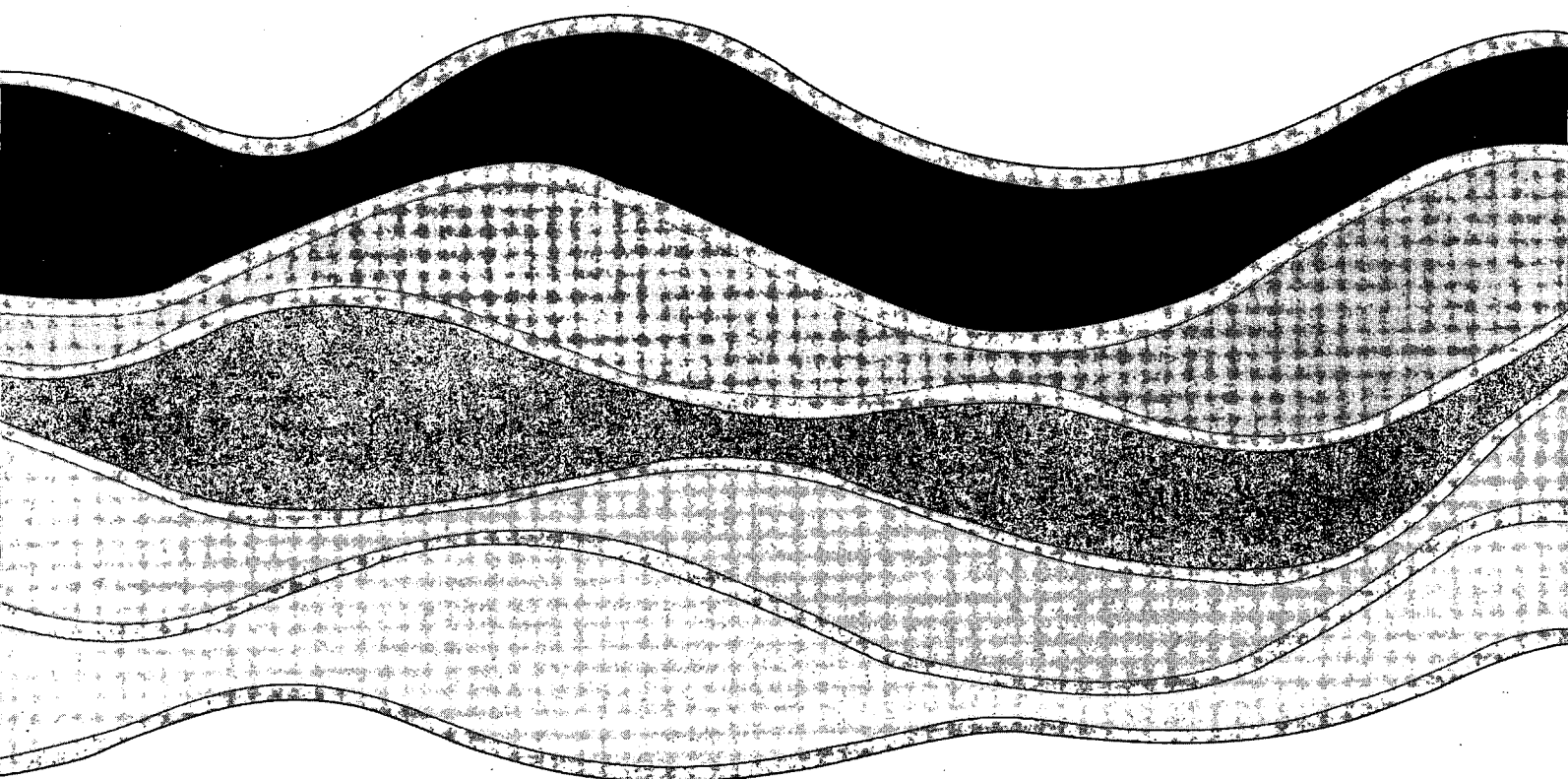
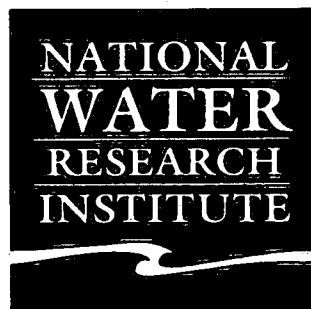


95-180



**PRELIMINARY REPORT ON THE CHARACTERISTICS
OF THE BOF SLAG AND FEASIBILITY OF
ITS APPLICATION IN REMEDIATION OF
ACID DRAINAGE GENERATING
METAL MINE TAILINGS**

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MANAGEMENT PERSPECTIVE

Many mine tailings generated by mining and extraction of metals contain different quantities of sulphides. During exposure to air and water, the water insoluble sulphides in the tailings become converted to water soluble sulphates which acidify surface and groundwater with subsequent contamination by different toxic metals and trace elements. Dry covers of different materials inhibiting acid drainage from the tailings have been considered and studied, particularly in situations where acid generation can not be effectively controlled by other means, such as subaqueous disposal of acid drainage generating tailings. A wide range of alternate dry cover materials from municipal compost to engineered composite covers has been considered and research proposed for several alternate materials. This study was carried out to obtain preliminary information on the neutralizing capacity of slag generated by steel production prior to considering the slag as a dry cover of acid generating mine tailings.

The results of the testing of the efficiency of the slag suggested that the slag is effective in neutralizing the acid drainage from the tailings. The addition of the slag to the tailings appears to be efficient even in small amounts, such as a mixture of tailings and slag at a ratio 20:1. A rapid increase of pH in the tailings leachate from approximately 2.5 to 7 and the duration of this value over a

4-month period indicated the high neutralizing potential of the slag. The analysis of the leachate after 3 months of continuous leaching of the tailings/slag mixture generally indicated desirable effects of precipitation of trace elements within the tailings resulting in their low mobility and eventual leaching into the environment. The use of the slag appears to be a viable economical alternative in the treatment of acid mine drainage on the basis of the results described in this report and considering large volumes of available slag. The advantage of the use of one waste material in treatment of another waste material should also be considered.

INTRODUCTION

The mining industry generates liquid and solid wastes which contain many toxic elements. Many mine tailings generated by mining and extraction of metals contain different quantities of sulphides. During exposure to air and water, the water insoluble sulphides in the tailings become converted to water soluble sulphates. These sulphates acidify surface and groundwater, which become subsequently contaminated by different metals and trace elements leached from the tailings.

Several studies have been carried out to develop acceptable techniques of inhibiting acid mine drainage from sulphur containing metal mine tailings. The feasibility of subaqueous disposal of acid generating tailings has been evaluated. Dry covers of different materials inhibiting acid drainage from the tailings have been considered and studied, particularly in situations where acid generation can not be effectively controlled by other means, such as subaqueous disposal of acid drainage generating tailings. A limited knowledge is available from studies of dry covers, particularly of basic geochemical processes by which dry covers may inhibit the acid drainage from the tailings. Studies of low permeability, oxygen consuming dry cover material have indicated that the basic requirements to limit molecular diffusion of atmospheric oxygen through interstitial pores of a cover can be accomplished. The

results of the studies also indicated several restrictions in using dry covers, such as the long-term effectiveness of dry covers as oxygen barriers; the net impact on the rate of oxidation of the tailings; poor availability of dry cover material in some locations; prohibitive cover construction costs; and effluent quality and quantity and its impact on the receiving environment. A wide range of alternate dry cover materials from municipal compost to engineered composite covers has been considered to date. Research has been proposed for several alternate materials (Mine Environment Neutral Drainage, 1994).

This study was carried out to obtain preliminary information on the neutralizing capacity of slag generated by steel production prior to considering the slag as a dry cover of acid generating mine tailings. The study was conducted at the National Water Research Institute, Canada Centre for Inland Waters, Burlington, Ontario. Tests involving the slag were carried out using the safety standards in operation of the slag recommended by Stelco, Hamilton, Ontario (Material Safety Data Sheet - Stelco 1990).

MATERIALS AND METHODS

Approximately 10 kg of BOF slag obtained from Stelco, Inc., Hamilton, Ontario, was stored in a metal bucket with a lid in a cold room at 4°C prior to the study. Prior to testing, the slag sample was subsampled into several portions of

approximately 200 g by quartering. Individual portions were kept in closed plastic containers at room temperature. The remainder of the slag was returned to the 4°C storage room.

Mine tailings were collected at the Mattabi Mining Co. site near Ignace, Ontario. Approximately 10 kg of the tailings were placed in large plastic bags. The bags with the tailings were sealed, shipped to the laboratory and stored at room temperature.

Approximately 150 kg of tailings were collected at the Timmins Mining Co. tailings disposal site near Timmins, Ontario, in winter 1995. The tailings were transported from the site and stored in plastic boxes covered with lids at room temperature.

The Mattabi Mining Co. tailings were used in the leaching and neutralizing tests with the slag. The tailings collected at Timmins were used in the characterization of general physico-chemical properties of acid generating mine tailings, particularly the permeability.

Characterization of the slag

The Moisture content of the slag was determined by freeze-drying using freeze dryer model 24DX48 VIRTIS, and by oven drying. After determining the moisture content, the freeze-dried slag was further used in the determination of grain

size by sieving.

