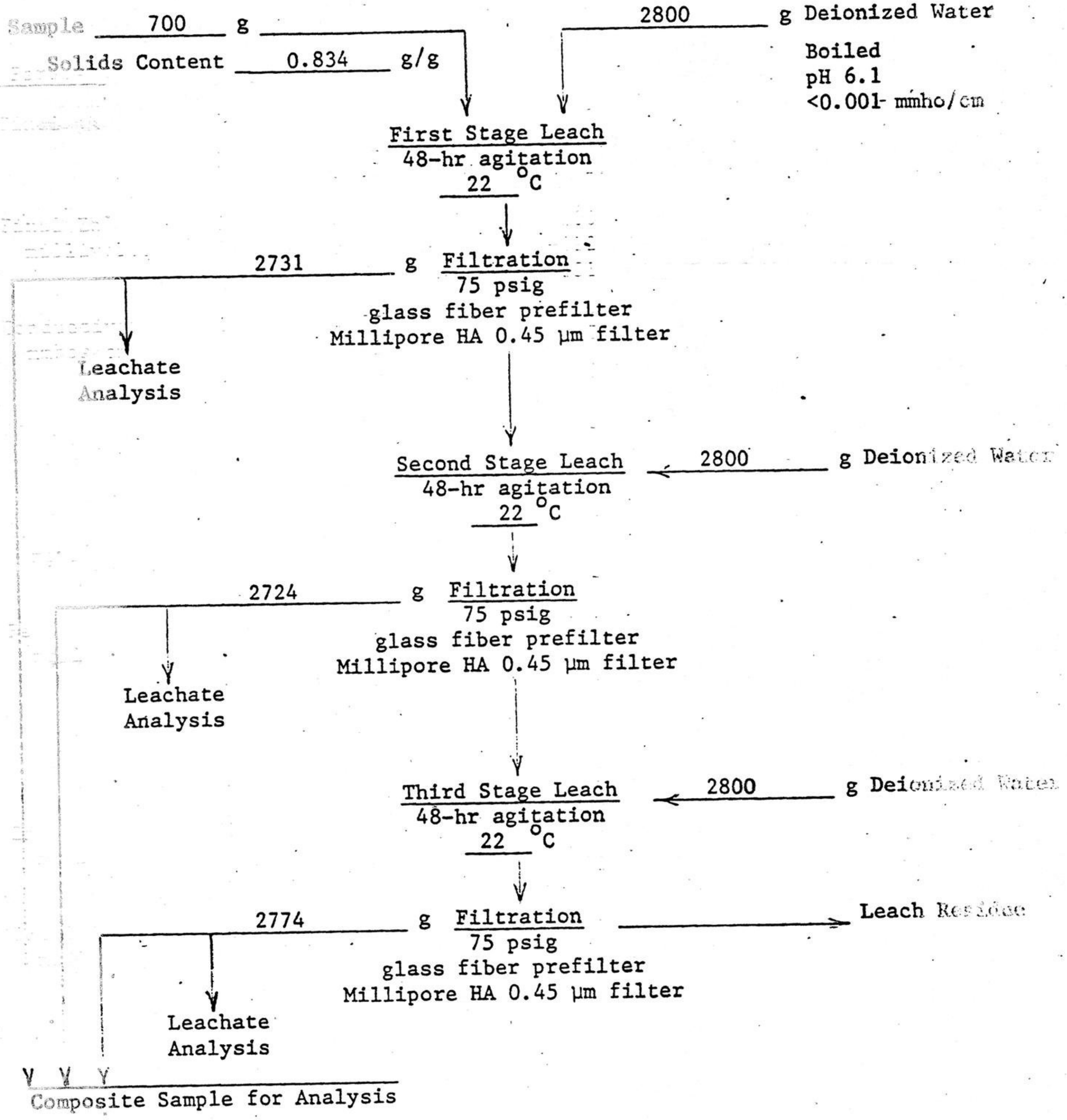
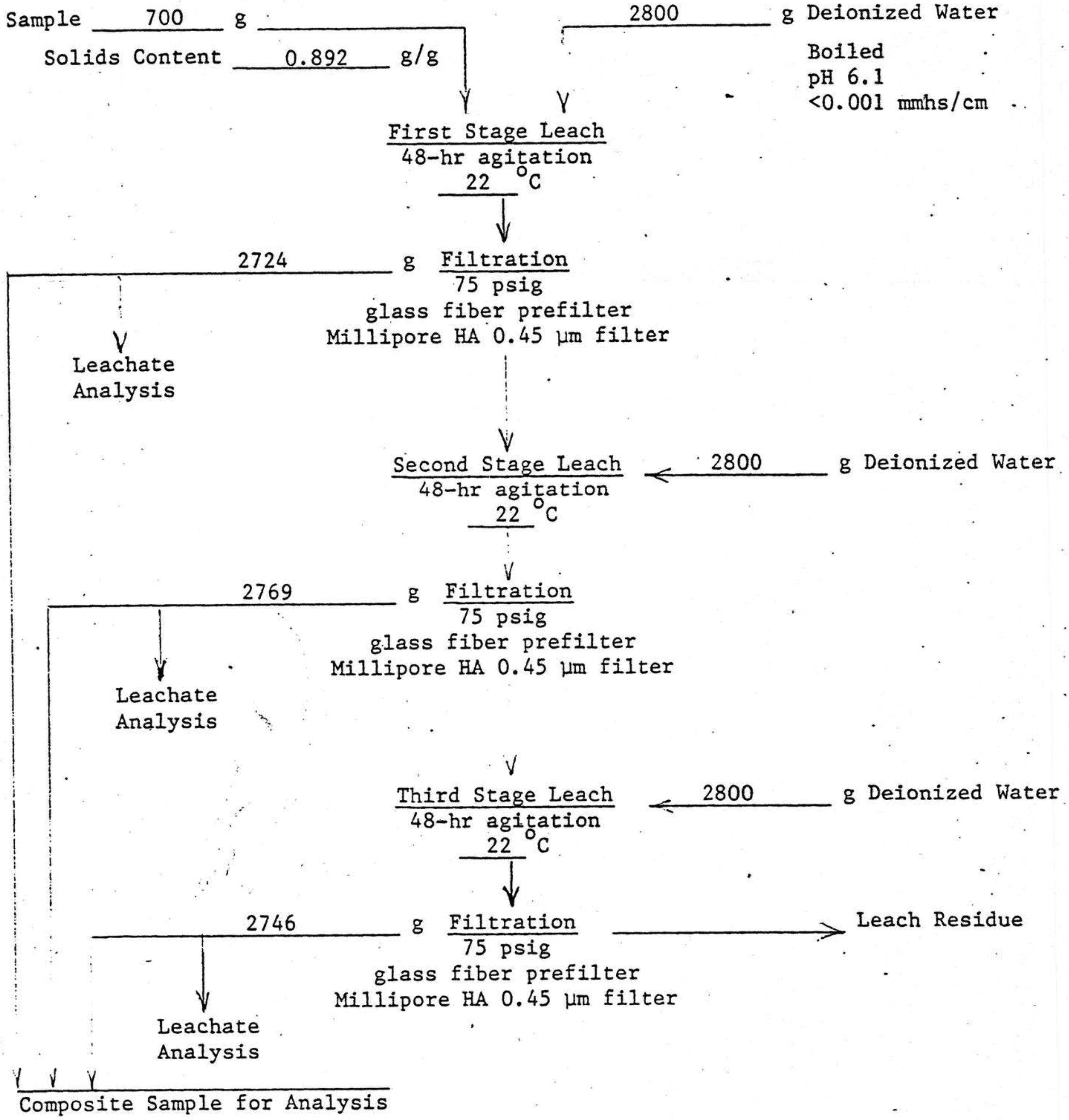


FIGURE I-4

AGITATED LEACHING PROCEDURE

Sample Designation Unclassified Zinc Tailings (massive ore), Sample No. 4

Project No. J10529 Operator M. Bishop Date 7-4-81



Sample 700 g

Solids Content 0.892 g/g

2800 g Deionized Water

Boiled
pH 6.1
<0.001 mmhs/cm

First Stage Leach
48-hr agitation
22 °C

2724 g Filtration
75 psig
glass fiber prefilter
Millipore HA 0.45 µm filter

Leachate Analysis

Second Stage Leach
48-hr agitation
22 °C

2800 g Deionized Water

2769 g Filtration
75 psig
glass fiber prefilter
Millipore HA 0.45 µm filter

Leachate Analysis

Third Stage Leach
48-hr agitation
22 °C

2800 g Deionized Water

2746 g Filtration
75 psig
glass fiber prefilter
Millipore HA 0.45 µm filter

Leach Residue

Leachate Analysis

Composite Sample for Analysis

FIGURE I-5

AGITATED LEACHING PROCEDURE

Sample Designation Pyrite Concentrate Fines (mixture massive/stringer ore) Sample No. 5

