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Samatosum

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MINERALIZATION AND METALLURGY OF THE SAMATOSUM PROJECT

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## ABSTRACT

The Samatosum Project is a base and precious metal deposit located in south central British Columbia. A dilute ore reserve of 766, 682 tonnes grading an average 1.1% copper, 3.0% zinc, 1.4% lead, 833 gram per tonne silver and 1.6 gram per tonne gold is available. The base metals occur as tetrahedrite, chalcopyrite, sphalerite and galena. Virtually all of the silver and gold are associated with the tetrahedrite.

The mill is operated at a rate of 21 tonnes per hour. Sulphur dioxide is used as a zinc depressant in the grinding circuit. Production of a Cu/Pb/Ag concentrate is accomplished by conventional flotation. This concentrate is then separated into Cu-Ag and Pb concentrates through a reverse flotation circuit utilizing sulphur dioxide, starch and sulphuric acid as lead depressants.

The tailings from the Cu/Pb/Ag circuit are treated with copper sulphate to activate the zinc. Production of a zinc concentrate is accomplished by conventional flotation. Dewatering of the concentrates to 6% moisture is done with a LAROX filter press.

Tailings effluent is treated with lime and sodium sulphide to precipitate out the heavy metals. The resulting sludge is treated in DYNASAND sand filtration units to produce a quality effluent.

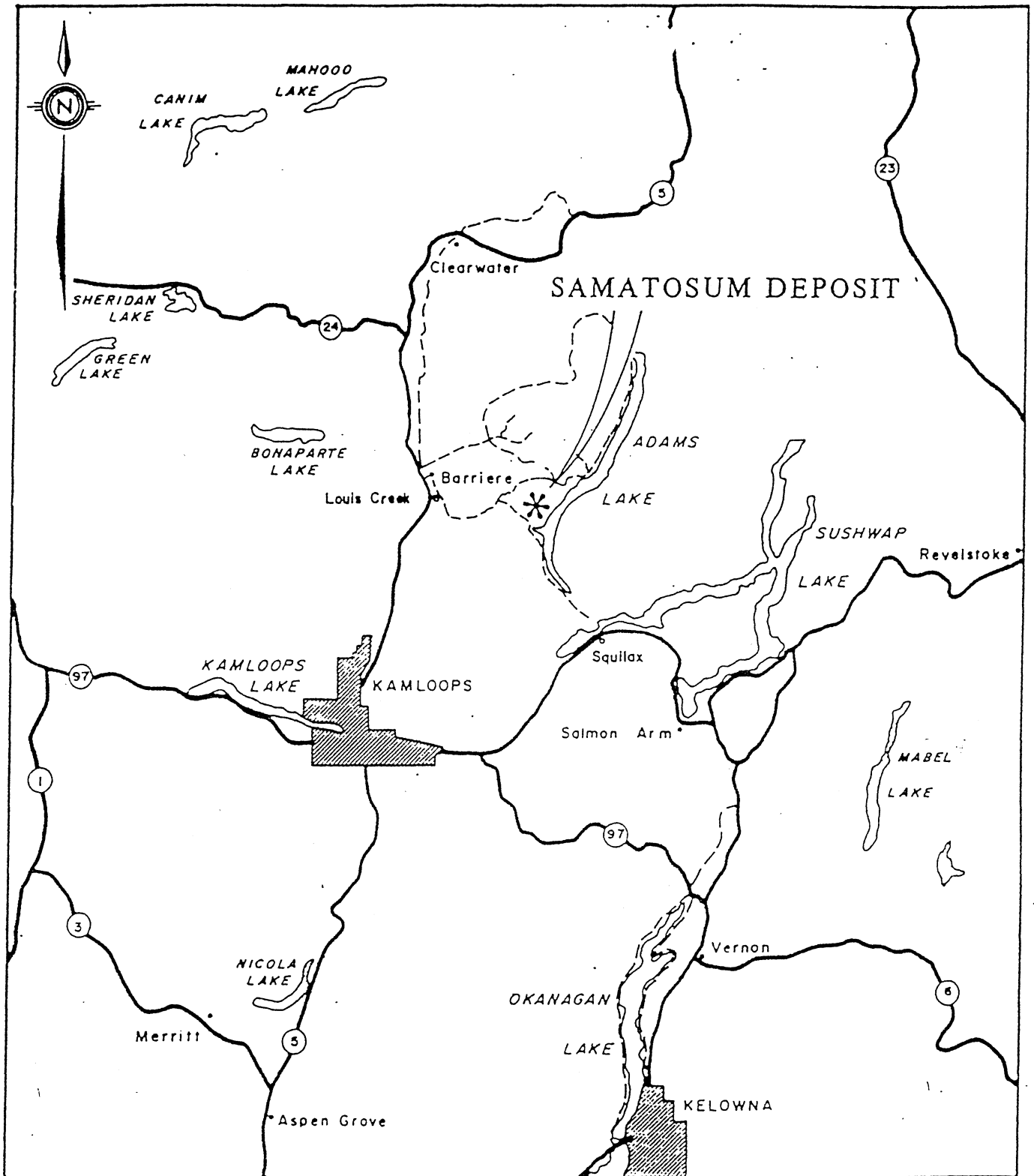
## Introduction

The Samatosum Project is a joint venture between Minnova Incorporated (70%) and Rea Gold Limited (30%). The property is a base and precious metal deposit located 35 kilometres east of Barriere and 95 kilometres north of Kamloops in south central British Columbia. (see figure 1).

The deposit was discovered in July 1986, after 12 months of extensive diamond drilling. A diluted ore reserve of 766,682 metric tonnes, grading on average 1.1% copper, 3.0% zinc, 1.4% lead, 833 gram per tonne silver and 1.6 gram per tonne gold, is available.

Feasibility studies for the project were carried out during the first 6 months of 1988. A brief outline of the studies is presented below:

Mineralogy studies were conducted by Dr. D.J.T. Carson of Norex in order to predict the metallurgical response of the minerals present. From this work, it was established that the important ore minerals of the Samatosum Project are tetrahedrite, sphalerite, galena, chalcopyrite and electrum while the gangue minerals are pyrite, quartz and dolomite. A viable mine plan of the ore body was also determined at this time.



SAMATOSUM DEPOSIT  
**LOCATION MAP**

FIGURE 1

Pilot Plant test work on the Samatosum ore was conducted by Ken Stowe of the Noranda Technology Centre, at the Lakefield Research Facility. Approximately 40 tonnes of ore grading 2.64% Cu, 2.08% Pb, 14.8% Zn, 2496 gram per tonne Ag and 6.94 gram per tonne Au were received at Lakefield on June 10, 1988. This ore was crushed to minus 3 mesh by jaw and impact crushing.

A total of 15 circuit configurations were tested at Lakefield. This test work generated the flow sheet and the choice of reagents used at the Samatosum Project. Results from the configuration on which the Samatosum Mill is based are given in Table 1 below.