The determination of the moisture content by oven drying was carried out in an Isotemp Oven model 655 F at 60°C for minimum 24 hours. After removal from the oven, the samples were cooled in a dessicator, and the loss of weight calculated.

The concentration of organic matter in the slag was determined by loss on ignition (LOI) using a Thermolyne 1400 furnace. A sample of the slag, dried in the oven for 24 hours at 60°C, cooled and weighed, was placed in the furnace for 2 hours at 500°C.

After oven drying the slag at 60°C for 24 hours with subsequential cooling in a desiccator and weighing, a magnetic stirbar was used in the separation of magnetic particles.

A LECO-12 carbon analyzer was used to determine the concentrations of organic and inorganic carbon in the slag. The procedure consisted of the following steps:

1. Drying the samples in the oven at 60°C for a minimum of 2 hours.
2. Weighing approximately 1.000 g of sample into a ceramic crucible.

3. In the determination of organic carbon, the temperature in the combustion furnace was maintained for 250 seconds at 500°C. The samples used for the determination of the concentrations of organic carbon were then set aside and used in the determination of the concentrations of inorganic carbon.

4. In the determination of inorganic carbon, the combustion temperature was maintained for 60 seconds at 1370°C.

Particle size distribution of dry slag was determined by sieving and sedigraph. Sieve shaker OCTAGON 200 (ENDECOTTS Ltd.) and the procedures described by Duncan and LaHaie (1979) were used in the determination of the particle size distribution in the slag. The method using the sedigraph is designed to process less than 5 g of samples composed mainly of silt and clay (Duncan and LaHaie, 1979). Therefore the fine particle fraction ($<63 \mu\text{m}$) obtained by sieving the slag was used in the sedigraph analysis. The procedure was described by Duncan (1993).

The determination of the density of the slag was carried out by using the AccuPyc 1330 Pycnometer following the procedure described by Micromeritics.

Characterization of the Timmins Mining Co. tailings

The methods used in the determination of the physico-chemical properties of the Timmins tailings were similar to those used for the BOF slag.

Neutralizing the leachate from Mattabi Mining Co. tailings by addition of slag

The testing of the neutralizing capacity of the slag was carried out by leaching Mattabi Mining Co. tailings with and without the addition of the slag. Six polypropylene tubes were used in the test: four, 30-cm long, 3.5-cm inside diameter tubes and two, 30-cm long, 10.5-cm inside diameter tubes. During the test all tubes were maintained in a vertical position by clamps mounted on laboratory stands. Glass wool pads, approximately 3 cm thick, were placed on the bottom of the tubes to prevent sliding of the tailings out of the tubes. The first tube was filled with a mixture of approximately 45 g of the tailings and 200 ml of distilled water. After the tailings settled on the bottom of the tube they were leached with 1,100 ml of distilled water. During the entire leaching period the tailings did not become exposed to the air. The effluent from the tube was collected and pH measured in each 100 ml portion of the effluent. A mixture of approximately 175 g of the tailings and 200 ml of distilled water was introduced into the second tube. After settling to the bottom of the tube, the tailings were leached with distilled water similar to those in the

first tube. After passing 1,000 ml of distilled water through each of the tubes with the tailings, a 100-ml aliquot of the effluent was acidified by 0.5 ml conc. HNO_3 and used for quantitative determination of metals and trace elements. Further, a 3-cm thick layer of the slag (approximately 58 g) was placed on the top of the tailings in each tube, and an additional 1,100 ml of distilled water was used to leach the tailings topped with the slag. The effluent was collected in 100-ml portions and pH was measured in each portion. The last 100-ml portion of the total 2,200 ml of the effluent was acidified with 0.5 ml of conc. HNO_3 and used in the quantitative determination of metals and trace elements in the effluent. A similar test was carried out using two, 30-cm long, 10.5-cm diameter polypropylene tubes and approximately 600 g of the tailings mixed with 200 ml of distilled water. After leaching of the tailings in the tubes by 900 ml of distilled water, a 5-cm layer of the slag (i.e., 165 g) was placed on the top of the tailings in each tube. An additional 900 ml of distilled water was passed through the tailings topped with the slag. The pH measurements of the effluent and the quantitative determination of metals and trace elements in the effluent were similar to those described for the first two tubes. One, 30-cm long, 3.5-cm inside diameter tube was filled with a mixture of 230 g of slag and 200 ml of distilled water. After settling to the bottom of the tube, the 9-cm long slag column was leached by 1,600 ml of distilled water. Further,

one, 30-cm long, 3.5-cm inside diameter tube was filled with a mixture of 220 g of slag and 200 ml of distilled water.

After settling on the bottom of the tube the 12-cm long slag column was leached by 700 ml of distilled water followed by 500 ml of leachate from the tailings. The pH of the 500-ml leachate was 2.72. The measurements of pH and the determination of the concentrations of major and trace elements in the leachates from both tubes were carried out similar to those used for the leachates from the first four tubes in the test.

Neutralizing the leachate from Timmins Mining Co. Tailings by addition of slag

Four plexiglass columns were used in the test. The columns were 30 cm long with 3.5 cm inside diameter. The columns were supported in a vertical position by laboratory clamps and were stoppered on the bottom by rubber stoppers with 8 mm holes in the centre of the stoppers. Tygon tubing was inserted into the holes to allow for the collection of leachate from the material in the columns, and clamps were placed on the tubing to allow control of the flow speed of the leachate. An approximately 2-cm thick layer of glass wool was placed on the top of the rubber stopper to prevent loss of tailings and to filter the leachate prior to its collection. Column No. 1 was used as a control. A slurry of 200 g of the tailings and distilled water was poured into the column. Mixtures of 200 g of tailings and 50, 20 and 10

g of the slag were made into a slurry with 100 ml of distilled water and poured into columns No. 2, 3 and 4. The 15-cm thick layers of the mixtures in the columns represented different tailings/slag application ratios. The clamps on the tygon tubing were closed and the columns were slightly tapped on all sides to eliminate void space and to speed up the settling of the material. Erlenmayer flasks were used to collect the leachate. The columns were placed on the top of the flasks to prevent evaporation of the leachate in the flasks. Fifty-ml aliquots of distilled water were used to leach the material until an adequate volume (approximately 200 ml) of the leachate was collected for measuring the pH and determination of the concentrations of major and trace elements. Each 50-ml aliquot of distilled water was allowed to completely pass into the tailings/slag mixture in the columns before adding the next aliquot. The design of the leaching test allowed periods of drying of the material in the columns, i.e., the material remained dry for extended time periods.

Permeability tests

For the testing of the permeability samples of the Timmins Mining Co. tailings and slag were prepared without drying or other sample preparation in order to simulate the field conditions. The falling-head permeability test is generally used for less pervious (fine sands to clay) soils and sediments. The coefficient of permeability, k , is defined as

the rate of discharge of water at a temperature of 20°C under conditions of laminar flow through a unit cross-sectional area of a soil medium under a unit hydraulic gradient. The coefficient of permeability has the dimension of a velocity and it is usually expressed in cm/s. The permeability of a soil depends primarily on the size and shape of the soil grains, the void ratio of the soil, and the shape and arrangement of the voids (U.S. Army Engineer Waterways Experiment Station 1970).

Three, 30-cm long, 3.5-cm diameter plastic tubes were used for permeability testing. Wire screens of 200 mesh and glass wool pads 0.5-cm thick were placed on the bottom of the tubes to prevent the loss of fine fractions of tailings from the columns. A scale with a 1-cm interval was attached to each column. Three samples, each about 200 g of tailings (in natural condition), were thoroughly mixed with a small amount of distilled water in the plastic beaker to avoid segregation. The samples were transferred into the columns and distilled water was added up to the highest level of the scale. The columns were divided into half the height of the beakers. The beakers were filled with distilled water until they started to overflow from the beakers. Time was measured as the head of water was falling down from the highest to the lowest level of the scale on the columns.

The following permeability tests were carried out using Timmins tailings and the slag:

A. Three columns were filled with a mixture of about 200 g of fresh tailings and about 10 g of $<500\ \mu\text{m}$ particles of the slag.

B. Two columns were filled with 190 g of tailings. A mixture of 10 g of tailings and 10 g of $<2\ \text{mm}$ particles of the slag was placed on the top after the tailings were allowed to settle for 72 hours.

C. Two columns were filled with 180 g of tailings and mix of 20 g of tailings and 10 g, 0-2 mm of slag was placed on the top after the tailings settled down for 72 hours.

D. Two columns were filled with 100 g of tailings. A mixture of 100 g of tailings and 10 g of slag particles $<500\ \mu\text{m}$ was placed on the top of the tailings after they were allowed to settle for 24 hours.

E. Two columns were filled with 100 g of tailings. A mixture of 100 g of tailings and 5 g of slag particles $<500\ \mu\text{m}$ was placed on the top of the tailings after they were allowed to settle for 24 hours. Further, 5 g of slag particles $<500\ \mu\text{m}$ was added on the top after next 24 hours.

In another series of tests, two columns were filled with the same material and using the identical procedure described above, changing the ratio of the tailings and slag to 10:1.

The coefficient of permeability, k , was computed using the following equation:

$$k = l/t * (\ln h_o/h_f) * R_t$$

where:

$t = t_f - t_o$ (s)

t_o = Initial time

t_f = Final time

l = Height of sample in the column (cm)

h_o = Initial head of water in the column (cm)

h_f = Final head of water in the column (cm)

R_t = Temperature correction factor for viscosity of water (U.S. Army Engineer Waterways Experiment Station 1970, Table VII-1).