Project No. J10529 Operator M. Bishop Date 7-4-81

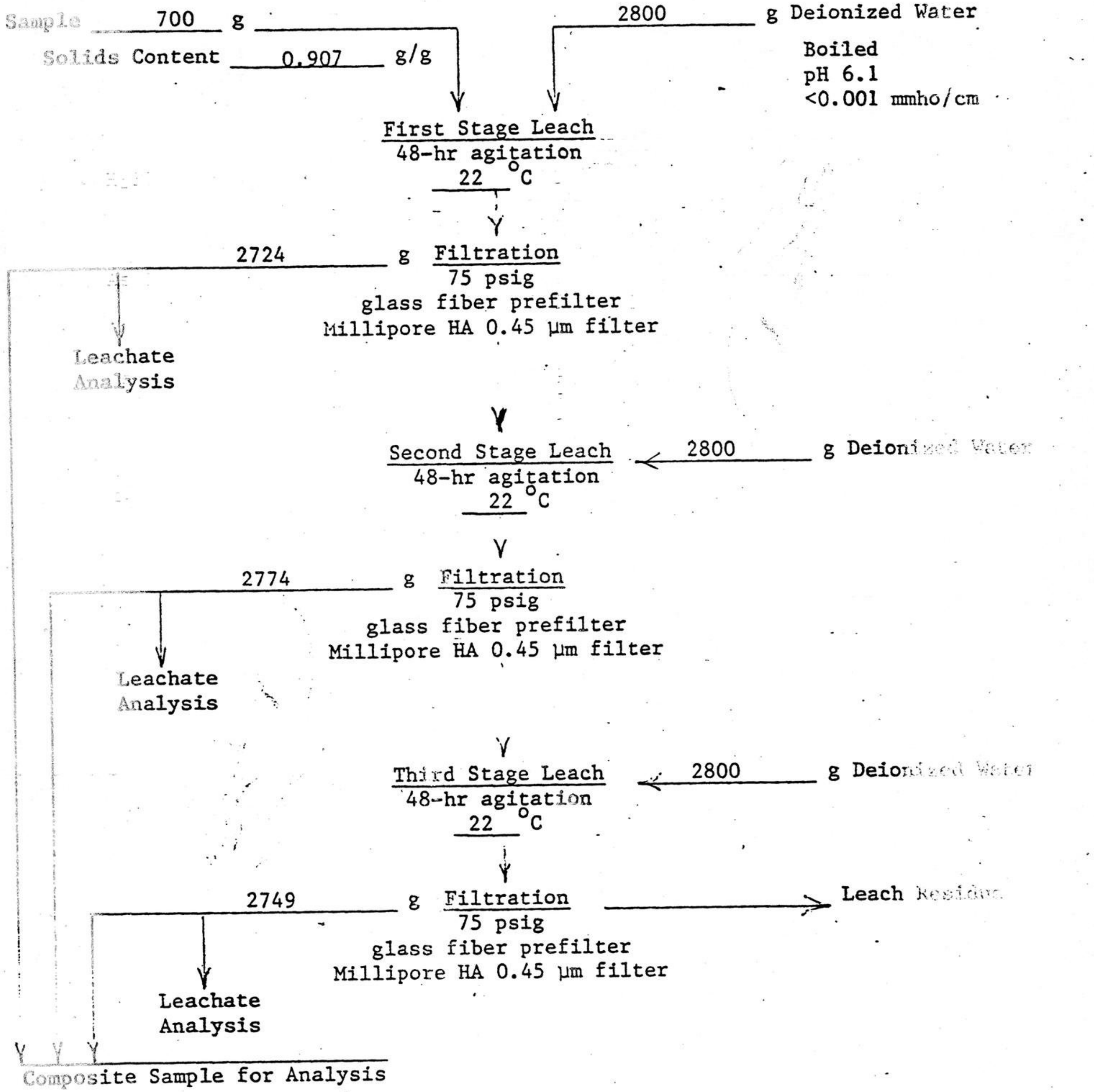


FIGURE I-6

AGITATED LEACHING PROCEDURE

Sample Designation Pyrite Tailing Fines (mixture massive/stringer ore), Sample No. 6

Project No. J10529 Operator M. Bishop Date 7-4-81

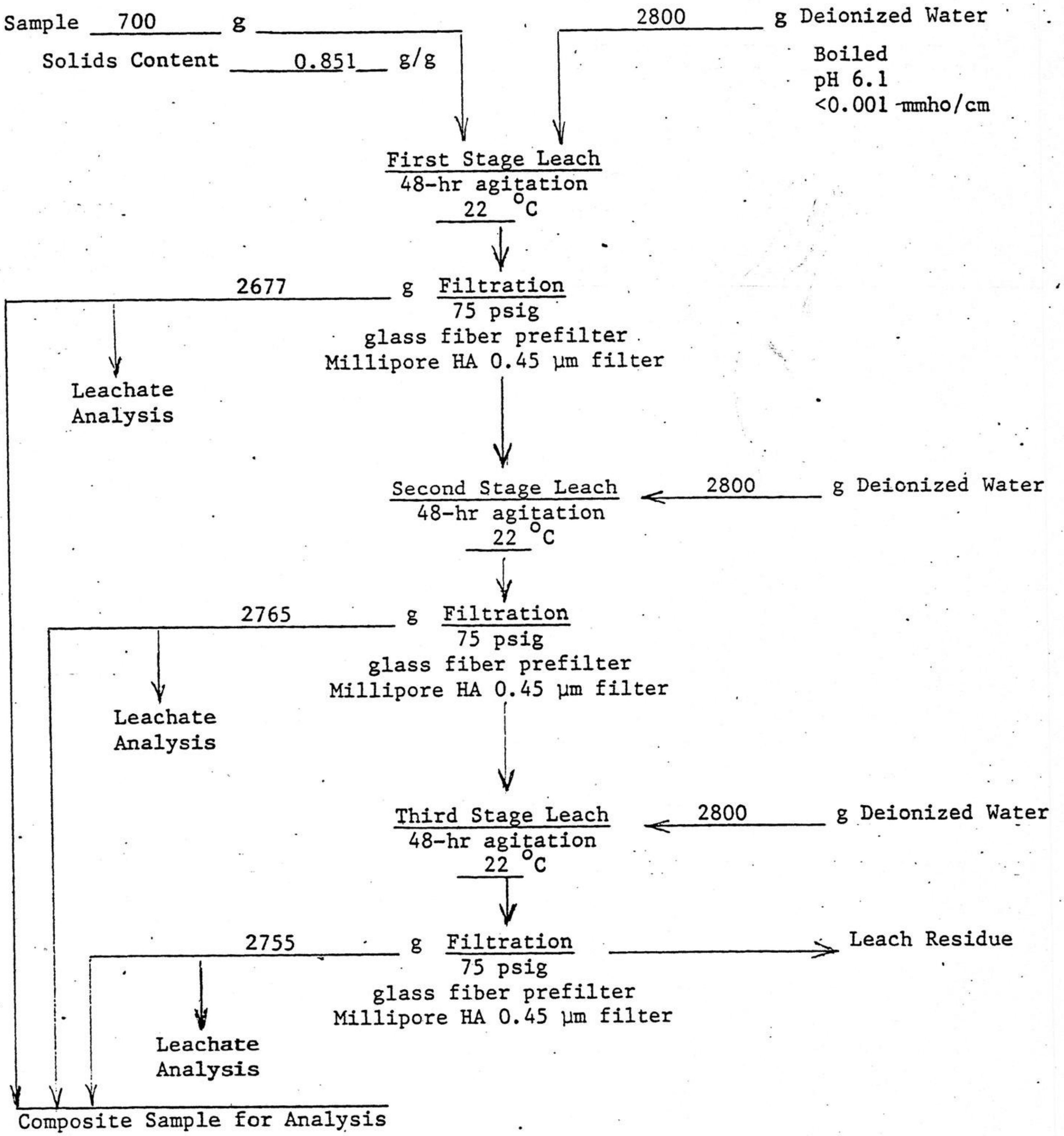


TABLE I-3

LEACHATE ANALYSIS RESULTS MULTI-CYCLE AGITATED LEACHING OF TAILINGS SAMPLES UNDER ACID CONDITIONS

		Sample No.					
Parameter	Leach Stage	1	2	3	4	5	6
Final pH	1	3.15	2.85	3.27	3.48	3.33	3.84
	2	3.02	3.14	3.09	3.31	3.55	3.98
	3	3.07	3.10	2.94	3.35	3.45	4.07
Final Eh millivolts	1	-284	-318	-300	-272	-264	-247
	2	-293	-297	-312	-286	-269	-266
	3	-289	-301	-331	-281	-264	-263
Conductivity mmhos/cm	1	4.87	6.30	2.59	3.66	3.28	3.31
	2	2.72	2.93	0.80	2.21	1.26	2.85
	3	1.06	1.28	0.73	0.99	0.89	1.28
As mg/1	1	1.7	0.5	3.4	1.8	3.0	0.1
	2	0.8	0.2	1.7	1.0	1.0	0.04
	3	0.5	0.07	1.4	0.6	0.5	0.02
Cu mg/1	1	121	244	10	57	98	39
	2	19	28	4	29	35	10
	3	6	12	4	10	18	4
Fe mg/1	1	403	996	701	123	363	159
	2	184	241	57	132	201	59
	3	156	168	40	105	132	24
Pb mg/1	1	<0.02	<0.02	3.3	3.0	3.1	2.7
	2	<0.02	<0.02	5.0	2.8	3.6	3.0
	3	<0.02	<0.02	5.1	4.0	4.1	3.1
Zn mg/1	1	962	1,000	90	336	393	103
	2	78	78	11	105	130	17
	3	27	14	7	76	78	6
[1]Total Sulfur mg/1	1	1,530	2,100	637	1,010	933	897
	2	633	715	111	513	267	766
	3	164	214	69	173	165	295
[1]SO_4 mg/1	1	4,500	6,260	1,910	2,980	2,780	2,760
	2	1,880	2,160	324	1,550	779	2,250
	3	477	627	198	507	481	880

[1] Includes that added to formulate initial pH 3.0 leach solution and that added during leach stages for pH control.

TABLE I-4

ANALYSIS RESULTS
LEACHATE COMPOSITES FROM 3 CYCLES OF LEACHING TAILINGS SAMPLES UNDER ACID CONDITIONS

Parameter[1]	Sample No. 1	2	3	4	5	6
[2]As	1.0	0.3	2.2	1.1	1.5	0.06
Ba	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Cd	1.0	1.1	0.1	0.5	0.6	0.1
Cr, total	0.07	0.10	0.03	0.03	0.05	0.02
[2]Cu	49	95	6	32	50	18
[2]Fe	248	468	266	120	232	81
[2]Pb	<0.02	<0.02	4.5	3.3	3.6	2.9
Mn	27	24	13	19	12	19
Hg	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002
Se	0.003	0.004	0.008	0.022	0.008	0.005
Ag	0.01	0.01	0.01	0.01	0.01	0.01
[2]Zn	356	364	36	172	200	42
Cl	0.6	0.6	0.8	0.6	0.6	8.6
F	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
NO_3	<0.4	<0.4	1.2	2.3	2.6	2.6
[2]SO_4	2,285	3,015	810	1,680	1,347	1,963
[2]Total S	776	1,010	272	565	455	653
pH	3.06	3.01	3.10	3.40	3.54	3.93
Eh, mv	-288	-316	-307	-279	-274	-258
Conductivity, mmhos/cm	3.11	3.78	1.44	2.33	1.98	2.61
Color, Units	0	1	0	1	1	1

[1] All values mg/l except pH, Eh, conductivity, and color.
[2] Values calculated from analysis results on cycle leachates.