TABLE 1 - PILOT PLANT RESULTS UPON WHICH THE  
 =====  
 SAMATOSUM PROJECT WERE BASED.  
 =====

STREAM	ASSAYS % OR G/T				RECOVERIES %			
	Cu	Pb	Zn	Ag	Cu	Pb	Zn	Ag
FLOTATION FEED	2.45	1.77	14.17	2363	100	100	100	100
Cu-Ag CONCENTRATE	24.3	5.52	11.9	24359	77.4	23.2	6.5	81.3
Pb CONCENTRATE	8.19	23.77	23.94	8318	11.2	42.2	5.6	11.9
Zn CONCENTRATE	1.06	1.61	58.9	628	8.9	21.1	83.6	5.4
TOTAL TAILS	0.09	0.3	0.89	48	2.6	16.9	7.9	1.7

The pilot plant studies identified the 3 Concentrates and grades that would be produced. Noranda Sales was then able to market the concentrates and obtain smelter contracts for the products from the Samatosum Project.

Effluent Treatment was studied by Paul Spira of the Noranda Research Centre. This led to development of the flowsheet and reagents for the Effluent Treatment Facility required to meet environmental guidelines and regulations.

The final project plan was submitted to the British Columbia Mine Steering Committee, with their Approval in Principle being granted on September 30, 1988. Construction of the plant commenced in October of 1988, with Proton Systems being appointed the engineering and construction management firm for the project. Positions for the project were filled with local personnel where ever possible, except when the experience and qualifications for the position could not be met locally.

Pre-stripping of the over burden for the open pit mine commenced in March of 1989. On May 21, 1989 ore was put through the mill at a rate of 10.5 metric tonnes per hour, one half of the designed capacity of the mill. By the end of June, 1989, full tonnage of 21 metric tonnes per hour was achieved. Commissioning and start up of the Effluent Treatment Plant took place during the early part of October, 1989, with quality effluent being discharged on October 17, 1989.

Total capital cost for the project was \$31.4 million dollars, including the effluent treatment plant.

A discussion of the Mineralogy and Metallurgy of the Samatosum Project now follows.

### Mineralogy of the Samatosum Deposit

The ore body is divided into 3 zones, based on the mineralization present. Zone I, located in the upper open pit is a massive sulphide ore body, while Zone II, located in the lower open pit is mainly a disseminated sulphide ore body. Zone III, contains an underground ore body very similar to the Zone II ore body.

The ore bearing sulphide minerals found at the Samatosum Deposit are:

- i). **Tetrahedrite:** This is the main silver, gold and copper bearer in the ore. It is medium to coarse grained and has a fairly consistent composition throughout the ore body, averaging 36.1% Cu, 3.4% As, 23.3% Sb, 24.7% S, 2.1% Fe, 4.9% Zn and 3.8% Ag.
- ii). **Chalcopyrite:** This medium grained copper bearer is found mainly in Zone I of the deposit. Due to the fracturing of the ore body and the close proximity to surface, severe oxidation is present in much of the chalcopyrite.
- iii). **Galena:** The lead bearing sulphide throughout the deposit is galena. It is very fine to coarse grained and has been seen to contain up to 1% silver.
- iv). **Sphalerite:** This is the main zinc bearing mineral in the ore. It is medium to coarse grained and contains 65.4% zinc with very low iron values. Much of the sphalerite located in Zone I contains "chalcopyrite disease", that is finely intergrown chalcopyrite within the sphalerite ore.

The main gangue minerals are pyrite (11%), quartz (32%) and dolomite (19%). A very small portion of the ore in Zone I is present as covellite. When encountered, preactivation of the zinc minerals occurs.

SAMATOSUM PROJECT - STOCKPILE CONFIGURATION

HG-1

HG-2

MG-1

MG-2



$Pb/Cu < 2$

$Pb/Cu > 2$

$Pb/Cu < 2$

$Pb/Cu > 2$

AND

AND

$Pb + Cu + Zn > 12$

$Pb + Cu + Zn < 12$

OR

AND

$Ag > 3000 \text{ g/t}$

$Ag > 250 \text{ g/t}$

$Ag < 3000 \text{ g/t}$

FIGURE 2

Due to the complex nature of the ore body at the Samatosum Project, sulphide mineralization tends to occur in pockets. These pockets, which can be several cubic meters in size, tend to be a high grade deposit of copper, lead or zinc, with essentially all of the silver values associated with the tetrahedrite. In order to minimize this effect and maintain a reasonable blend to the mill, 3 stock piles have been formed. These stock piles and their parameters are shown in figure 2.

### Process Description

Ore, which is mined by Ledcor, a mining contractor, is hauled to the appropriate stock pile with 35 and 50 tonne trucks. The ore is then fed to the grizzly, (which has 18 inch openings), with a Caterpillar 980 loader. Correct proportions are used to ensure a proper blend to the mill. Any over size material is broken with a small rock breaker mounted on a backhoe. (see figure 3).

The minus 18 inch ore is passed to a 75 metric tonne hopper before being fed to the 36" x 48" Traylor jaw crusher via an apron feeder. The output from the jaw crusher, which is minus 3 inch material, is fed to a vibrating screen with a bottom deck opening of minus 1/2 inch. The over size material from the screen is further reduced in size via a 5 1/2 foot Simons short head crusher set at 3/8 inches. The output from the short head crusher is again passed over the screen. The minus 1/2 inch material from the screen is stored in a 1200 tonne (live) fine ore bin. The crushing plant is oversized and thus allows a 5 day, 8 hour per day operating schedule.

Ore is fed to the mill via 2 slot feeders at a rate of 21 tonnes per hour. Grinding is done in a 8' x 10' ball mill, charged with 2 and 3 inch steel balls. The mill density is maintained at 73 to 75 per cent solids (Ore SG-2.9). Sulphur dioxide is added to the ball mill grinding water at a rate of 0.5 to 0.7 kilograms per tonne. The sulphur dioxide is used to help depress the zinc during the copper, silver and lead recovery stage.

The ball mill, is in closed circuit with a single cyclone. Overflow from the cyclone is controlled at 36 to 38 per cent solids with a minus 200 mesh fraction of 58 to 62 % . A relatively coarse grind is used due to the coarse grained nature of the sulphide minerals present. Areofloat 242 promoter is added to the cyclone overflow in order to increase lead recovery.