Evaporation of water from the surface of the 3.5-cm diameter column at 22.2°C under a laboratory fume hood was 0.16 cm³/hr. For example, if the falling-head permeability test lasted 8 hours, 1.28 cm³ of water should be added before the lower water level was measured. Recalculated k values, corrected for evaporation, were close to the uncorrected values, in the order of 10⁻⁵ cm/s. Therefore this correction was neglected.

Quantitative determination of major and trace elements in tailings, slag and leachates

The quantitative determination of major and trace elements (Al, Ca, Fe, Mn, K, Na, Mg, Cd, Cr, Co, Cu, Ni, Pb, V and Zn) in the slag, tailings from the Timmins Mining Co. and leachates was carried out by inductively coupled plasma atomic emission spectroscopy (ICP-AES) using Jobin Yvon Model 74. The tailings and the slag were digested in 3 replicates using aqua regia. Aqua regia was added to Teflon containers containing 0.3 g of samples with subsequent mixing. The samples were allowed to de-gas at room temperature overnight to prevent vigorous reaction during heating. The containers with the samples were covered with Teflon lids to protect the samples from contamination while allowing gas to escape. The samples were digested in a microwave oven (Floyd Inc., Model RMS 150) at mode 3-5 steps program. Conditions used in the microwave digestion and ICP-AES operation are shown in Appendix 1, Tables 1 and 2, respectively. Digested samples were cooled and filtered through 0.4 μ m Nuclepore Polycarbonate filters into volumetric flasks. Calibration standards were prepared by diluting stock solutions of ICP multi-element calibration standards (High Purity Standards, Charleston, South Carolina, distributed by Delta Scientific, Mississauga, Ontario). Two standards (0 and 10 mg/l) made up in 2% HNO_3 were used to calibrate the ICP-AES. The detection limits, defined as concentration equivalent to 2x standard deviation

obtained from all blank samples ($n=8$) are shown in Appendix 1, Table 3. Certified reference material 2704 (Buffalo River Sediment) of the National Institute of Standards and Technology, U.S. Department of Commerce, Maryland, was used in the quality control of the analysis. Subsamples of the certified reference material were digested following the method used for the slag and tailings samples.

The quantitative determination of the major and trace elements in the Mattabi Mining Co. tailings was carried out by x-ray fluorescence spectrometry using powder pellets. The method was described in details by Mudroch (1985).

RESULTS AND DISCUSSION

By visual observation, the colour of the slag was dark brown with a grey layer on the surface. Microscope examination of the slag revealed porous, adhesive material, containing a high percentage of CaO , ash and some magnetic material.

Pieces of biological material such as dry grass, leaves and wood were found in the slag. The moisture content of the slag determined by the freeze-drying was 3.05%, and that determined by oven drying was 3.35%. Chemical composition of the slag used in all tests is shown in Table 1. The values in the table are the average of analysis of three individual subsamples of the slag. The concentration of organic matter in the slag by loss on ignition was 1.18%. The separation of magnetic fraction showed that approximately 60% of the slag

was magnetic particles. The results of the sieving analysis showed that the slag contained approximately 80% of gravel and coarse sand, and approximately 2% of <63 μm particles. Similar results were obtained by wet sieving and sieving of the freeze-dried slag samples (Table 2).

The density of different particle size fractions of the slag is shown in Table 3 and Figure 1. The density ranged from 3.16 to 3.70 g/cm^3 and gradually decreased with decreasing particle size.

By visual observations the Timmins Mining Co. tailings consisted of grey fine material with a yellow-brown layer on the surface. The moisture content of the tailings was 27.6%. The results of the determination of the physico-chemical properties of the Timmins tailings are shown in Tables 4 to 6 and Figures 2 and 3.

The chemical composition of the Mattabi Mining Co. tailings is shown in Table 7. During the 12-day leaching period of the Mattabi Mining Co. tailings by using over 1 L of distilled water in first two tubes, the pH of the leachate ranged from 2.24 to 4.05. After the addition of a layer of slag on the surface of the tailings, the pH of the leachate increased within one day to 8.15 and on the following day to 11.48. During 8 days of subsequent leaching of the tailings with added slag by 900 ml of distilled water, the pH of the

leachate was constantly >11 (Figure 4). Similar results were obtained by leaching the tailings in the two, 30-cm long, 10.5-cm inside diameter tubes. The pH of the leachate from the slag during an 8-day leaching period was between 11.66 and 12.55 (Figure 5). The concentrations of major and trace elements in the leachates from the tailings before and after the addition of the slag are shown in Tables 8 and 9. Generally, the concentrations of most of the major and trace elements in the leachates decreased after the addition of the slag to the top of the tailings. The decrease was different in each column, most likely due to the different tailings/slag ratio in each of the four tubes. The tailings/slag ratios were approximately 0.8, 3.0 and 3.7 in tubes one and two and in the other two tubes, respectively. In addition, the diameter of the first two tubes was smaller than that of the other two tubes used in the test. However, all tests confirmed the potential of the slag in neutralizing the acid effluent from the tailings with subsequent retention of most of the elements determined in the leachates.

The concentrations of Fe, Mn, Cd, Co, Cr, Cu, Ni, Pb, V and Zn in the leachate from the slag decreased by continuing the initial leaching of the slag with 700 ml of distilled water (Table 10). The slag was efficient in removing most of the elements from the acidic leachate from the tailings. For example, it reduced the concentrations of Zn and Cu in the

leachate from the tailings from approximately 100 mg/L to approximately 1 and 0.75 mg/L, etc. (Table 10).

The addition of the slag effectively rised the pH of the leachate from the Timmins Mining Co. tailings at all application ratios (Figure 6). At the tailings/slag ratio 20:1 pH increased to approximately 7 and remained similar during the 4-month test (Figure 6). This indicated the high neutralizing capacity of the slag most likely due to the high content of MgO and CaO. The tailings/slag ratio 10:1 or 4:1 may be considered in neutralizing subsurface layers of the tailings. However, since revegetation of the surface of the tailings is one of the objectives of the study, the tailings/slag ratio 20:1 may render the root zone (surface 15 cm) too alkaline for the establishment of vegetation on the tailings.

Initially, the leachate from the columns was only monitored for pH. However, after leaching the tailings and the tailings/slag mixtures by 500 ml of distilled water, the leachate was monitored for the concentrations of major and trace elements. Aliquots of leachate collected at 1- to 3-month intervals were acidified and analysed. It should be noted that the concentrations of the elements in the leachates shown in Figure 7 may be low since some of the elements could have been leached within the first 3 weeks of the test. In general, the concentrations of trace elements

in the leachates after 1 month of leaching were greater than those in the leachates collected after the 3-month leaching (Figure 7) indicating precipitation of the elements as hydroxides. The trace elements in the leachates collected after 1 month may be present in more soluble form in the tailings. On the other hand, those leached after 3 months may indicate steady-state concentrations in the evaluation of the long-term impact of the leachate from the tailing/slag mixture.

The permeability of different mixtures of the Timmins Mining Co. tailings and the slag was similar. A clear trend observed was a slight decrease in the k-value with time, which was probably due to gradual settling of tailings or mix of tailings and the slag. However, the permeability of the Timmins Mining Co. tailings was slightly different than that of the different mixtures of the tailings and the slag (Table 11, Figure 8). The k-values were between 1.03×10^{-4} to 7.21×10^{-5} cm/s which is characteristic for materials in the boundary between fine sand and silt (Kézdi 1974).

The permeability test of the mix of fresh Timmins tailings and the slag in ratio 20:1 did not indicate a permeability decrease by the addition of the slag. The k-values were between 1.10×10^{-4} and 2.46×10^{-4} cm/s (Table 11, Figure 8). The permeability of a mixture of the tailings and <5 mm slag particles at a ratio 10:1 placed on top of the tailings

ranged from 1.09×10^{-4} to 1.29×10^{-4} cm/s indicating a small decrease in the permeability during the 9-day test period (Table 11, Figure 8). The development of a thin white layer at the contact between the slag-tailings mix and the underlying tailings suggested that the decrease of the permeability was most likely due to the precipitation of CaSO_4 .

The permeability of the slag and the tailings with different ratios of the tailings and the slag, different grain size of the slag and different thickness of the layers of the slag on the top of the tailings remained almost unchanged with the k-values ranging between 1.22×10^{-4} and 2.16×10^{-4} cm/s (Table 8, Figure 9).

The results of the permeability tests using different grain sizes of the slag and ratios of the slag and the tailings indicated that these "mixes" or "layers" could be used to mitigate the generation of acid effluent from tailings without significant change in permeability. On the other hand, tests in larger columns and small-scale field tests should be considered before applying k-values obtained in the laboratory to field conditions. It is further recommended that additional permeability tests be carried out with different ratios of the tailings and the slag and with layers of different thickness of the slag placed on the top of the tailings.