TABLE I-5

H_2SO_4 INPUT TO LEACH STAGES[1,2] MULTI-CYCLE AGITATED LEACHING OF TAILINGS SAMPLES UNDER ACID CONDITIONS

	Leach Stage	Sample No.					
		1	2	3	4	5	6
H_2SO_4, mg	1	183	183	301	1,359	1,065	1,594
	2	595	536	477	1,418	1,241	2,182
	3	447	712	501	830	830	1,359
As Sulfur, mg	1	60	60	98	444	348	520
	2	194	175	156	463	405	712
	3	146	232	164	271	271	444
As Sulfate, mg	1	179	179	295	1,331	1,043	1,561
	2	583	525	467	1,389	1,216	2,137
	3	438	697	491	813	813	1,331

[1] Includes H_2SO_4 in leach solutions plus H_2SO_4 added for pH control

[2] Volume of leach solution to each stage = 2.8ℓ

FIGURE I-7

ACID CONDITIONS
AGITATED LEACHING PROCEDURE

Sample Designation Zinc Tailings Fines (massive ore), Sample No. 1

Project No. J10529 Operator M. Bishop Date 2/2/82 thru 2/8/82

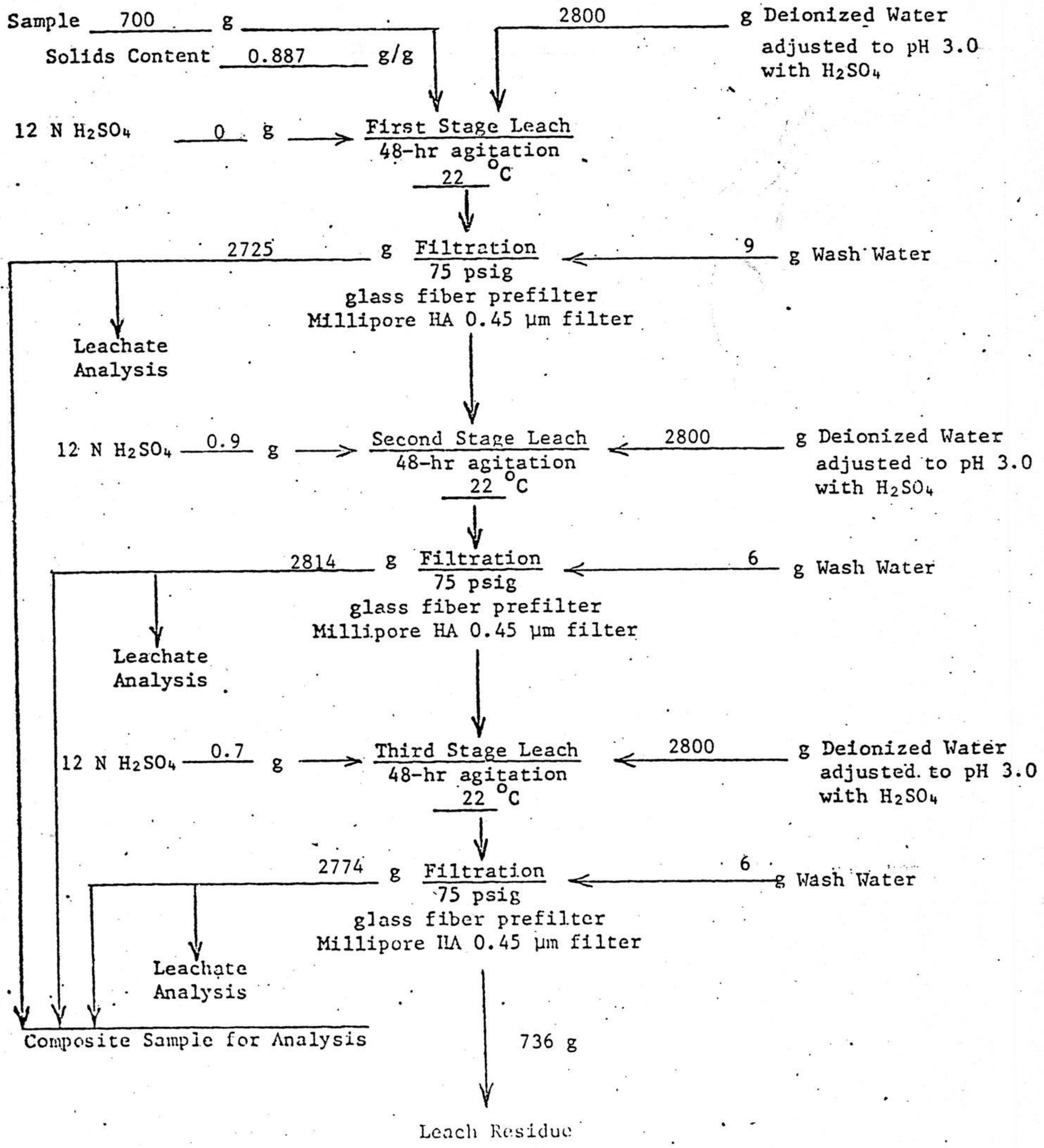


FIGURE I-8

ACID CONDITIONS
AGITATED LEACHING PROCEDURE

Sample Designation Zinc Tailings Fines (mixture massive/stringer ore), Sample No. 2