# SAMATOSUM PROJECT - FLOW SHEET

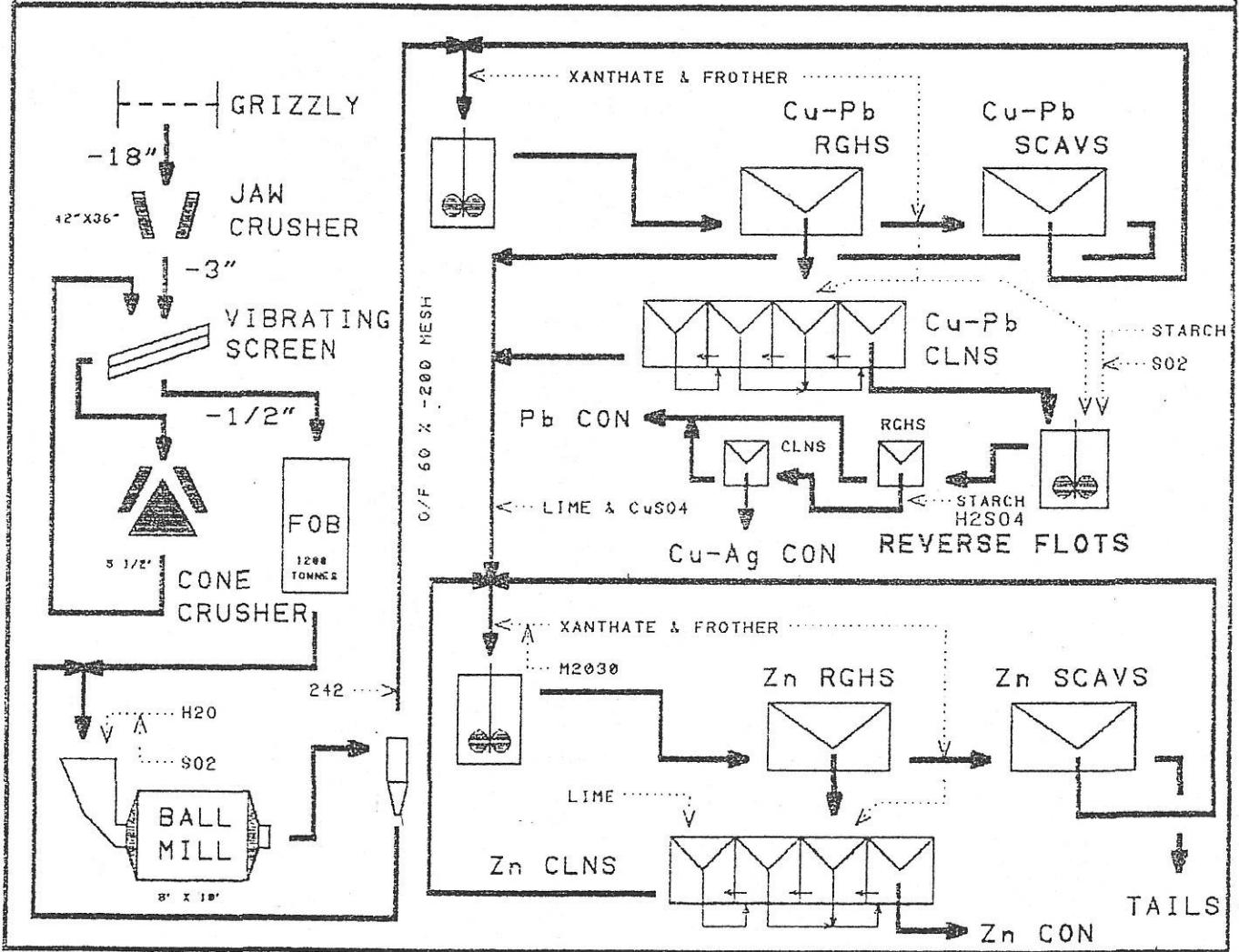


FIGURE 3



The flotation feed is then conditioned with potassium amyl xanthate. Frother, in the form of MIBC, is added to the conditioner. The conditioned material is fed to 7 Outokumpu cells configured as Cu-Ag/Pb Roughers, Mids and Scavengers. Xanthate and frother are added to the tails from the Roughers before feeding the Mids and Scavengers. Concentrate from the Scavengers is returned to the conditioner while the tails are directed to the Zinc Circuit.

The Cu-Ag/Pb Rougher Concentrate can contain up to 50 per cent of zinc due to the hard pull required to recover the copper, silver and lead values. The concentrate is again dosed with xanthate and frother before being introduced into the Cu-Ag/Pb Cleaners. The cleaners consist of 4 Denver cells working in a counter current fashion. Gangue material and zinc are sent as tails to the Zinc Circuit while the Cu-Ag/Pb Cleaner Concentrate is sent to the Cu-Ag/Pb Separation Circuit.

The use of reagents in the Cu-Ag/Pb Roughers, Scavengers and Cleaners must be minimized in order to prevent activation of the lead in the Cu-Ag/Pb Separation Circuit and to minimize recovery of zinc to the Cu-Ag/Pb Circuit. More reliance is placed on mechanical means (eg. - air and pulp bed level) than on the use of reagents to meet recoveries.

The Cu-Ag/Pb Cleaner Concentrate is conditioned with starch (Casco British Gum 9760), sulphur dioxide, sodium ethyl xanthate and frother. Starch and sulphur dioxide are added in order to depress the lead during the separation step, while the xanthate and frother are used to collect the copper and silver minerals. The conditioned pulp is sent to 2 Denver reverse cells arranged as roughers. Starch and sulphuric acid are added to the rougher concentrate which is then cleaned in 2 Denver cells. A pH of 3.8 to 4.2 is maintained throughout the reverse flotation circuit.

The concentrate pulled during the cleaning step reports to the Cu-Ag Concentrate, while the combined tails from the roughers and cleaners report to the Pb Concentrate.

Copper sulphate is added to the combined tails from the Cu-Ag/Pb Scavengers and Cleaners in order to activate the zinc. Hydrated lime slurry is added in order to bring the pH up to 10.5 to 10.8. The Zinc Flotation Feed is conditioned with xanthate, frother and Minerac M2030 prior to being fed to the Zinc Circuit. The conditioned material is fed to 7 Outokumpu cells configured as Zn Roughers, Mids and Scavengers. Concentrate from the Scavengers is returned to the conditioner while the tails are directed to the Mill Total Tails Stream.

The Zn Rougher Concentrate is again dosed with xanthate and frother before feeding the Zn Cleaners. The Zn Cleaners consist of 4 Denver cells working in a counter current fashion. The pH in the Cleaner Circuit is maintained at 11.2 to 11.8 via the addition of hydrated lime slurry. Tails from the Zn Cleaner are recirculated to the conditioner while the concentrate reports to the Zn Concentrate.

#### Dewatering

Each concentrate reports to its own individual thickener (see figure 4). Flocculant (Cyanamid N-300-S) is added to the thickener feeds as a settling agent. Lime is added to the Pb thickener feed in order to help depress the fine lead. The thickeners are pumped out on an intermittent basis to the stock tanks, with the underflow being maintained at 50 per cent solids. (SG of solids = 4.0 to 4.2). The stock tanks are set up as backfill tanks and only slurried, via high pressure water addition, when filtering is required.

As stock tank levels dictate, each concentrate is pumped individually to the Larox Filter Press, where the concentrate is dewatered to 6 per cent moisture. The Cu-Ag Concentrate is packaged into 1.8 tonne bags. Twenty five percent of the Cu-Ag Concentrate produced is shipped to DOWA in Japan while the remainder goes to SGM in Belgium. The Pb and Zinc concentrates are shipped by truck to Asarco in Montana and Cominco in Trail, respectively.

Gold and silver values are paid for in varying degrees in each concentrate. Copper, lead and zinc are paid for in their respective concentrates, while antimony and lead are paid for in the concentrate destined for SGM in Belgium and lead is paid for in the zinc concentrate.

#### Effluent Treatment

The Samatosum Project is committed to ensuring that all effluent leaving the tailings management area meets all environmental criteria established for the property. To this effect, an effluent treatment plant utilizing "Dynasand" filters will be employed. (see figure 5).

Effluent from the tailings management area is pumped to the first reactor where hydrated lime slurry is added to adjust the pH to 10.5 and thus precipitate the dissolved metals in the effluent. A second reactor follows, to add to the overall reaction time available. Sodium sulphide is added to the third reactor in order to control the dissolved level of cadmium. A total of 60 minutes of reaction time is available through the 3 reactors. The treated effluent is then passed to 3 Dynasand filters installed in parallel. The filters continuously backwash a portion of the sand bed with filtrate.