CONCLUSIONS

The results of the testing of the efficiency of the slag suggested that the slag is effective in neutralizing the acid drainage from the tailings. The addition of the slag to the tailings appears to be efficient even in small amounts, such as a mixture of tailings and slag at a ratio 20:1. A rapid increase of pH in the tailings leachate from approximately 2.5 to 7 and the duration of this value over a 4-month period indicated the high neutralizing potential of the slag. The analysis of the leachate after 3 months of continuous leaching of the tailings/slag mixture generally indicated desirable effects of precipitation of trace elements within the tailings resulting in their low mobility and eventual leaching into the environment. The results of the tests and considering the production of the large volumes of slag, the use of the slag appears to be a viable economical alternative in the treatment of acid mine drainage. The advantage of the use of one waste material in treatment of another waste material should also be considered.

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TABLE 1. Chemical composition of Stelco Inc. slag

Element	%	Element	ug/g
CaO	28.0	Cd	35
MgO	7.0	Co	360
Al ₂ O ₃	2.1	Cu	65
Fe (total)	18.7	Ni	45
Mn (total)	4.1	Pb	250
K ₂ O	0.25	V	80
Na ₂ O	0.01	Zn	430
Cr ₂ O ₃	0.29		
Organic C	0.43		
Inorganic C	3.23		

TABLE 2. Grain size distribution of slag

Phi	Interval (mm,um)	Wet Sample		Dry Sample		Freeze Dry Sample	
		(g)	(%)	(g)	(%)	(g)	(%)
-2.0	>4	83.4482	45.98	78.1221	43.80	87.44	46.18
-1.5	2.8-4	19.1979	10.58	20.9654	11.75	17.29	9.13
-1.0	2.0-2.8	15.9233	8.77	15.1073	8.47	13.93	7.36
-0.5	1.4-2.0	10.9173	6.02	10.9544	6.14	12.59	6.65
0.0	1.0-1.4	8.7583	4.83	8.1297	4.56	9.05	4.78
0.5	0.71-1.0	8.5170	4.69	7.6895	4.31	7.05	3.72
1.0	500-710	6.8625	3.78	5.1755	2.90	7.23	3.82
1.5	355-500	5.5111	3.04	5.9258	3.32	6.13	3.24
2.0	250-355	8.7720	4.83	5.1835	2.91	6.11	3.23
2.5	180-250	5.3751	2.96	6.0321	3.38	5.68	3.00
3.0	125-180	7.4133	4.09	5.3492	3.00	5.43	2.87
3.5	63-125	0.4894	0.27	3.3626	1.89	4.38	2.31
4.0	63-90	0.2467	0.14	2.7455	1.54	2.49	1.32
	<63	0.0417	0.02	3.6359	2.04	4.54	2.40
		181.4738	100.00	178.3785	100.00	189.34	100.00

TABLE 3. Density of different particle size fractions of slag

fraction (mm)	Sample weight (g)	density (g/cm ³)
1.0-1.4	12.8970	3.70
0.7-1.0	13.0093	3.67
0.5-1.0	12.9315	3.65
0.5-0.7	11.4697	3.65
0.25-0.5	11.1402	3.58
0.18-0.5	10.3316	3.61
0-0.5	12.1732	3.44
0-0.25	11.1061	3.37
0-0.18	9.4935	3.25
0-0.15	10.5009	3.31
0-0.063	9.9334	3.16

TABLE 4. Density of different particle size fractions of Timmins Mining Co. tailings

fraction (mm)	no.	Sample weight (g)	density (g/cm ³)
0-1.4	1	8.8821	2.70
0-1.4	5	8.2207	2.71
0-1.4	8	8.5166	2.68
0-1.4	9	9.4392	2.70
0-1.4	10	7.5171	2.70
0.125-0.18	3-1	8.1879	2.70
0-0.063	3-2	6.0809	2.78

TABLE 5. Grain size distribution of Timmins Mining Co. tailings

Interval (mm, um)	Sample no. 1		Sample no. 2		Sample no. 3	
	(g)	(%)	(g)	(%)	(g)	(%)
	29.8868		29.7982		29.8543	
>4	-		-		-	
2.8-4	-		-		-	
2.0-2.8	-		-		-	
1.4-2.0	-		-		-	
1.0-1.4	0.0041	0.01	0.0026	0.01	0.0096	0.03
0.71-1.0	0.0285	0.10	0.0427	0.14	0.0631	0.21
500-710	0.1185	0.40	0.1763	0.59	0.1730	0.58
355-500	0.6092	2.04	0.5907	1.98	0.6175	2.07
250-355	1.6581	5.55	1.6099	5.40	1.6557	5.55
180-250	3.0288	10.13	2.9833	10.01	2.9285	9.81
125-180	8.5705	28.68	8.4445	28.34	8.1873	27.42
90-125	3.2328	10.82	3.3291	11.17	3.2036	10.73
63-90	3.7284	12.48	3.5560	11.93	3.4296	11.49
<63	8.4859	28.39	8.4240	28.27	9.1815	30.75
	29.4648	98.59	29.1591	97.86	29.4494	98.64

TABLE 6. Chemical composition of Timmins Mining Co. tailings

Element	%	Element	ug/g
CaO	2.5	Cd	10
MgO	0.5	Cr	30
Al ₂ O ₃	1.0	Cu	90
Fe (total)	4.4	Ni	25
Mn (total)	0.03	Pb	430
K ₂ O	0.18	V	80
Na ₂ O	0.01	Zn	430

TABLE 7. Chemical composition of Mattabi Mining Co. tailings

Element	%	Element	ug/g
SiO ₂	39.0	Co	160
CaO	0.6	Cr	70
MgO	0.8	Cu	870
Al ₂ O ₃	8.6	Pb	3,800
Fe (total)	23.3	Zn	3,000
Mn (total)	0.08		
K ₂ O	0.35		
Na ₂ O	0.12		
S	25.2		

TABLE 8. Concentration ranges of major and trace elements in leachates from Mattabi Mining Co. tailings before addition of the slag

Major Elements	Conc. range mg/L
Al	281 - 376
Ca	524 - 741
Fe	2,123 - 2,615
K	0.62 - 0.64
Mg	57 - 165
Na	0.18 - 0.21
Mn	6.20 - 6.24
Trace elements	
Cd	0.95 - 1.01
Co	3.59 - 4.72
Cr	0.62 - 0.72
Cu	82.85 - 120.60
Ni	1.19 - 1.27
Pb	21.42 - 30.68
V	0.66 - 0.96
Zn	102.80 - 107.00
pH of the leachates:	2.24 to 2.47

TABLE 9. Concentrations of major and trace elements in leachates from Mattabi Mining Co. tailings after addition of the slag (concentrations in mg/L)

Element	Test No.			
	1	2	3	4
Al	0.73	0.51	1.58	1.54
Ca	327	171	1,031	912
Fe	0.03	0.05	0.62	0.93
K	0.07	0.02	0.16	0.22
Mg	0.02	<0.01	0.58	1.33
Na	1.38	0.04	1.87	0.36
Mn	0.03	0.03	0.10	0.12
Cd	<0.01	<0.01	0.04	0.06
Co	<0.01	<0.01	0.09	0.12
Cr	0.04	0.03	0.18	0.21
Cu	0.04	0.03	0.38	0.70
Ni	<0.01	<0.01	0.15	0.19
Pb	0.50	1.17	3.89	3.92
V	<0.01	<0.01	0.36	0.47
Zn	0.20	0.12	0.71	1.12
pH:	11.57	11.67	11.86	12.17

TABLE 10. Concentrations of major and trace elements after leaching the slag with distilled water (d.W.) and leachate from untreated Mattabi Mining Co. tailings (concentrations in mg/L)

Element	700 ml d.w.	1,600 ml d.w.	900 ml leachate
Al	2.17	1.42	1.84
Ca	1,319	746	2,355
Fe	0.75	0.03	0.44
K	0.42	0.28	1.97
Mg	0.71	0.02	0.56
Na	1.14	1.78	0.56
Mn	0.14	0.06	0.21
Cd	0.05	0.01	0.20
Co	0.11	0.02	0.51
Cr	0.24	0.06	0.67
Cu	0.43	0.02	0.75
Ni	0.19	0.02	0.85
Pb	0.90	0.13	3.61
V	0.47	0.03	1.82
Zn	0.90	0.37	1.08
pH:	12.10	12.20	12.12

TABLE 11. Permeability of tailings and slag

Mix of tailings and slag < 0.5 mm at ratio 20:1

Time (days)	k (cm/s)*10 ⁻⁴		
1	2.46	1.56	1.64
2	2.19	1.43	1.63
3	2.31	1.42	1.52
4	2.02	1.22	1.19
5	1.88	1.10	1.15
6	2.26	1.12	1.15
7	2.39	1.58	1.23
8	2.08	1.30	1.24
9	1.75	1.26	1.20

190 g of tail. + mix 10 g of tail. and 10 g of slag < 2 mm on the top

Time (days)	k (cm/s)*10 ⁻⁴	
1	1.61	1.62
2	1.82	1.56
3	2.16	1.87
4	1.87	1.45
5	1.87	1.4
6	1.91	1.58
7	1.84	1.87
8	1.81	1.86
9	1.73	1.60

Timmins tailings

Time (days)	k (cm/s)*10 ⁻⁵	*k (cm/s)*10 ⁻⁴	
1	6.80	1.31	7.21
2	5.46	1.15	6.22
3	5.36	1.17	5.84
4	4.79	1.14	5.73
5	6.03	1.08	5.98
6	5.34	1.09	5.96
7	5.61	1.17	5.88
8	4.95	1.03	5.08
9	4.86	1.04	5.12