Project No. J10529 Operator M. Bishop Date 2/2/82 thru 2/8/82

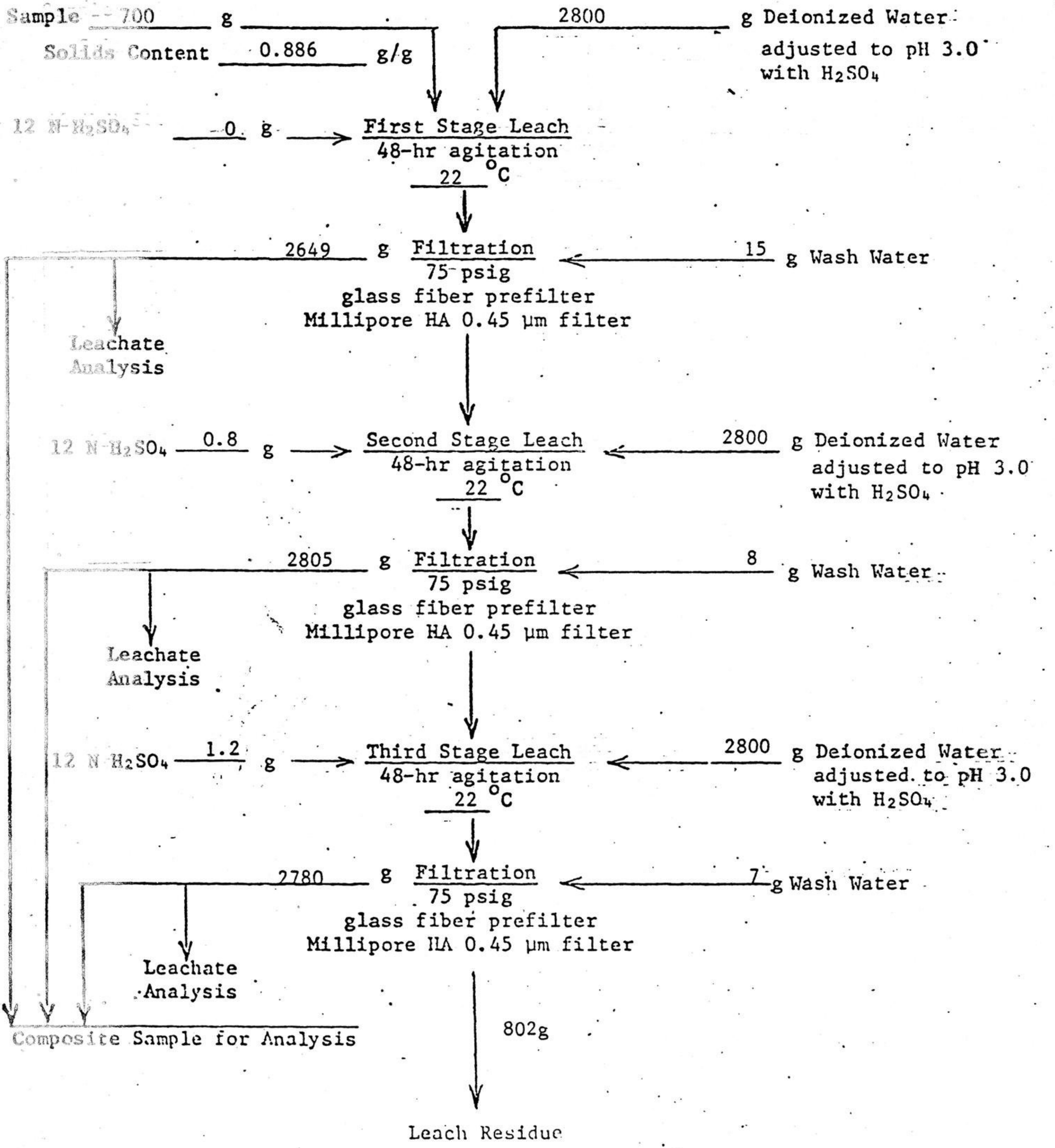


FIGURE I-9

ACID CONDITIONS
AGITATED LEACHING PROCEDURE

Sample Designation Combined Mine Backfill, Sample No. 3

Project No. J10529 Operator M. Bishop Date 2/2/82 thru 2/8/82

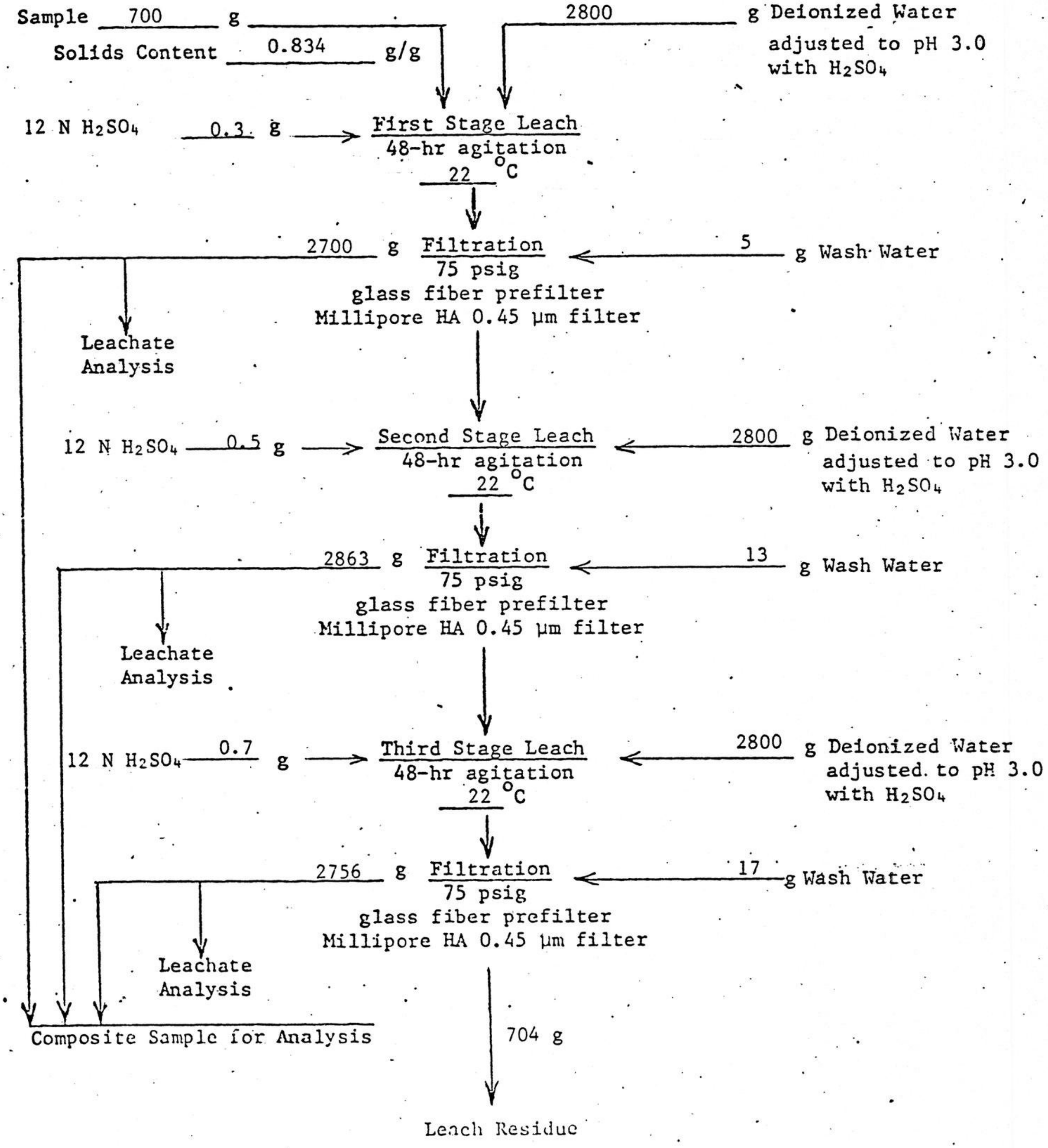


FIGURE I-10

ACID CONDITIONS
AGITATED LEACHING PROCEDURE

Sample Designation Unclassified Zinc Tailings (massive ore), Sample No. 4

Project No. J10529 Operator M. Bishop Date 2/2/82 thru 2/8/82

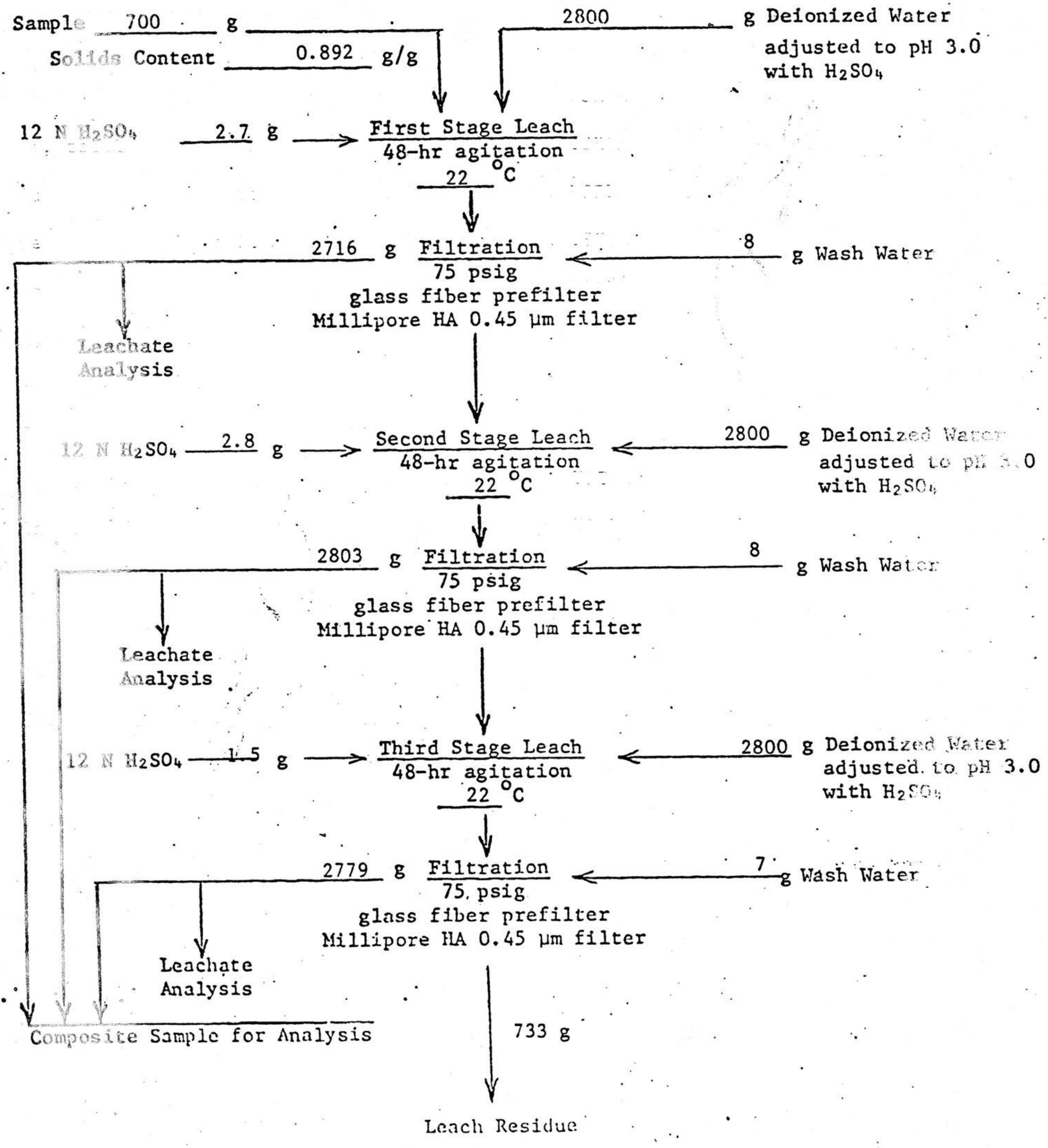


FIGURE I-11

ACID CONDITIONS
AGITATED LEACHING PROCEDURE

Sample Designation Pyrite Concentrate Fines (mixture massive/stringer ore), Sample No. 5

Project No. J10529 Operator M. Bishop Date 2/2/82 thru 2/8/82

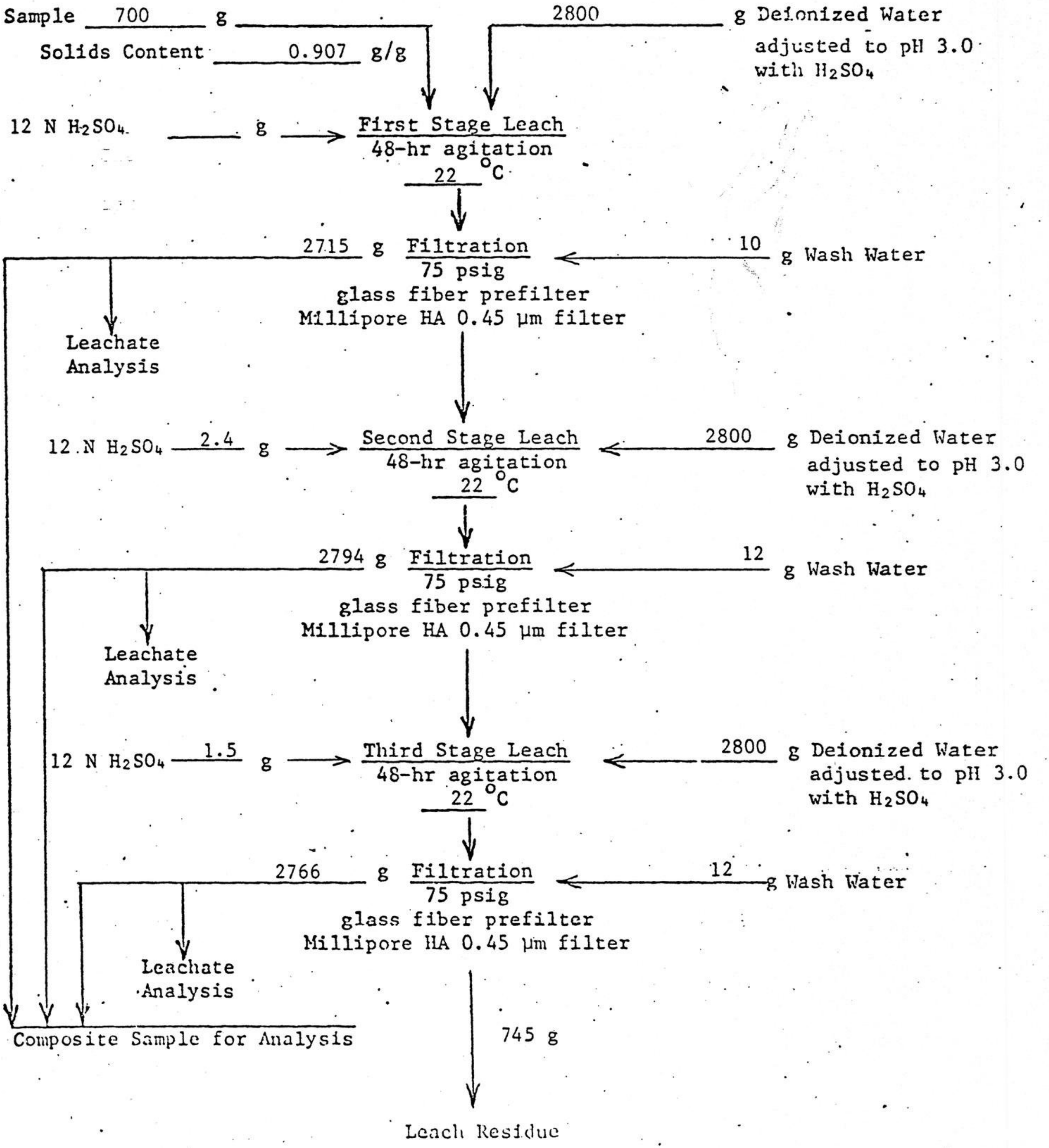


FIGURE I-12

ACID CONDITIONS
AGITATED LEACHING PROCEDURE

Sample Designation Pyrite Tailings Fines (mixture massive/stringer ore), Sample No. 6