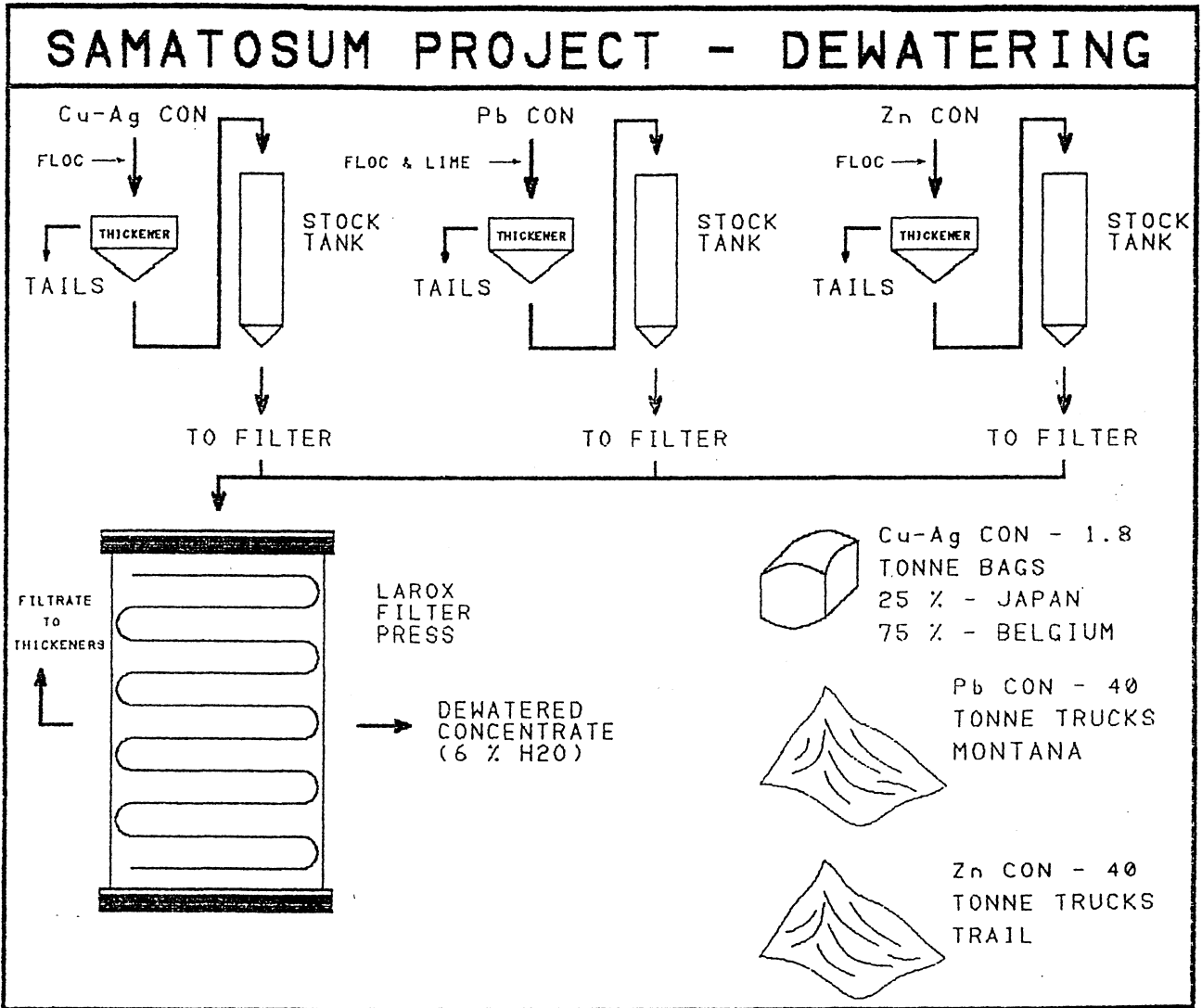


FIGURE 4

# SAMATOSUM PROJECT - EFFLUENT TREATMENT

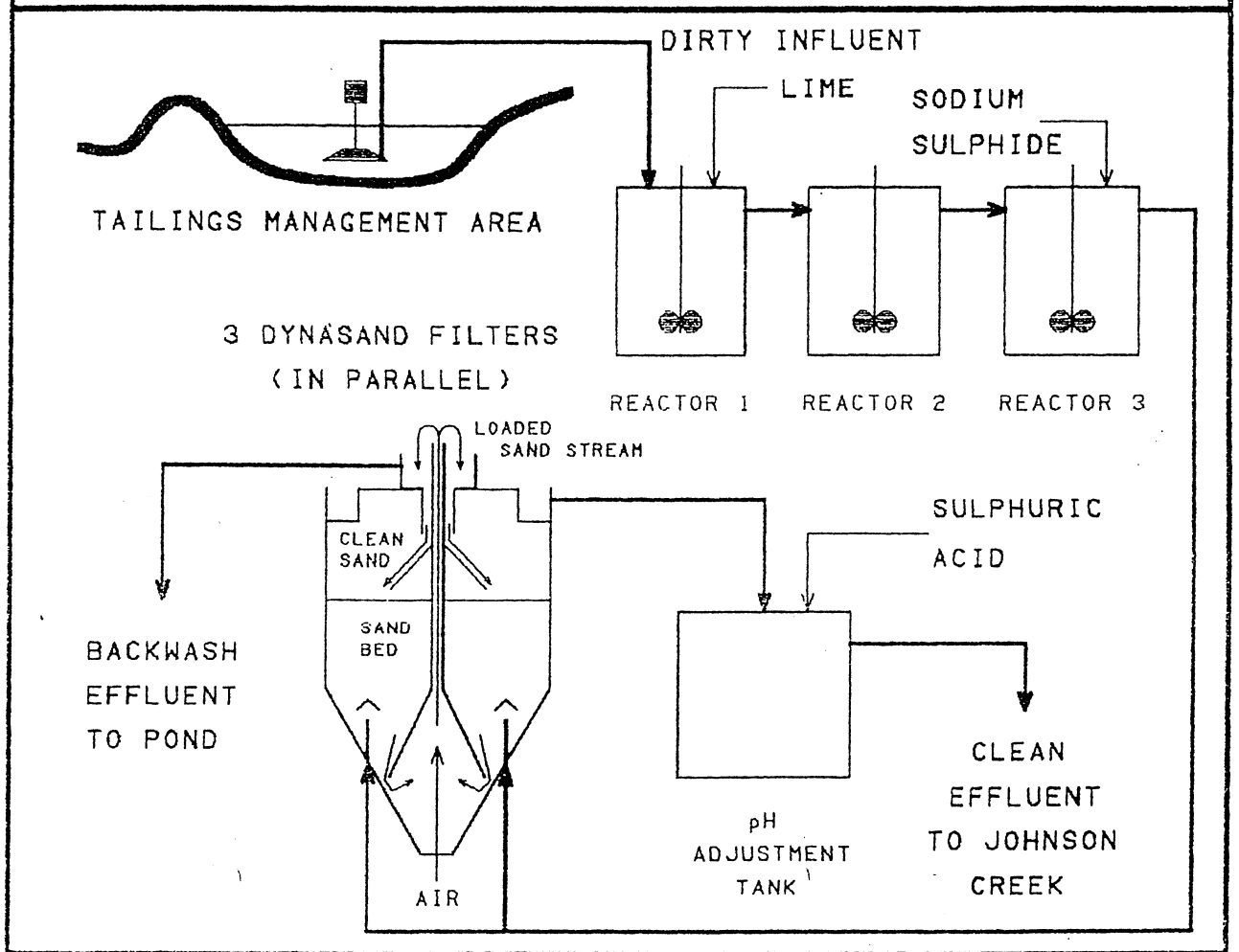


FIGURE 5

The backwash or reject stream is directed to the tailings pond for deposition . The clear filtrate is discharged to Johnson Creek by gravity, after the pH has been lowered to 7.0 with the addition of sulphuric acid.