100 g of tail. + mix of 100 g of tail. and 10 g of slag < 0.5 mm on the top

Time (days)	k (cm/s)*10 ⁻⁴	
1	1.8	1.53
2	1.76	1.22
3	1.4	1.12
4	1.41	1.07
5	1.47	1.06
6	1.26	1.01
7	1.22	1.01
8	1.22	1.02
9	1.21	1.01

The slag < 5 mm on the top at ratio 10:1

Time (days)	k (cm/s)*10 ⁻⁴
1	1.09
2	1.13
3	1.09
4	1.11
5	1.12
6	1.12
7	1.29
8	1.25
9	1.20

100 g of tail. + mix of 100 g of tail. and 5 g of slag + 5 g of slag < 0.5 g on the top

Time (days)	k (cm/s)*10 ⁻⁴	
1	1.63	1.63
2	1.42	1.32
3	1.46	1.25
4	1.13	1.21
5	1.35	1.36
6	1.27	1.32
7	1.2	1.32
8	1.22	1.42
9	1.28	1.44

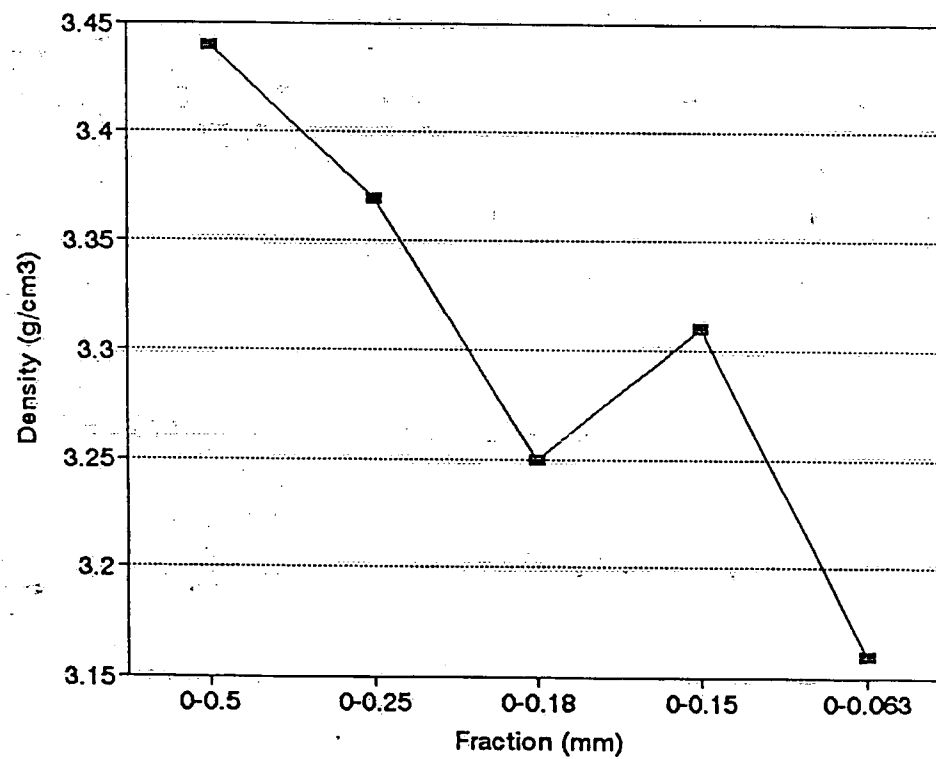
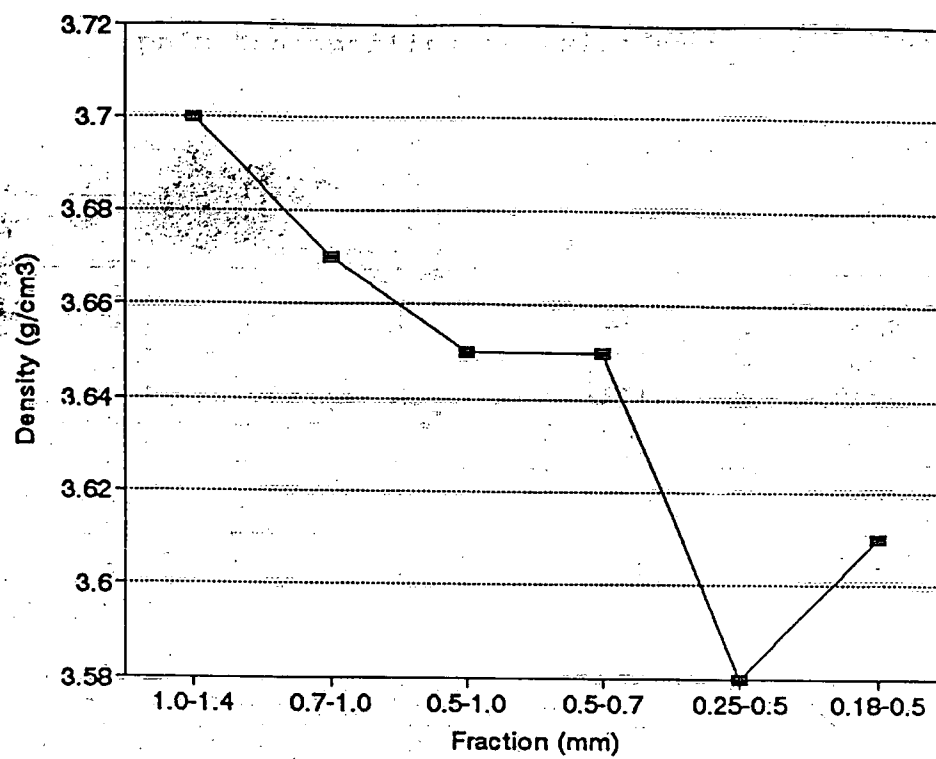