Project No. J10529 Operator M. Bishop Date 2/2/82 thru 2/8/82

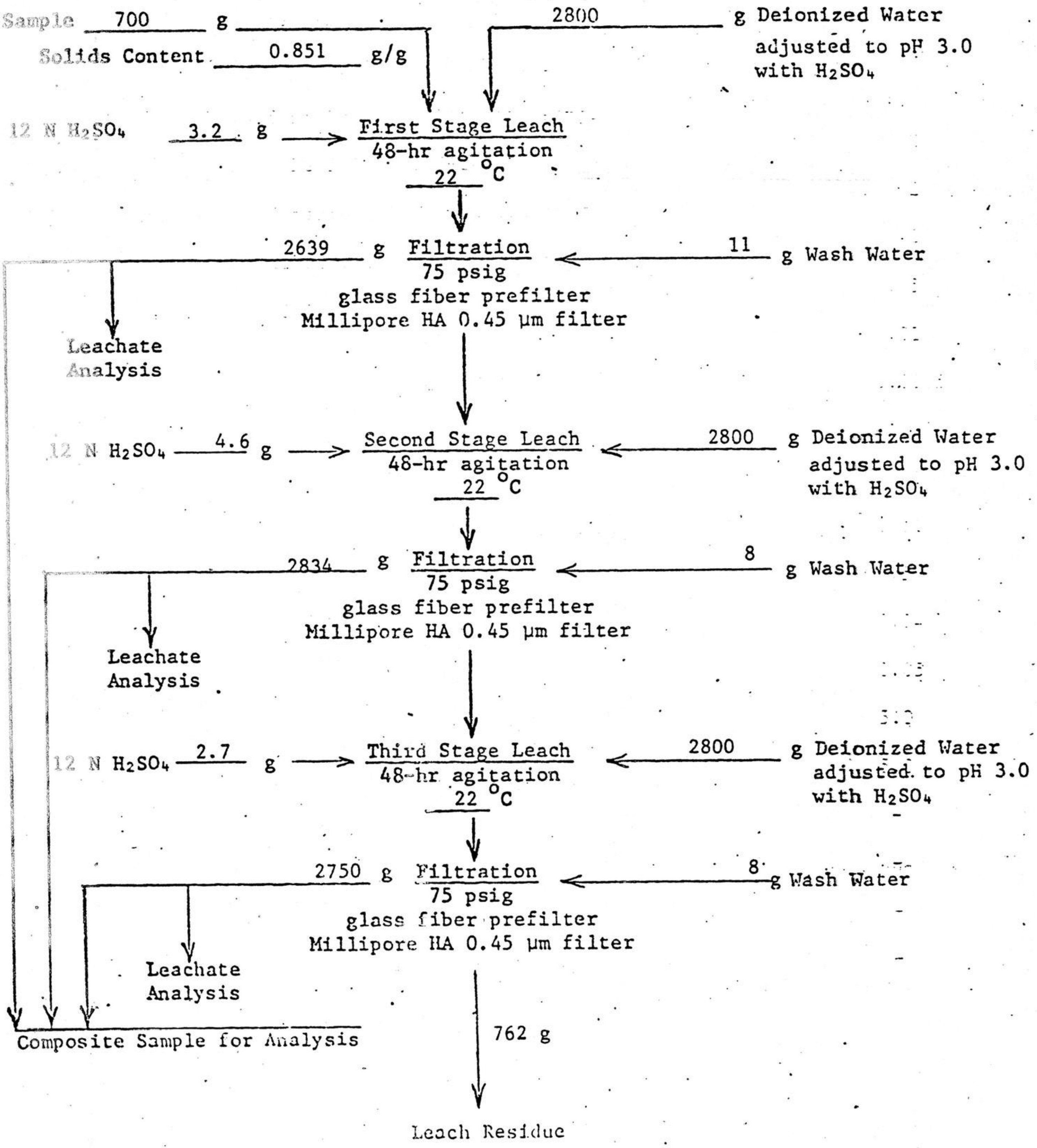


TABLE I-6

AGITATED LEACHING OF TAILINGS UNDER ALKALINE CONDITIONS

		Sample		
		2	3	5
Sample Wt., g		700	700	700
% Moisture		11.4	16.6	9.3
D.I. Water Wt., g		540	465	570
Wt. % Solids		50	50	50
Ca(OH)$_2$ addition, g		1.9	0.8	5.0
Leach time, hrs		48	48	48
Leachate Analyses	**Hours**			
pH	0	10.01	10.21	10.40
	1	9.49	9.93	10.10
	4	8.96	9.28	9.29
	24	8.04	9.03	8.75
	48	6.48	8.02	7.46
emf, mv[1]	0	-	-	-
	1	+110	+112	+125
	4	+109	+108	+108
	24	-125	-102	-102
	48	-70	-125	-128
S_2O_3,$^=$, mg/l	0	-	-	-
	1	288	150	230
	4	255	250	388
	24	50	255	138
	48	9	25	21
Cu, mg/l	48	0.07	0.02	0.08
Fe, mg/l	48	<0.02	<0.02	<0.02
Pb, mg/l	48	<0.05	<0.05	<0.05
Zn, mg/l	48	89	0.19	8.3
SO_4, mg/l	48	2640	2320	2630

Sample No.	Description
2	Zinc Tailings Fines (mixture massive/stringer ore)
3	Combined Mine Backfill
5	Pyrite Concentrate Fines (mixture massive/stringer ore)

---

[1] Platinum electrode:calomel reference electrode

TABLE I-7

ANALYTICAL METHODS USED AND LIMITS OF DETECTION ON LEACHATES FOR PARAMETERS LISTED IN TABLES

Parameter	Methods	Detection Limit, mg/l
As	Graphite Furnace - AAS, USEPA[1]	0.01
Ba	Atomic Absorption Spectroscopy, USEPA	0.1
Cd	Atomic Absorption Spectroscopy, USEPA	0.005
Cr, total	Atomic Absorption Spectroscopy, USEPA	0.02
Cu	Atomic Absorption Spectroscopy, USEPA	0.02
Fe	Atomic Absorption Spectroscopy, USEPA	0.02
Pb	Atomic Absorption Spectroscopy, USEPA	0.05
Mn	Atomic Absorption Spectroscopy, USEPA	0.01
Hg	Cold Vapor, Flameless AAS, USEPA	0.0005
Se	Graphite Furnace, AAS, USEPA	0.003
Ag	Atomic Absorption Spectroscopy, USEPA	0.01
Zn	Atomic Absorption Spectroscopy, USEPA	0.01
Cl	Colorimetric, USEPA	0.5
F	Specific Ion Electrode - Potentiometric, USEPA	0.04
NO_3	Colorimetric, USEPA	0.05
SO_4	Gravimetric, USEPA	5.0
S_2O_3	Colorimetric[2]	5.0
pH	Electrometric, APHA[3]	-
Eh	Platinum Electrode	-
Conductivity	Bridge, APHA	-
Color	Chloroplantinate Visual Comparator, APHA	-

[1] USEPA, 1979, Methods for Chemical Analysis of Water and Wastes, EPA-600/4-79-020.

[2] Rolia, E., Barbeau, F., 1980, Estimation of individual thio-salts in flotation mill solutions, Talunta, Vol. 27, p. 596-598.

[3] APHA, 1975, Standard Methods for the Examination of Water and Wastewater, 14th Edition.

APPENDIX J

COLUMN SEEPAGE STUDIES ON TAILINGS SAMPLES

Column seepage studies were conducted on three tailings samples. These samples were:

No. 1	Zinc Tailings Fines (massive ore)
No. 2	Zinc Tailings Fines (mixture massive/stringer ore)
No. 3	Combined Mine Backfill

The objective of these studies was to provide information on the characteristics of seepage water from the designated tailings samples and on physical and chemical changes of the tailings samples due to leaching effects.

Four separate conditions for seepage were studied. These were:

- talings submerged under alkaline water (deionized water adjusted to pH 10 with calcium hydroxide).
- tailings submerged under deionized water (pH of 5.6).
- tailings submerged under acidic water (deionized water adjusted to pH 3 with sulfuric acid).
- tailings alternately submerged under deionized water and exposed to the air.

A set of three columns, each containing one of the three tailings samples, were operated for each study condition over a period of 31 weeks. Figure J-1 illustrates the configuration and dimensions of the test columns.

Characteristics of the water used in the study are given in Table J-1.

General parameters and conditions for the operation of the columns are given in Table J-2.

Due to the differing physical characteristics of the tailings studied, and also to some degree due to the effects of the different waters used, drainage times varied among the test columns. In the case of the column set operated under alternate submerged-exposed conditions with low hydraulic head, the relatively permeable combined mine backfill material would drain in a few hours and have exposure to the air for about seven days, the massive ore tailings fines would drain down in 4 to 5 days with exposure of the surface for about 2 to 3 days of the two-week period, while the massive/stringer ore tailings fines would drain more slowly with exposure of the surface to the air for only about 1 to 2 days of every period.

The results of analysis on leachate samples drained from the columns are presented in Table J-3 thru J-14.

The results of permeability measurements carried out on the samples in the study columns near the beginning and at the end of the study period are given in Table J-16.