The plant is capable of treating up to 150 cubic meters of effluent per hour. Discharge volume from the plant will be based on a dilution ratio of 1:10 to Johnson Creek. Startup and commissioning of the plant was carried out in early October, 1989. Discharge of final effluent to the environment commenced on October 17, 1989 at approximately 50 cubic meters per hour. Effluent quality has been well within the guidelines as shown below with all toxicity tests being passed.

TABLE 2 - Maximum Concentration in Effluent (ppm)

<u>Parameter</u>	<u>Maximum Permitted Concentration</u>
Sb	0.125
As	0.100
Cd	0.010
Cu	0.030
Fe	0.300
Pb	0.040
Zn	0.300
Ag	0.004
pH	6.5 to 9.0

Effluent treatment reagent costs are in the order of \$85 per 1000 cubic meters of effluent discharged. Consumption rates are shown in Table 3.

TABLE 3 - E.T. Plant Reagent Consumption

<u>Reagent</u>	<u>Consumption (Kg/cubic meter)</u>
Lime	0.0033
Sodium Sulphide	0.0056
Sulphuric Acid	0.0041

Capital cost for the Effluent Treatment Plant was \$1.1 million.

#### On Stream Analysis

The Samatosum Mill is equipped with 2 Outokumpu Courier 30 on stream x-ray analyzers. Each analyzer is equipped to analyze 5 sample streams for Cu, Pb, Ag, Zn, Fe and scatter. (Scatter is inversely proportional to the density of the sample stream).

Calibration of the OSA units is a complicated and involved procedure. Approximately 60 calibration samples must be taken for each stream. As a stream is sampled, the x-ray counts for Cu, Pb, Ag, Zn, Fe and Scatter are noted.

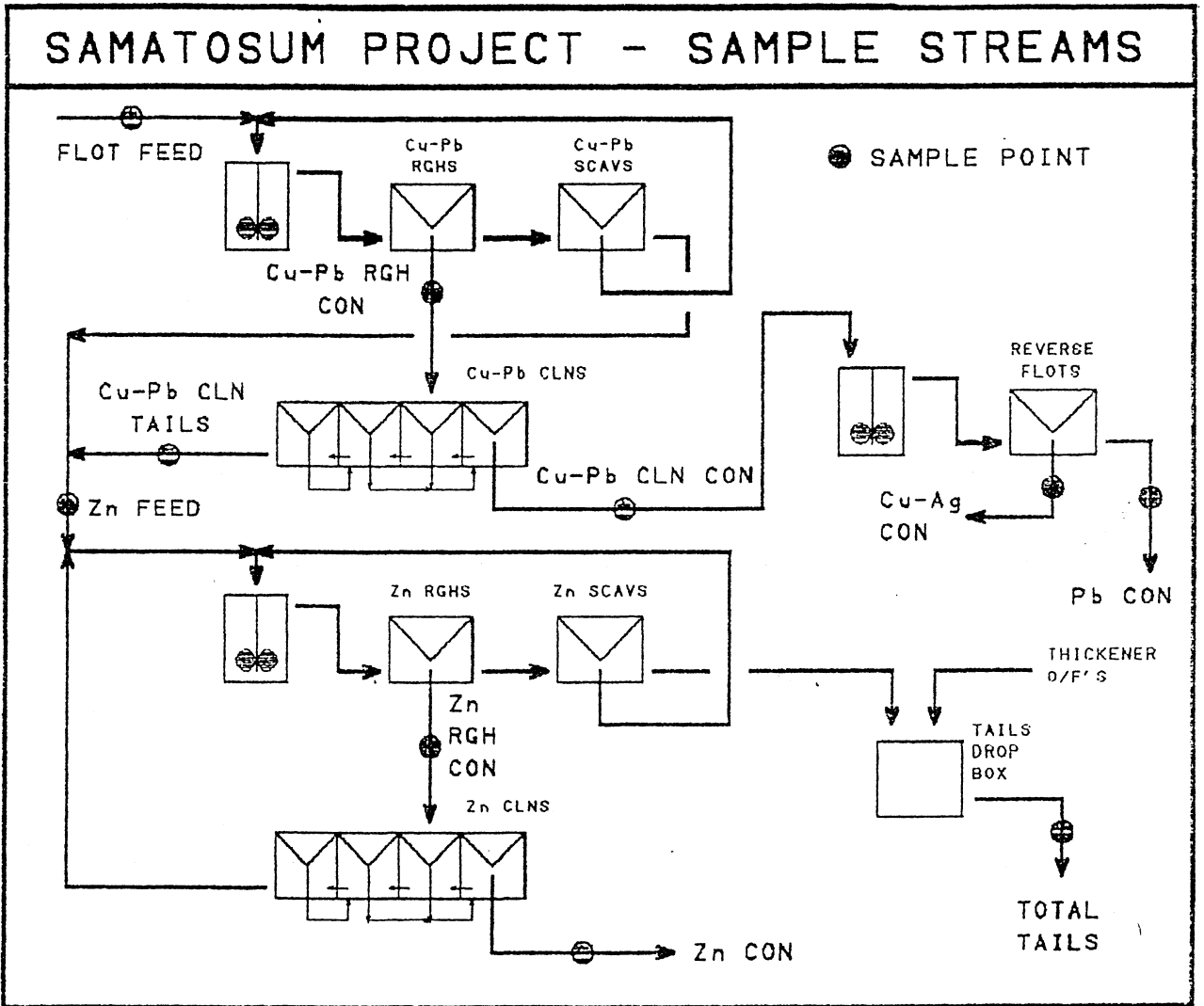


FIGURE 6

The sample is then analyzed for each of the elements in the assay lab. When enough samples have been taken for a particular stream, regression analysis is applied to determine the assay of each element versus the x-ray counts for each of the 6 variables.

eg. For the % Cu in the flotation feed the regression analysis indicates that:

$$\begin{aligned} \% \text{ Cu in Feed} = & C1 * N[\text{Cu}] * N[\text{Ag}] + \\ & C2 * N[\text{Cu}] / N[\text{Scatter}] + \\ & C3 * N[\text{Cu}] * N[\text{Pb}] + \\ & C4 * N[\text{Cu}] * N[\text{Zn}] + \\ & C5 * N[\text{Cu}] * N[\text{Fe}] + \\ & \text{INTERCEPT} \end{aligned}$$

where  $C_i$  = the slope as determined by the regression analysis.

$N$  [Element] = x-ray counts for the element.