FIGURE 1. Density of slag

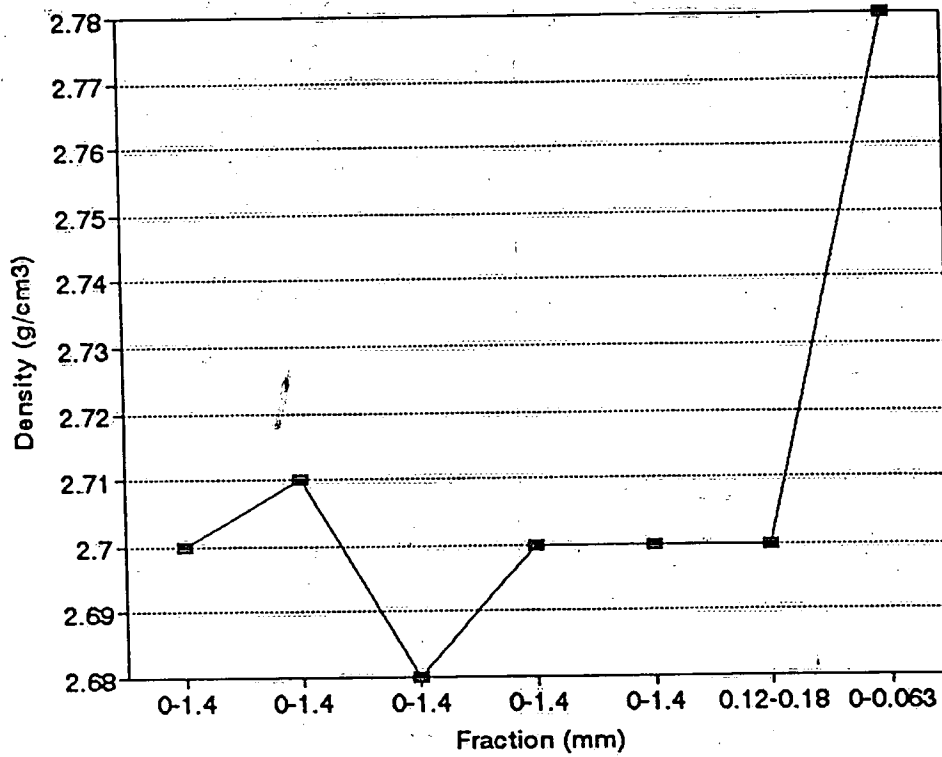


FIGURE 2. Density of Timmins Mining Co. tailings

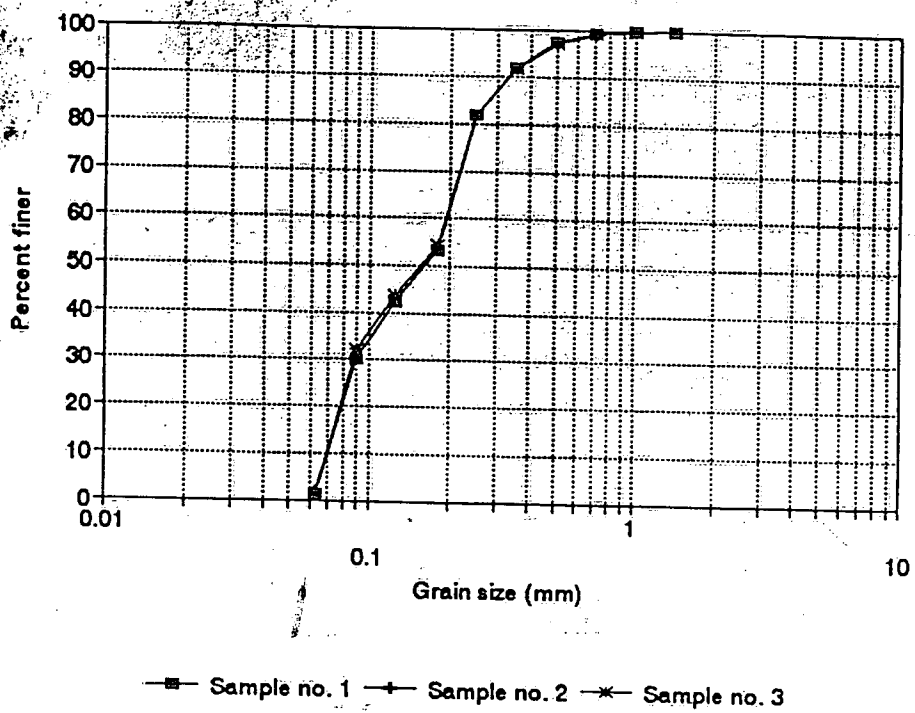


FIGURE 3. Grain size distribution of Timmins Mining Co. tailings

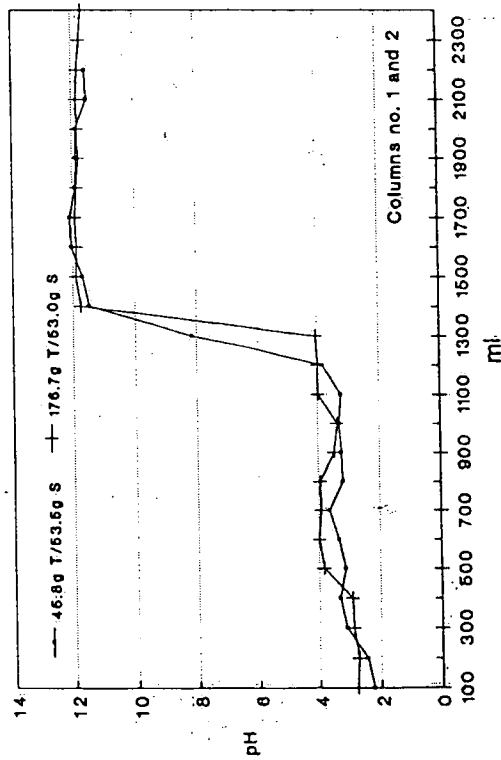
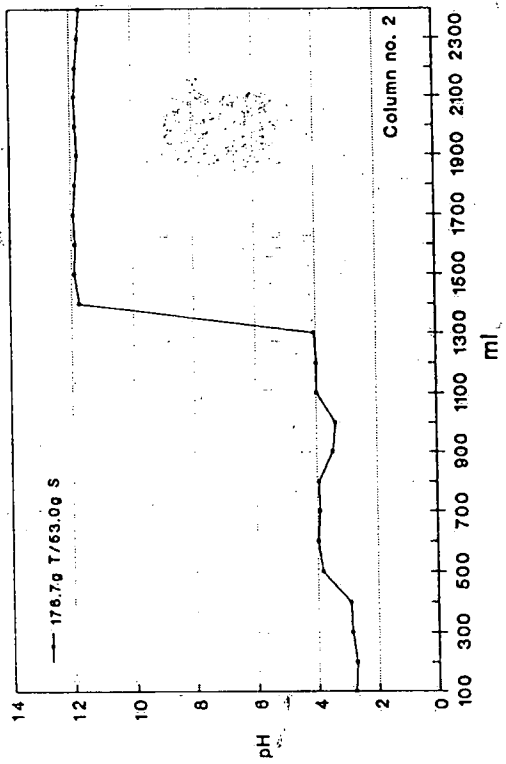
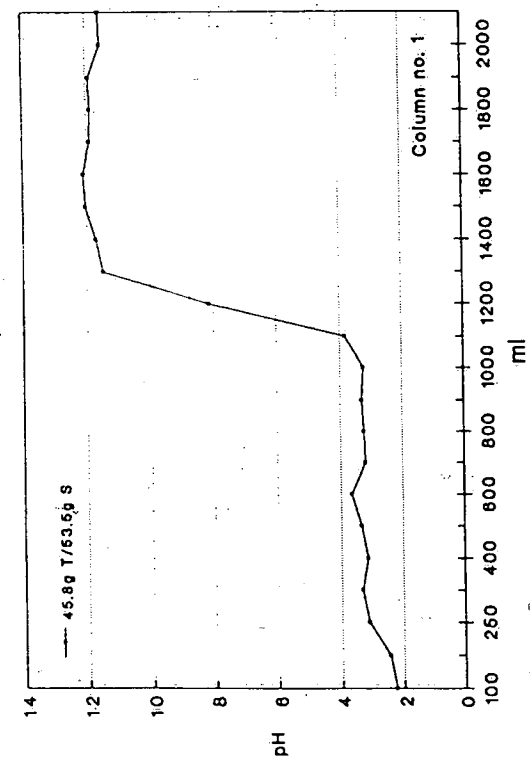
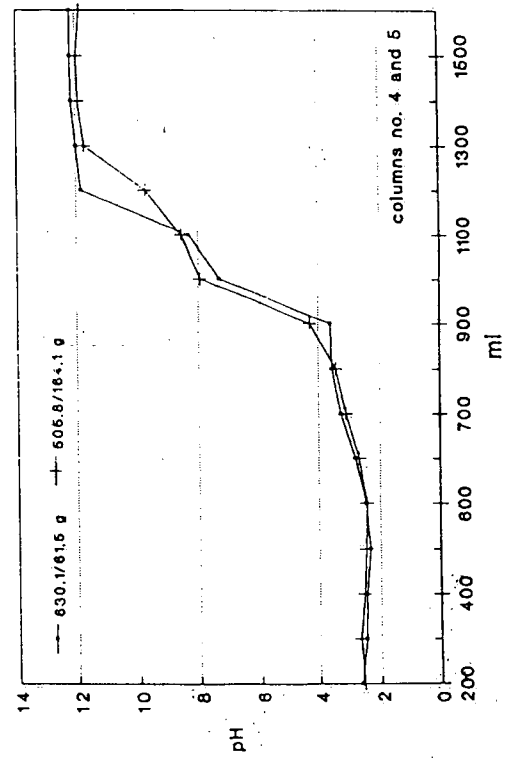
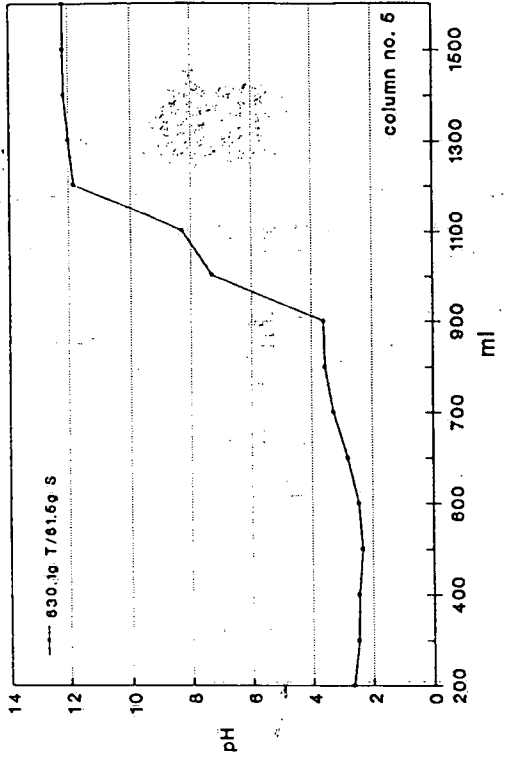
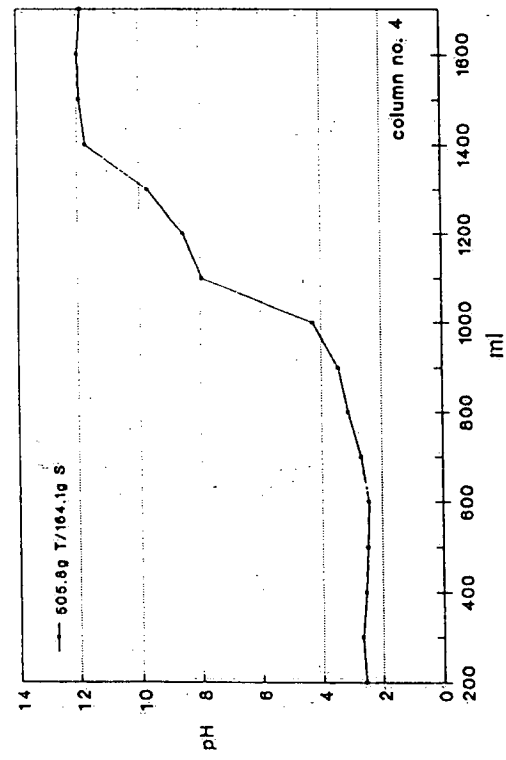


FIGURE 4. Changes in pH of the leachates from Mattabi Mining Co. tailings with slag applied on the top of the tailings (3.5 cm inside diameter tubes)



Mattabi Tailings + BOF Slag
 Leaching with distilled water
 Columns no. 4,5 - 5.0 cm diameter