FIGURE J-1

COLUMN DIMENSIONS
COLUMN SEEPAGE STUDIES

	Sample No.		
	1	2	3
A, cm	72	70	73
B, cm	15	17	14
C, cm	15	18	15

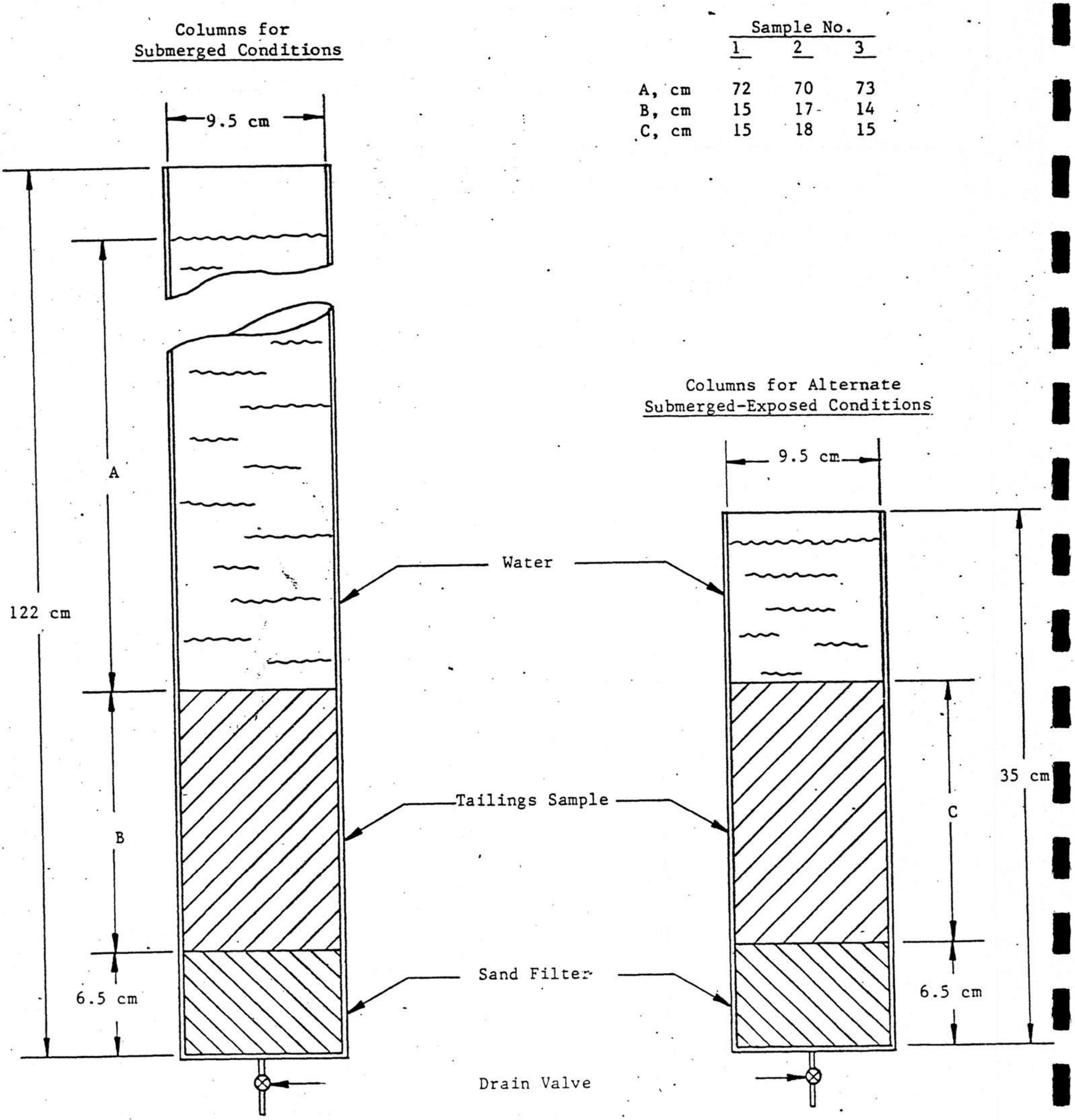


TABLE J-1

CHARACTERISTICS OF WATER USED IN COLUMN SEEPAGE STUDIES

	Alkaline	Deionized	Acid
pH	10.0	5.6	3.0
emf, millivolts	-110	-230	-360
Conductivity, mmhos/cm	0.03	<0.001	0.28
Ca, mg/l	4	-	-
SO_4, mg/l	-	-	35
Total Sulfur, mg/l	-	-	12

TABLE J-2

GENERAL PARAMETERS AND CONDITIONS COLUMN SEEPAGE STUDIES

Tailings Samples		
	No. 1	Zinc Tailings Fines (massive ore)
	No. 2	Zinc Tailings Fines (mixture massive/stringer ore)
	No. 3	Combined Mine Backfill

Sample Characteristics	1	2	3
Weight (dry basis), g	2,000	2,000	2,000
Sp. Gravity, g/cm^3	3.84	3.48	3.38
Particle Size, % - 10 microns	49	58	2
Pore Volume in Settled Slurry, cm^3/g of sample	0.31	0.42	0.24

Leachate Sampling

One sample, equivalent to 1 pore volume of settled slurry plus 1 pore volume of the sand filter in place at bottom of column, was drained from each column every 2 weeks (except for one sampling period of 3 weeks which occurred during the Christmas holiday season). Samples drained from the columns operated under alternate submerged-exposed conditions were reduced in volume to the extent of water evaporated during the period of exposure. Fifteen sample sets were collected over a 31 week period.

Sample volumes were as follows:

	No. 1 Sample Columns	No. 2 Sample Columns	No. 3 Sample Columns
Submerged, ml	840	1000	690
Submerged-Exposed, ml	730[2]	910[2]	608[2]

[1] Falling head procedure as measured in study columns

[2] Average of 15 samples

Leachate Analysis Parameters

pH and emf (an indicator of oxidation-reduction potential, measured by platinum electrode-calomet reference electrode) readings were taken on all samples at every sampling period. The following parameters were analyzed on selected samplings over the period of the study as deemed appropriate: conductivity, copper, iron, lead, zinc, sulfate, total sulfur, calcium, potassium and magnesium. Prior to analysis, samples were filtered through 0.45 micron filters. Tables J-3 thru J-14 show the distribution of analyses performed during the course of the study and the analysis results.

TABLE J-3

LEACHATE ANALYSIS RESULTS
COLUMN SEEPAGE STUDIES
ZINC TAILINGS FINES (massive ore)
ALKALINE SUBMERGED CONDITIONS

Column No. 1A				mg/ℓ								
Weeks	pH[1]	emf[2]	Conductivity[3]	Cu	Fe	Pb	Zn	SO$_4$	Total S	Ca	K	Mg
2	7.62	-86	2.96	0.01	0.01	0.05	16.50	1,816	668	-	-	-
4	7.82	-50	2.54	0.02	0.01	<0.02	16.70	1,597	555	-	-	-
7	7.78	-6	2.32	0.02	<0.01	<0.02	8.60	1,464	500	-	-	-
9	7.54	-39	2.29	0.01	<0.01	<0.02	6.66	1,444	489	-	-	-
11	7.62	-24	2.08	0.01	0.02	<0.02	4.65	1,460	489	-	-	-
13	7.60	-23	1.82	<0.01	<0.01	<0.02	2.38	1,065	362	-	-	-
15	7.88	-31	0.52	<0.01	<0.01	-	0.19	-	46	100	-	-
17	8.04	-31	-	-	-	-	-	-	-	-	-	-
19	8.07	-40	0.32	<0.01	0.01	-	<0.01	-	22	67	-	-
21	8.00	-60	-	-	-	-	-	-	-	-	-	-
23	8.02	-53	0.24	<0.01	<0.01	-	0.03	-	14	56	-	-
25	7.95	-12	-	-	-	-	-	-	-	-	-	-
27	8.12	-44	0.26	<0.01	<0.01	-	0.01	-	12	49	-	-
29	8.01	-66	-	-	-	-	-	-	-	-	-	-
31	7.73	-62	0.23	<0.01	<0.01	-	0.02	-	10	44	0.4	1.89

---

[1] Units

[2] Platinum electrode: calomel reference electrode, millivolts

[3] Millimhos/centimeter

TABLE J-4

LEACHATE ANALYSIS RESULTS
COLUMN SEEPAGE STUDIES
Zinc Tailings Fines (mixture massive/stringer ore)
ALKALINE-SUBMERGED CONDITIONS

Column No. 2A

				mg/ℓ								
Weeks	pH[1]	emf[2]	Conductivity[3]	Cu	Fe	Pb	Zn	SO_4	Total S	Ca	K	Mg
2	7.46	-38	3.36	0.01	0.14	0.05	2.83	1,998	683	-	-	-
4	7.75	-34	2.55	0.01	0.15	<0.02	1.79	1,695	589	-	-	-
7	7.49	-50	2.28	0.02	0.05	<0.02	0.85	1,477	497	-	-	-
9	7.51	-	2.29	0.01	<0.01	<0.02	0.57	1,457	483	-	-	-
11	7.56	+12	2.04	0.01	0.02	<0.02	0.29	1,480	495	-	-	-
13	7.49	-4	2.21	<0.01	<0.01	<0.02	0.32	1,444	482	-	-	-
15	7.53	-11	2.23	<0.01	<0.01	-	0.32	-	483	530	-	-
17	7.64	-56	-	-	-	-	-	-	-	-	-	-
19	7.97	-47	1.10	<0.01	0.03	-	<0.01	-	183	267	-	-
21	8.09	-33	-	-	-	-	-	-	-	-	-	-
23	8.08	-21	0.29	<0.01	<0.01	-	<0.01	-	15	51	-	-
25	8.13	-20	-	-	-	-	-	-	-	-	-	-
27	8.74	+43	0.23	<0.01	0.02	-	<0.01	-	12	27	-	-
29	8.86	+4	-	-	-	-	-	-	-	-	-	-
31	8.84	+67	0.21	<0.01	<0.01	-	<0.01	-	10	22	0.5	2.65