INTERCEPT = the intercept value as determined by the regression analysis.

The 10 sample streams used at the Samatosum Mill were chosen to allow efficient metallurgical operation of the mill and to ensure valid mill balances could be calculated. (see figure 6). Each stream is assayed for Cu, Pb, Zn and Ag every 10 minutes and the assay results are displayed in the form of historical bar graphs on the OSA terminals. The flotation operator may change the time of the assay reporting from the base-line 10 minutes interval, to an hour, shift, day, week or month interval in order to determine the trends present. Up 10 years of historical data may be saved within the OSA unit.

Due to the wide fluctuations in grade coming into the plant (see table 4) monitoring of OSA assays, and more importantly the assay trends, is crucial to the smooth operation of the plant. By monitoring the trends, the operator can quickly react to changes in the circuit, making the appropriate corrections to reagents, etc.

TABLE 4 - FEED GRADE VARIATIONS

ELEMENT	FEED GRADE
Cu	0.4 TO 1.5%
Pb	< 1.0 TO 7.0%
Zn	3.0 TO 14.0%
Ag	600 TO 2500 g/t

## Metallurgical Problems and Current Operating Data

Mill production commenced on May 21, 1989 at 10.5 metric tonnes per hour. At this time, reagent additions were set based on the rates used at Lakefield during the pilot plant testing. Calibration of the on stream analyzer also commenced at this time. As time progressed, and the assays from the on stream analyzer become more reliable, fine tuning of the circuit commenced.

Three main metallurgical problems have been experienced to date, namely:

1. High recoveries of zinc to the Cu-Ag/Pb Circuit.
2. Poor separation of Cu-Ag/Pb in the Reverse Flotation Circuit.
3. Poor recoveries of copper, lead and silver to the Cu-Ag/Pb Circuit.

Pre-activation of zinc in the Cu-Ag/Pb circuit occurs periodically, regardless of the sulphur dioxide dosage to the ball mill. This problem, occurs due to the presence of covellite, oxidized chalcopyrite (or tetrahedrite to a lesser extent) or chalcopyrite disease of the sphalerite. The oxidized copper mineral tend to react in a similar fashion to copper sulphate. This causes pre-activation of the zinc in the Cu-Ag/Pb circuit. With chalcopyrite disease of the sphalerite, finely disseminated copper minerals are present in the sphalerite. Even with extremely fine grinding of the ore, separation of the copper from the zinc is not possible. The sphalerite with chalcopyrite disease then reacts as if it were copper mineral in the copper/lead circuit.

When these problems occur, sulphur dioxide addition is increased in order to reduce zinc activation. Normal circuit parameters are maintained in the Cu-Ag/Pb Rougher, Scavenger and Cleaner Circuits so as to maintain copper, silver and lead recoveries. The zinc that reports to the Cu-Ag/Pb Cleaner Concentrate is then attacked in the Reverse Circuit by increasing the sulphur dioxide and starch to this circuit. The combination of these 2 reagents effectively depresses the zinc into the Pb Concentrate, with slightly elevated zinc assays in the Cu-Ag Concentrate. This route results in lower zinc recovery to the zinc circuit, but does allow the recovery of copper, silver and lead to remain unchanged.

Poor separation of copper, silver and lead in the Reverse Flotation circuit occurs when the Pb to Cu ratio in the feed is fairly high. At this time, cross contamination of the Cu-Ag and Pb Concentrates occurs due to the over abundance of lead in the circuit. This cross contamination has been traced, through screen analysis, to the fine fractions in the 2 concentrates.



Over 85% of the lead reporting to the Cu-Ag Concentrate has been found to be minus 400 mesh material, while over 75 per cent of the copper reporting to the Pb Concentrate has been found to be minus 400 mesh material. Laboratory testing has shown that all starches do not react the same in the role of lead depression in the reverse flotation circuit. Through this test program, the initial starch used, Jaguar Guar Gum 8707 has been replaced with CASCO British Gum 9760. This material is more selective in depressing lead in the reverse flotation circuit.

Further laboratory testing has revealed that a pH of 3.8 to 4.2 must be maintained throughout the reverse flotation circuit. This is now accomplished via the addition of sulphur dioxide to the reverse circuit conditioner and sulphuric acid to the reverse cleaners. In order to maintain Cu recoveries to the Cu-Ag Concentrate, a selective Cu collector must be employed. Sodium ethyl xanthate has been found to be the required collector. With the above treatment scheme in place, Cu/Pb separation problems have virtually been eliminated.

Poor lead separation from the Cu-Ag Concentrate has also been traced to the presence of covellite coating the galena particles. When this occurs, the galena reacts as if it were copper in the reverse circuit and thus reports to the Cu-Ag Concentrate.

Poor recoveries of copper, silver and lead have been traced to two problems, namely oxidized ore and an ore type in which all minerals are intergrown with each other. Laboratory testwork and pilot plant studies are to be carried out on these ore types.

Year-to-date data from startup to December 31, 1989 is outlined in figures 7 to 14. Results are still improving and are nearing those obtained in the pilot plant at the Lakefield Research Facility. Reagent consumptions are shown in Table 5, while projected cost estimates for 1990 are given in table 6.

TABLE 5 - MILL REAGENT CONSUMPTION

TABLE 6 - MILL COSTS - 1990 BUDGET

REAGENT	CONSUMPTION (Kg/TONNE)	AREA	OPERATING COST (\$/TONNE MILLED)
Xanthate	0.054	MINE DEPARTMENT	67.05
Sulphur Dioxide	1.111		
Cu/Pb Promoter (242)	0.029	MILL DEPARTMENT	28.91
Zn Promoter (M2030)	0.058		
Frother	0.383	TOTAL COST	109.53
Copper Sulphate	1.153		
Starch	0.240		
Lime	3.646		
Grinding Balls	0.937		