FIGURE 5. Changes in pH of the leachates from Mattabi Mining Co. tailings with slag applied on the top of the tailings (10.5 cm inside diameter tubes)

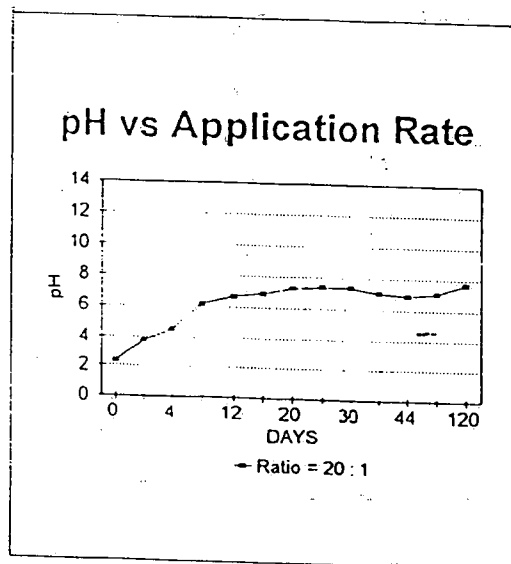
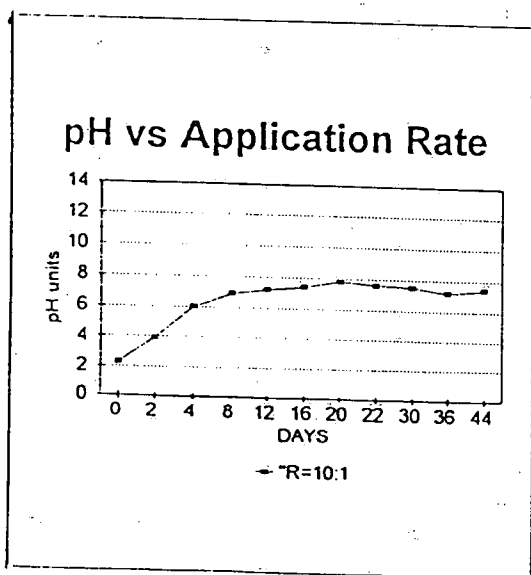
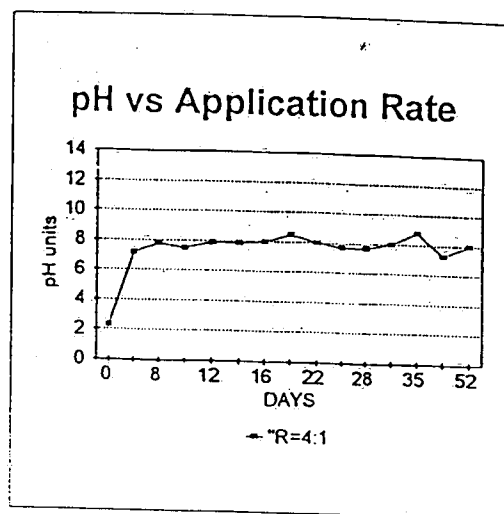
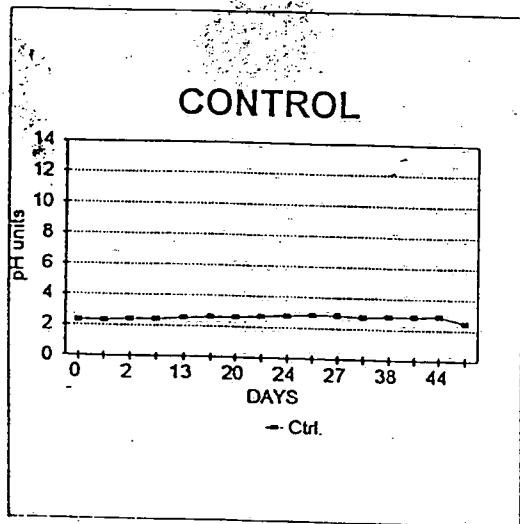


FIGURE 6. Changes in pH of the leachate from different mixtures of Timmins Mining Co. tailings and slag

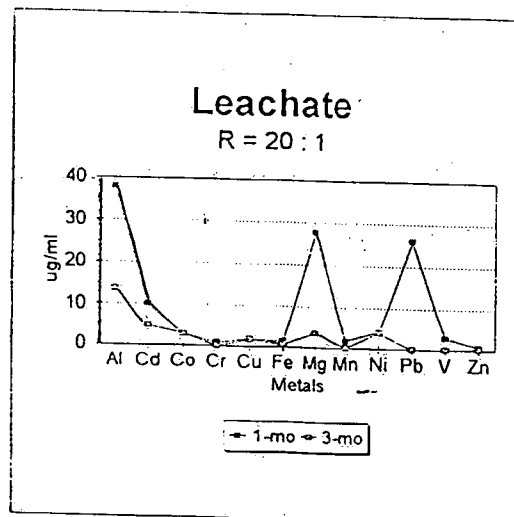
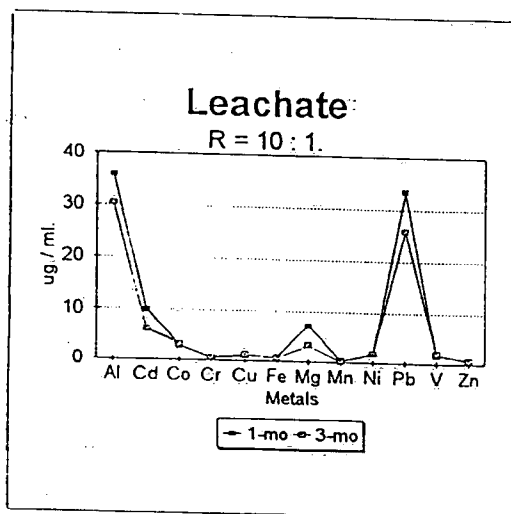
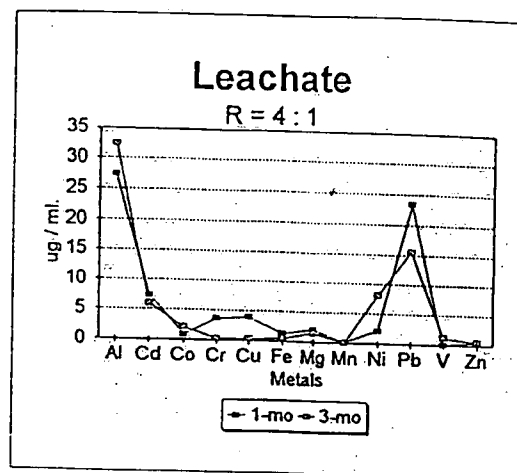
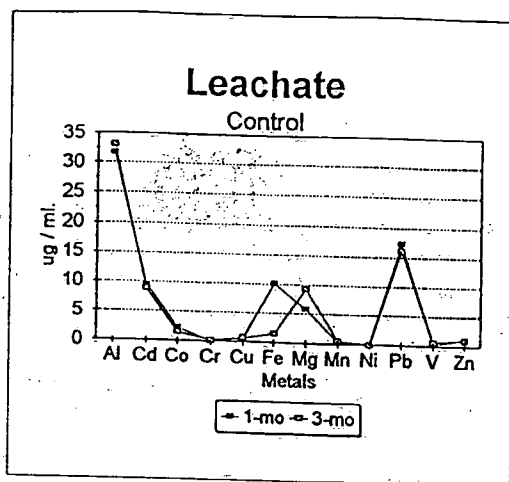
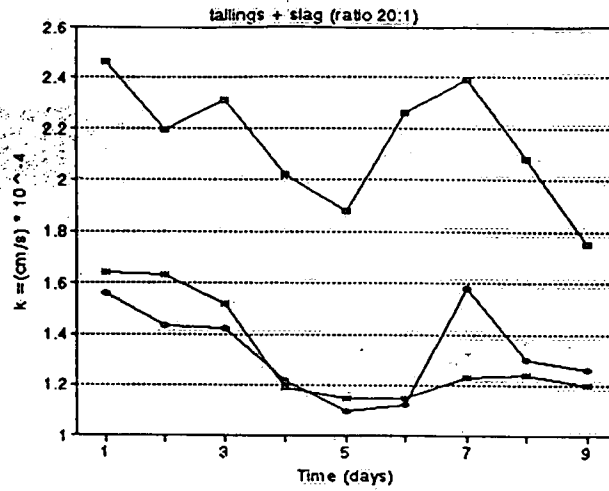
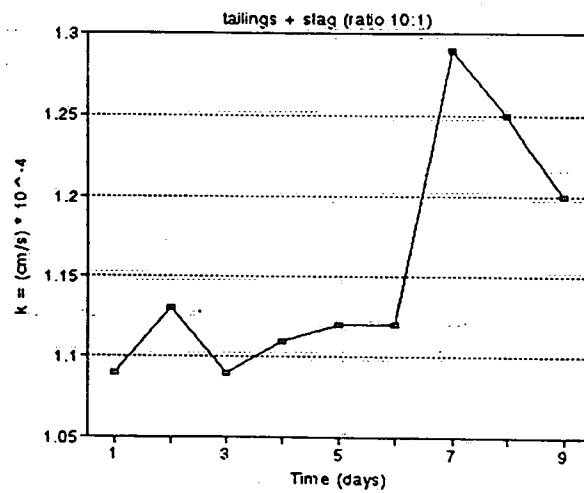
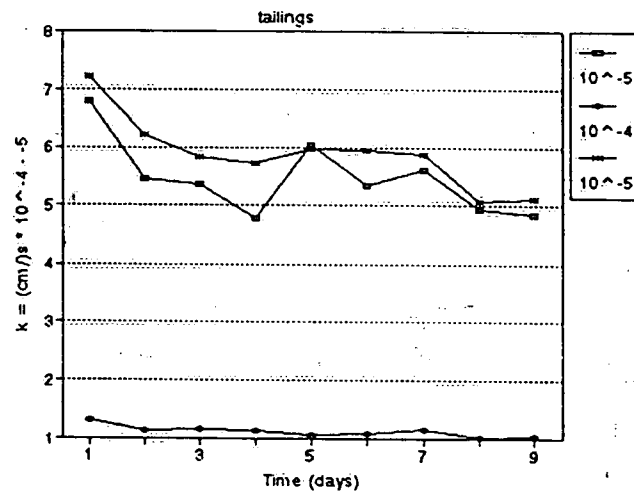