[1] Units

[2] Platinum electrode: calomel reference electrode, millivolts

[3] Millimhos/centimeter

TABLE J-5

LEACHATE ANALYSIS RESULTS
COLUMN SEEPAGE STUDIES
COMBINED MINE BACKFILL
ALKALINE SUBMERGED CONDITIONS

Column No. 3A

				mg/ℓ								
Weeks	pH[1]	emf[2]	Conductivity[3]	Cu	Fe	Pb	Zn	SO_4	Total S	Ca	K	Mg
2	7.69	-91	1.11	<0.01	<0.01	<0.02	0.11	510	215	-	-	-
4	7.38	-91	0.59	0.01	0.03	<0.02	0.11	235	88	-	-	-
7	7.69	-96	0.36	<0.01	0.02	<0.02	0.04	120	40	-	-	-
9	7.76	-74	0.30	<0.01	<0.01	<0.02	<0.01	93	30	-	-	-
11	7.80	+103	0.26	<0.01	<0.01	<0.02	<0.01	93	30	-	-	-
13	7.81	+25	0.26	<0.01	<0.01	<0.02	<0.01	71	23	-	-	-
15	7.80	+94	0.24	<0.01	0.02	-	<0.01	-	21	36	-	-
17	7.99	+79	-	-	-	-	-	-	-	-	-	-
19	8.49	-55	0.21	<0.01	0.02	-	<0.01	-	16	30	-	-
21	9.18	-35	-	-	-	-	-	-	-	-	-	-
23	9.25	-33	0.21	<0.01	<0.01	-	<0.01	-	13	22	-	-
25	9.37	-1	-	-	-	-	-	-	-	-	-	-
27	9.43	+26	0.19	<0.01	<0.01	-	<0.01	-	10	19	-	-
29	9.40	+97	-	-	-	-	-	-	-	-	-	-
31	9.39	+74	0.19	<0.01	<0.01	-	<0.01	-	10	17	0.8	2.6

[1] Units

[2] Platinum electrode: calomel reference electrode, millivolts

[3] Millimhos/centimeter

TABLE J-6

LEACHATE ANALYSIS RESULTS
COLUMN SEEPAGE STUDIES
ZINC TAILINGS FINES (massive ore)
DEIONIZED-WATER-SUBMERGED CONDITIONS

Column No. 1B				mg/ℓ								
Weeks	pH[1]	emf[2]	Conductivity[3]	Cu	Fe	Pb	Zn	SO_4	Total S	Ca	K	Mg
2	7.38	-74	2.94	0.01	<0.01	0.02	19.50	1,908	665	-	-	-
4	7.77	-62	2.53	0.02	<0.01	<0.02	16.70	1,617	568	-	-	-
7	7.55	-2	2.31	0.01	0.03	<0.02	6.78	1,474	507	-	-	-
9	7.48	-	2.31	0.01	<0.01	<0.02	4.04	1,444	494	-	-	-
11	7.49	-28	2.13	0.01	0.02	<0.02	1.72	1,480	442	-	-	-
13	7.60	-18	1.91	<0.01	<0.01	<0.02	0.08	1,189	406	-	-	-
15	7.74	+103	0.70	<0.01	0.01	-	<0.01	-	76	144	-	-
17	7.91	-17	-	-	-	-	-	-	-	-	-	-
19	7.99	-15	0.33	<0.01	<0.01	-	<0.01	-	17	64	-	-
21	8.03	+44	-	-	-	-	-	-	-	-	-	-
23	7.96	-26	0.28	<0.01	<0.01	-	<0.01	-	11	53	-	-
25	7.93	-21	-	-	-	-	-	-	-	-	-	-
27	8.21	-15	0.24	<0.01	<0.01	-	<0.01	-	9	46	-	-
29	8.01	-46	-	-	-	-	-	-	-	-	-	-
31	8.07	+72	0.22	<0.01	<0.01	-	0.03	-	8	42	0.33	1.93

[1] Units

[2] Platinum electrode: calomel reference electrode, millivolts

[3] Millimhos/centimeter

TABLE J-7

LEACHATE ANALYSIS RESULTS
COLUMN SEEPAGE STUDIES
ZINC TAILINGS FINES (mixture massive stringer ore)
DEIONIZED WATER-SUBMERGED CONDITIONS

Column No. 2B

Weeks	pH[1]	emf[2]	Conductivity[3]	mg/ℓ								
				Cu	Fe	Pb	Zn	SO_4	Total S	Ca	K	Mg
2	7.29	-15	3.26	<0.01	0.14	<0.02	3.20	2,011	680	-	-	-
4	7.62	-20	2.57	0.01	0.15	<0.02	1.93	1,714	577	-	-	-
7	7.93	+56	2.32	0.02	0.10	<0.02	0.62	1,414	599	-	-	-
9	7.53	-	2.31	0.01	0.16	<0.02	0.33	1,461	489	-	-	-
11	7.49	+48	2.10	0.01	0.02	<0.02	0.03	1,480	472	-	-	-
13	7.46	-25	2.11	<0.01	<0.01	<0.02	0.06	1,431	484	-	-	-
15	7.60	-22	2.22	<0.01	0.04	-	0.05	-	481	510	-	-
17	7.96	-26	-	-	-	-	-	-	-	-	-	-
19	8.06	-35	0.89	<0.01	0.02	-	<0.01	-	133	222	-	-
21	8.06	+5	-	-	-	-	-	-	-	-	-	-
23	8.13	+9	0.26	<0.01	<0.01	-	<0.01	-	13	49	-	-
25	8.80	+63	-	-	-	-	-	-	-	-	-	-
27	8.87	+54	0.22	<0.01	<0.01	-	<0.01	-	11	29	-	-
29	8.89	+72	-	-	-	-	-	-	-	-	-	-
31	8.96	+43	0.20	<0.01	0.03	-	<0.01	-	9	22	0.44	2.31

[1] Units

[2] Platinum electrode: calomel reference electrode, millivolts

[3] Millimhos/centimeter

TABLE J-8

LEACHATE ANALYSIS RESULTS
COLUMN SEEPAGE STUDIES
COMBINED MINE BACKFILL
DEIONIZED WATER-SUBMERGED CONDITIONS

Column No. 3B

Weeks	pH[1]	emf[2]	Conductivity[3]	mg/ℓ Cu	Fe	Pb	Zn	SO_4	Total S	Ca	K	Mg
2	7.46	-91	1.12	<0.01	<0.01	<0.02	0.06	517	219	-	-	-
4	7.44	-71	0.50	<0.01	<0.01	<0.02	0.03	168	71	-	-	-
7	7.68	-90	0.33	<0.01	<0.01	<0.02	<0.01	106	36	-	-	-
9	7.77	-	0.28	<0.01	<0.01	<0.02	<0.01	87	29	-	-	-
11	7.82	+61	0.28	<0.01	<0.01	<0.02	<0.01	88	30	-	-	-
13	7.88	-36	0.24	<0.01	<0.01	<0.02	<0.01	65	22	-	-	-
15	7.82	+81	0.22	<0.01	0.01	-	<0.01	-	18	33	-	-
17	8.00	+39	-	-	-	-	-	-	-	-	-	-
19	8.44	-38	0.18	<0.01	0.02	-	<0.01	-	12	27	-	-
21	9.05	-23	-	-	-	-	-	-	-	-	-	-
23	9.35	+3	0.18	<0.01	<0.01	-	<0.01	-	11	17	-	-
25	9.50	+85	-	-	-	-	-	-	-	-	-	-
27	9.59	+52	0.18	<0.01	<0.01	-	<0.01	-	11	17	-	-
29	9.35	+52	-	-	-	-	-	-	-	-	-	-
31	9.54	+96	0.18	<0.01	<0.01	-	<0.01	-	10	15	0.56	2.73

[1] Units

[2] Platinum electrode: calomel reference electrode, millivolts

[3] Millimhos/centimeter

TABLE J-9

LEACHATE ANALYSIS RESULTS
COLUMN SEEPAGE STUDIES
ZINC TAILINGS FINES (massive ore)
ACID-SUBMERGED CONDITIONS

Column No. 1C

				mg/ℓ								
Weeks	pH[1]	emf[2]	Conductivity[3]	Cu	Fe	Pb	Zn	SO_4	Total S	Ca	K	Mg
2	7.41	-66	2.85	0.01	<0.01	0.04	20.20	1,889	664	–	–	–
4	7.54	-37	2.44	0.01	<0.01	<0.02	17.60	1,567	545	–	–	–
7	7.39	-5	2.26	0.02	0.03	<0.02	9.50	1,462	501	–	–	–
9	7.43	–	2.29	0.01	<0.01	<0.02	6.96	1,451	490	–	–	–
11	7.64	-30	2.15	0.01	0.02	<0.02	5.08	1,460	507	–	–	–
13	7.73	-23	2.12	<0.01	<0.01	<0.02	3.48	1,360	457	–	–	–
15	7.95	-43	0.86	<0.01	0.01	–	0.62	–	122	176	–	–
17	7.92	-33	–	–	–	–	–	–	–	–	–	–
19	7.83	-64	0.51	<0.01	0.02	–	0.04	–	54	91	–	–
21	7.90	-57	–	–	–	–	–	–	–	–	–	–
23	8.00	-52	0.47	<0.01	<0.01	–	0.04	–	48	84	–	–
25	8.04	-12	–	–	–	–	–	–	–	–	–	–
27	8.16	-64	0.43	<0.01	<0.01	–	<0.01	–	40	72	–	–
29	7.83	-65	–	–	–	–	–	–	–	–	–	–
31	8.04	-78	0.41	<0.01	<0.01	–	0.02	–	37	72	0.46	7.17

[1] Units
[2] Platinum electrode: calomel reference electrode, millivolts
[3] Millimhos/centimeter

TABLE J-10

LEACHATE ANALYSIS RESULTS
COLUMN SEEPAGE STUDIES
ZINC TAILINGS FINES (mixture massive/stringer ore)
ACID-SUBMERGED CONDITIONS