FIGURE 7  
MILL FEED GRADE ASSAYS  
YTD VALUES for 1989

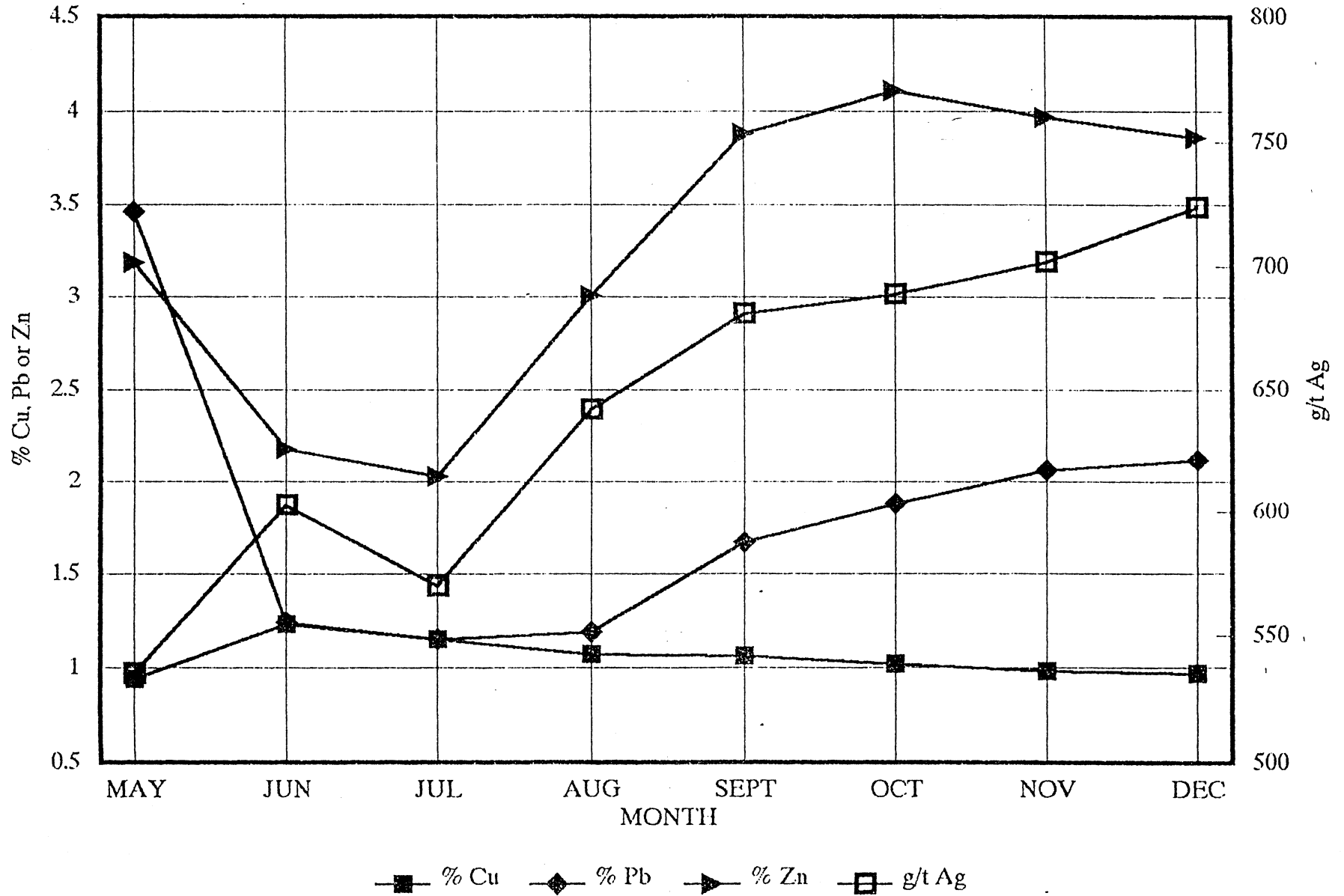


FIGURE 8

# Cu-Ag CONCENTRATE GRADE ASSAYS

YTD VALUES for 1989

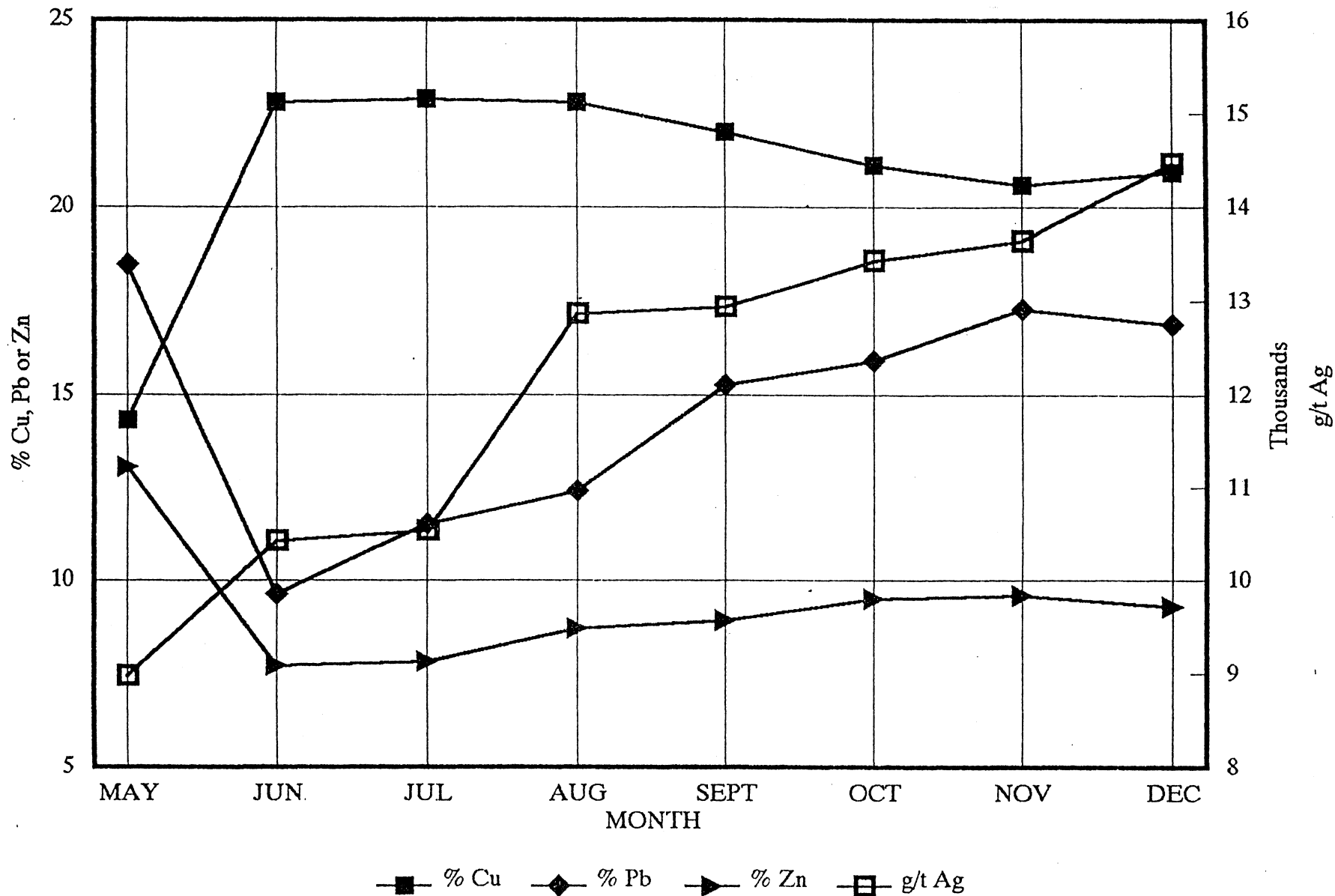


FIGURE 9