FIGURE 7. Concentrations of major and trace elements in leachates from different mixtures of Timmins Mining Co. tailings and slag after 1 and 3 months of leaching



Mix of 200 g tailings
and 10 g of slag



200 g of tailings and
20 g of slag 0-5 mm
on the top

FIGURE 8. Permeability of tailings and slag

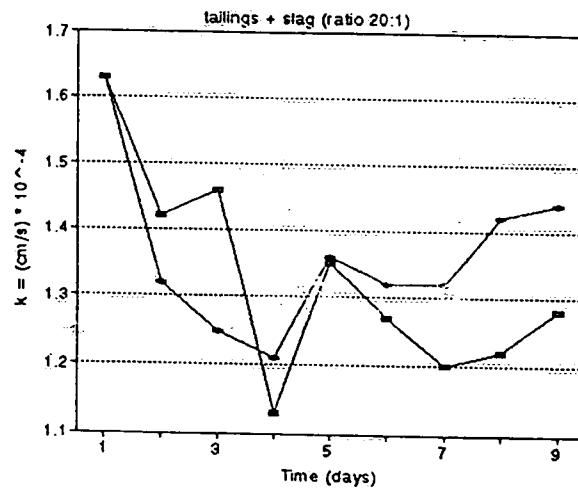
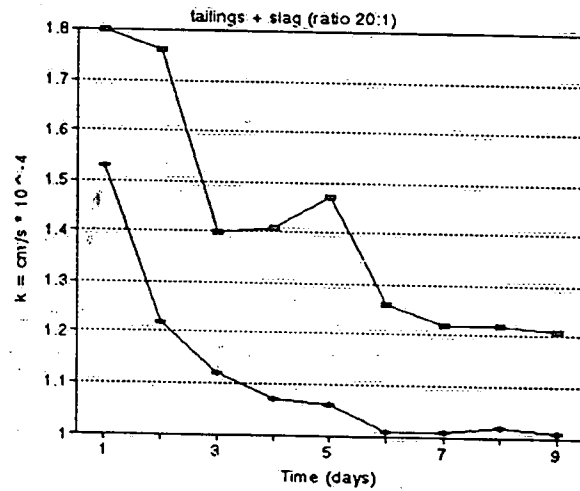
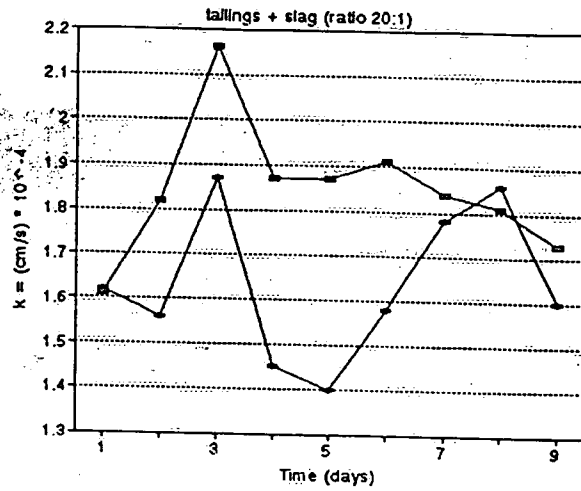


FIGURE 9. Permeability of a mixture of tailings and slag (ratio 20:1)

APPENDIX 1

TABLE 1. Five-steps program used in sample digestion by microwave

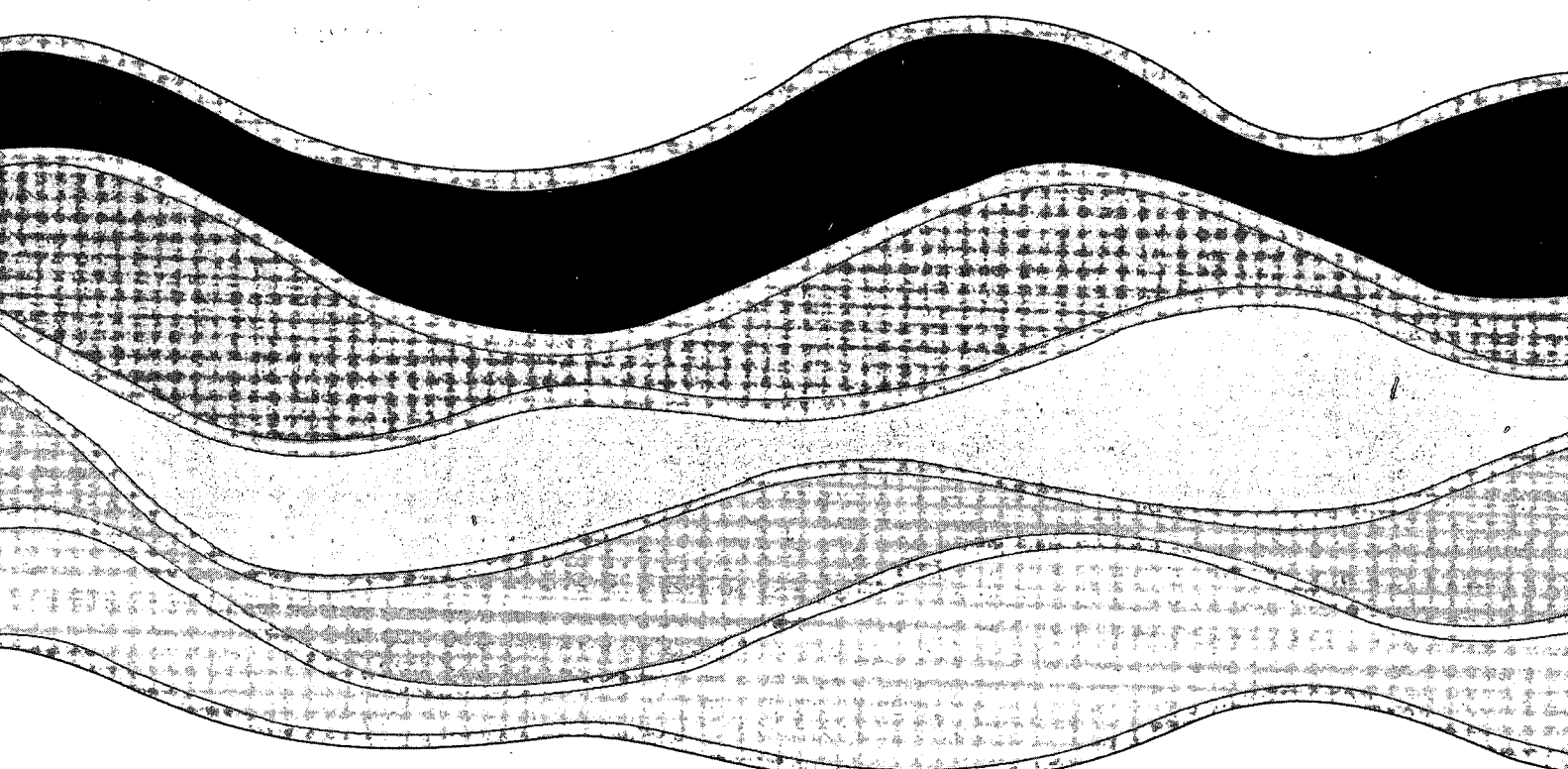
Stages	Pressure (psi)	Dwell Time (min)
1	25	3
2	50	3
3	75	3
4	100	5
5	125	5

TABLE 2. ICP-AES operating conditions

RF power	1100 W
Coating gas	0.3 L/min
Auxiliary gas	0.0 L/min
External flow (cooling)	12 L/min
PMT high voltage	650 V
Sample uptake rate	1.2 ml/min
Intergration time	1.0 sec
Viewing height	15.5 mm

TABLE 3. ICP-AES detection limits

Trace element/ Wavelength (nm)		Detection limit (ug/L)
Al	394.4	6
Cd	226.5	34
Co	228.6	9
Cr	267.7	9
Cu	324.7	10
Fe	259.9	12
Mg	279.5	3
Mn	257.6	2
Ni	231.6	20
Pb	220.3	25
V	292.4	2
Zn	213.8	9



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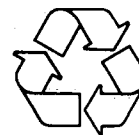
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