Column No. 2C

				mg/ℓ								
Weeks	pH[1]	emf[2]	Conductivity[3]	Cu	Fe	Pb	Zn	SO_4	Total S	Ca	K	Mg
2	7.60	+8	3.19	0.01	0.24	0.05	3.90	2,026	677	–	–	–
4	7.37	–15	2.52	0.01	0.15	<0.02	2.56	1,680	576	–	–	–
7	7.29	–62	2.34	0.02	0.31	<0.02	1.36	1,486	504	–	–	–
9	7.32	–	2.34	0.01	0.25	<0.02	0.80	1,471	496	–	–	–
11	7.39	+39	2.10	0.01	0.02	<0.02	0.52	1,480	505	–	–	–
13	7.41	–80	2.13	<0.01	0.16	<0.02	0.40	1,438	481	–	–	–
15	7.59	–51	1.85	<0.01	0.02	–	0.19	–	373	433	–	–
17	7.69	–54	–	–	–	–	–	–	–	–	–	–
19	7.68	–18	1.49	<0.01	0.02	–	0.04	–	285	366	–	–
21	7.84	–52	–	–	–	–	–	–	–	–	–	–
23	7.99	–46	0.88	<0.01	<0.01	–	<0.01	–	141	175	–	–
25	8.03	–18	–	–	–	–	–	–	–	–	–	–
27	8.03	–19	0.43	<0.01	0.02	–	<0.01	–	46	77	–	–
29	7.94	+54	–	–	–	–	–	–	–	–	–	–
31	7.96	–32	0.40	<0.01	<0.01	–	<0.01	–	38	65	0.76	7.56

[1] Units
[2] Platinum electrode: calomel reference electrode, millivolts
[3] Millimhos/centimeter

TABLE J-11

LEACHATE ANALYSIS RESULTS
COLUMN SEEPAGE STUDIES
COMBINED MINE BACKFILL
ACID-SUBMERGED CONDITIONS

Column No. 3C

				mg/ℓ								
Weeks	pH[1]	emf[2]	Conductivity[3]	Cu	Fe	Pb	Zn	SO_4	Total S	Ca	K	Mg
2	7.35	-93	1.14	<0.01	<0.01	0.02	0.07	539	220	-	-	-
4	7.27	-68	0.66	<0.01	<0.01	<0.02	0.02	255	92	-	-	-
7	7.75	-89	0.39	<0.01	<0.01	<0.02	<0.01	131	45	-	-	-
9	7.76	-	0.34	<0.01	<0.01	<0.02	0.01	109	37	-	-	-
11	7.88	+30	0.32	<0.01	<0.01	<0.02	<0.01	117	40	-	-	-
13	7.86	+6	0.30	<0.01	<0.01	<0.02	<0.01	101	33	-	-	-
15	8.00	-1	0.30	<0.01	<0.01	-	<0.01	-	33	43	-	-
17	8.22	+28	-	-	-	-	-	-	-	-	-	-
19	8.25	+60	0.42	<0.01	<0.01	-	<0.01	-	49	60	-	-
21	8.47	-79	-	-	-	-	-	-	-	-	-	-
23	8.53	-62	0.40	<0.01	<0.01	-	<0.01	-	44	50	-	-
25	8.68	+45	-	-	-	-	-	-	-	-	-	-
27	8.86	-30	0.34	<0.01	<0.01	-	<0.01	-	38	38	-	-
29	8.71	+35	-	-	-	-	-	-	-	-	-	-
31	8.82	+85	0.36	<0.01	<0.01	-	<0.01	-	36	37	1.03	12.6

[1] Units
[2] Platinum electrode: calomel reference electrode, millivolts
[3] Millimhos/centimeter

TABLE J-12

LEACHATE ANALYSIS RESULTS
COLUMN SEEPAGE STUDIES
ZINC TAILINGS FINES (massive ore)
DEIONIZED WATER-ALTERNATE SUBMERGED/EXPOSED CONDITIONS

Column No. 1D

				mg/ℓ								
Weeks	pH[1]	emf[2]	Conductivity[3]	Cu	Fe	Pb	Zn	SO_4	Total S	Ca	K	Mg
2	7.83	-33	3.12	0.01	<0.01	<0.02	8.75	2,061	730	-	-	-
4	7.73	-51	2.78	0.01	0.01	<0.02	8.50	1.757	628	-	-	-
7	7.99	-70	2.35	0.02	0.03	<0.02	7.00	1,475	513	-	-	-
9	7.98	-	2.33	0.01	<0.01	<0.02	5.32	1,456	498	-	-	-
11	7.67	-46	2.19	0.01	0.02	<0.02	6.73	1,490	502	-	-	-
13	7.96	-80	2.20	<0.01	<0.01	<0.02	4.75	1,453	482	-	-	-
15	8.09	-70	2.25	<0.01	0.03	-	3.66	-	489	602	-	-
17	8.08	-37	-	-	-	-	-	-	-	-	-	-
19	8.32	-29	0.50	<0.01	0.03	-	0.47	-	44	95	-	-
21	8.20	-41	-	-	-	-	-	-	-	-	-	-
23	8.09	-64	0.46	<0.01	<0.01	-	0.96	-	48	84	-	-
25	8.20	-79	-	-	-	-	-	-	-	-	-	-
27	8.07	-107	0.40	<0.01	<0.01	-	0.99	-	39	69	-	-
29	8.18	-84	-	-	-	-	-	-	-	-	-	-
31	8.15	-20	0.32	<0.01	<0.01	-	0.11	-	24	54	0.46	5.46

[1] Units
[2] Platinum electrode: calomel reference electrode, millivolts
[3] Millimhos/centimeter

TABLE J-13

LEACHATE ANALYSIS RESULTS
COLUMN SEEPAGE STUDIES
ZINC TAILINGS FINES (mixture massive/stringer ore)
DEIONIZED WATER-ALTERNATE SUBMERGED/EXPOSED CONDITIONS

Column No. 2D				mg/ℓ								
Weeks	pH[1]	emf[2]	Conductivity[3]	Cu	Fe	Pb	Zn	SO_4	Total S	Ca	K	Mg
2	8.24	-41	3.29	0.01	<0.01	<0.02	1.15	2,304	790	-	-	-
4	7.90	-47	3.17	0.01	0.02	<0.02	1.12	2,155	740	-	-	-
7	8.00	-84	2.31	0.02	0.02	<0.02	0.85	1,503	513	-	-	-
9	7.95	-	2.30	0.01	<0.01	<0.02	0.55	1,452	494	-	-	-
11	7.83	-44	2.10	0.01	0.02	<0.02	0.73	1,490	498	-	-	-
13	7.79	-98	2.17	<0.01	<0.01	<0.02	0.73	1,446	479	-	-	-
15	7.94	-58	2.22	<0.01	0.04	-	0.62	-	482	582	-	-
17	8.12	-72	-	-	-	-	-	-	-	-	-	-
19	7.98	-70	2.16	<0.01	<0.01	-	0.74	-	482	584	-	-
21	7.90	-40	-	-	-	-	-	-	-	-	-	-
23	8.38	-17	0.44	<0.01	<0.01	-	<0.01	-	40	87	-	-
25	8.36	-5	-	-	-	-	-	-	-	-	-	-
27	8.33	-5	0.29	<0.01	<0.01	-	0.09	-	19	52	-	-
29	8.28	-41	-	-	-	-	-	-	-	-	-	-
31	8.21	-8	0.26	<0.01	<0.01	-	<0.01	-	16	44	0.62	3.53

[1] Units
[2] Platinum electrode: calomel reference electrode, millivolts
[3] Millimhos/centimeter

TABLE J-14

LEACHATE ANALYSIS RESULTS
COLUMN SEEPAGE STUDIES
COMBINED MINE BACKFILL
DEIONIZED WATER-ALTERNATE SUBMERGED/EXPOSED CONDITIONS

Column No. 3D

				mg/ℓ								
Weeks	pH[1]	emf[2]	Conductivity[3]	Cu	Fe	Pb	Zn	SO_4	Total S	Ca	K	Mg
2	7.02	-126	1.34	0.01	<0.01	<0.02	0.06	761	254	-	-	-
4	7.40	-80	0.97	<0.01	0.13	<0.02	0.11	500	167	-	-	-
7	7.98	-143	0.58	<0.01	0.02	<0.02	0.09	247	82	-	-	-
9	7.95	-	0.47	<0.01	<0.01	<0.02	0.03	184	60	-	-	-
11	7.84	-60	0.39	<0.01	<0.01	<0.02	0.18	140	54	-	-	-
13	7.64	-129	0.43	<0.01	<0.01	<0.02	0.25	169	56	-	-	-
15	7.93	-106	0.44	<0.01	0.01	-	0.29	-	56	59	-	-
17	7.98	-120	-	-	-	-	-	-	-	-	-	-
19	7.91	-122	0.37	<0.01	<0.01	-	0.40	-	49	47	-	-
21	7.84	-119	-	-	-	-	-	-	-	-	-	-
23	7.82	-117	0.39	<0.01	<0.01	-	0.43	-	51	50	-	-
25	7.83	-107	-	-	-	-	-	-	-	-	-	-
27	7.84	-104	0.36	<0.01	<0.01	-	0.41	-	47	45	-	-
29	7.79	-153	-	-	-	-	-	-	-	-	-	-
31	7.62	-101	0.33	<0.01	<0.01	-	0.63	-	42	40	0.68	12.7

[1] Units
[2] Platinum electrode: calomel reference electrode, millivolts
[3] Millimhos/centimeter

TABLE J-15

PERMEABILITY MEASUREMENTS[1] ON TAILINGS SAMPLES[2] USED IN COLUMN SEEPAGE STUDIES

	Permeability, cm/sec	
	After 4 Weeks of Operation	After 31 Weeks of Operation
Sample 1		
Alkaline Conditions	2.1×10^{-5}	1.8×10^{-5}
D.I. Water Conditions	2.2×10^{-5}	2.1×10^{-5}
Acid Conditions	2.6×10^{-5}	2.4×10^{-5}
Sample 2		
Alkaline Conditions	1.9×10^{-5}	1.8×10^{-5}
D.I. Water Conditions	2.0×10^{-5}	1.9×10^{-5}
Acid Conditions	2.0×10^{-5}	2.4×10^{-5}
Sample 3		
Alkaline Conditions	2.3×10^{-3}	2.6×10^{-3}
D.I. Water Condtiions	2.3×10^{-3}	2.4×10^{-4}
Acid Conditions	2.2×10^{-3}	2.6×10^{-3}

[1] Falling Head Method

$$\text{Permeability} = \left(\frac{\text{sample length, cm}}{\text{drainage time, sec}}\right) \ln \left(\frac{\text{initial hydraulic head, cm}}{\text{final hydraulic head, cm}}\right)$$

[2] Sample 1: Zinc Tailings Fines (massive ore)
Sample 2: Zinc Tailings Fines (mixture massive/stringer ore)
Sample 3: Combined Mine Backfill

APPENDIX K

LITERATURE REFERENCES ON CHARACTERISTICS OF SIMILAR TAILINGS

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