# Pb CONCENTRATE GRADE ASSAYS

YTD VALUES FOR 1989

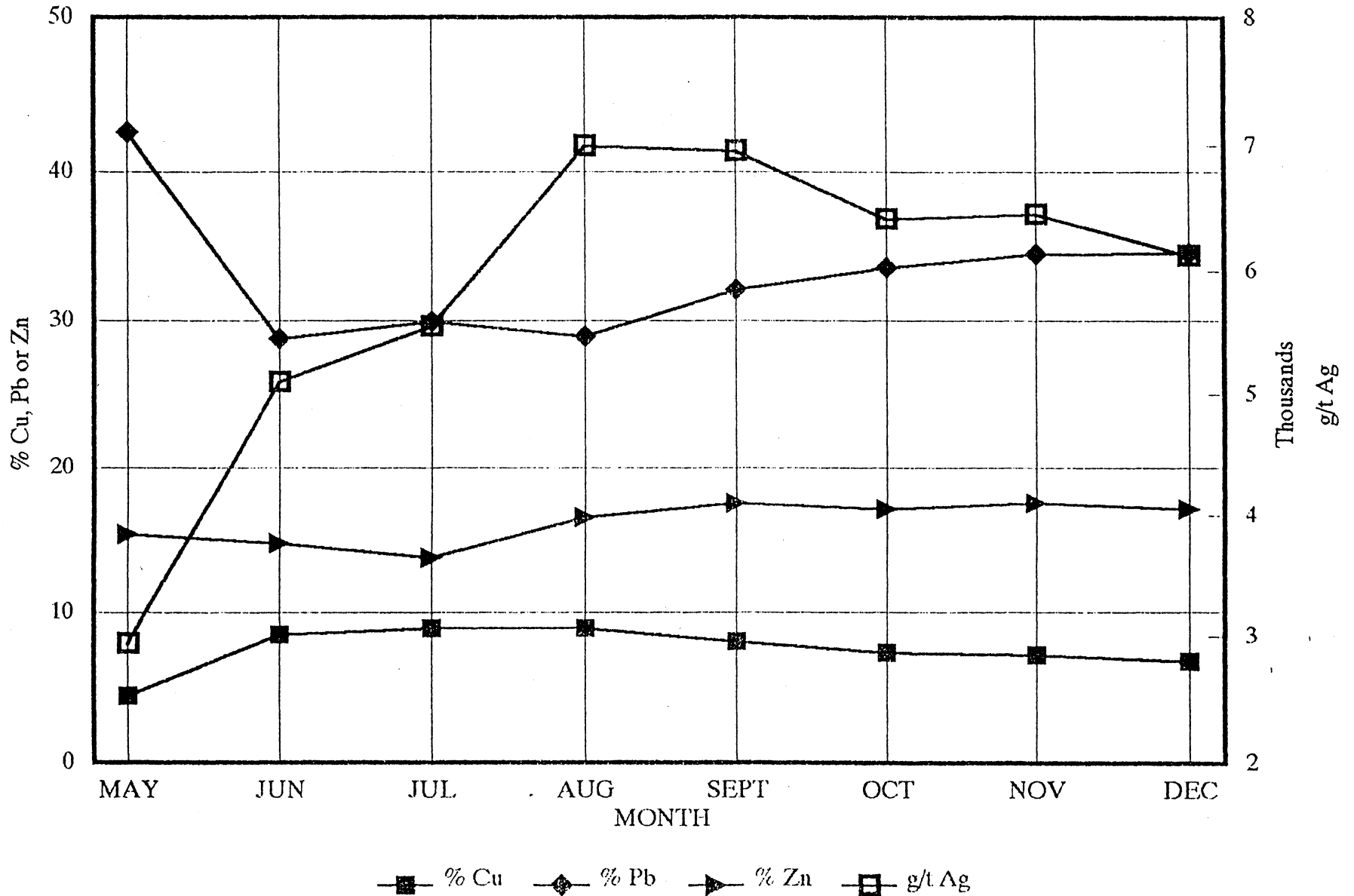


FIGURE 10

# Zn CONCENTRATE GRADE ASSAYS

YTD VALUES FOR 1989

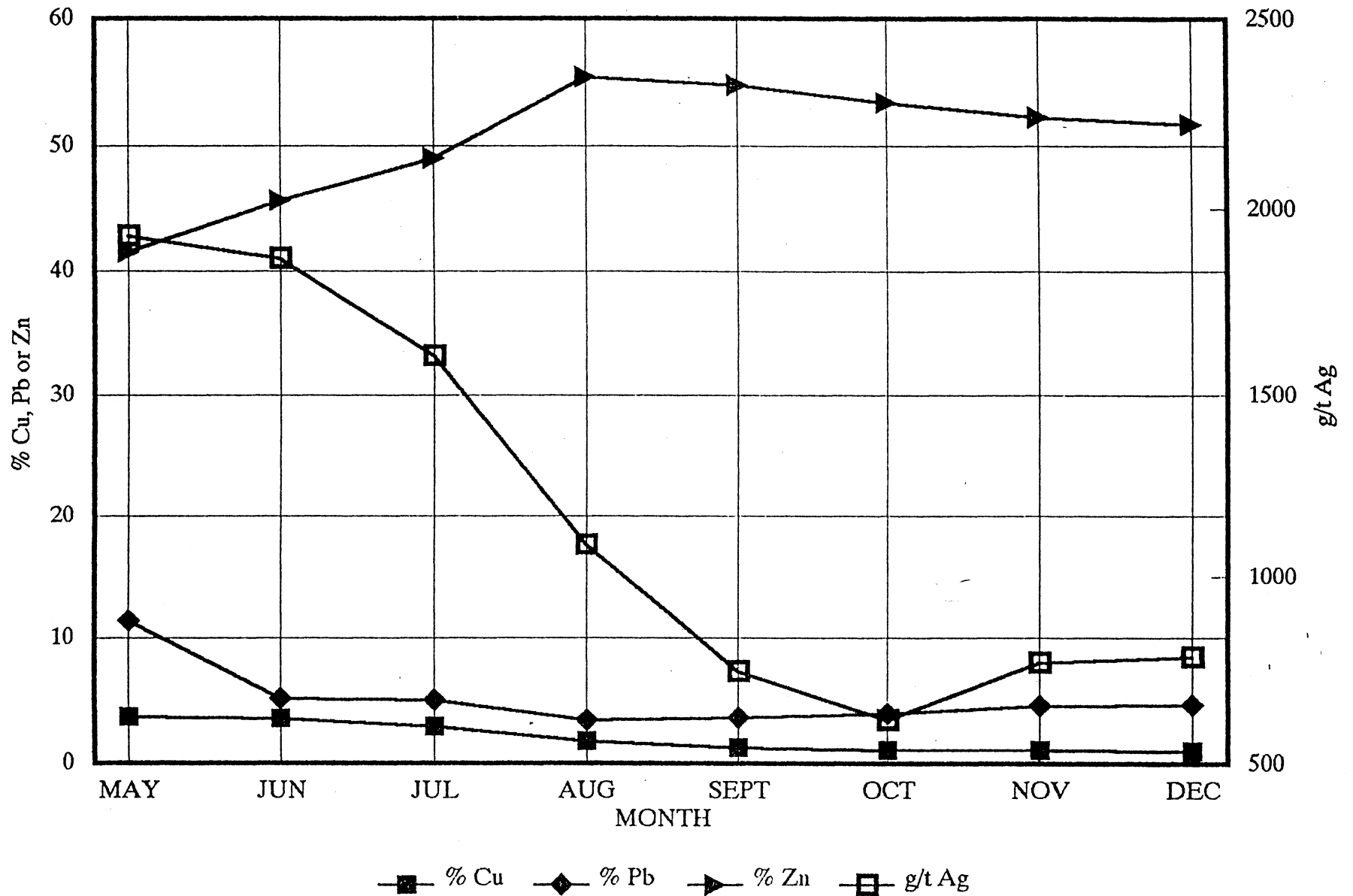


FIGURE 11

# % COPPER RECOVERY

YTD VALUES for 1989

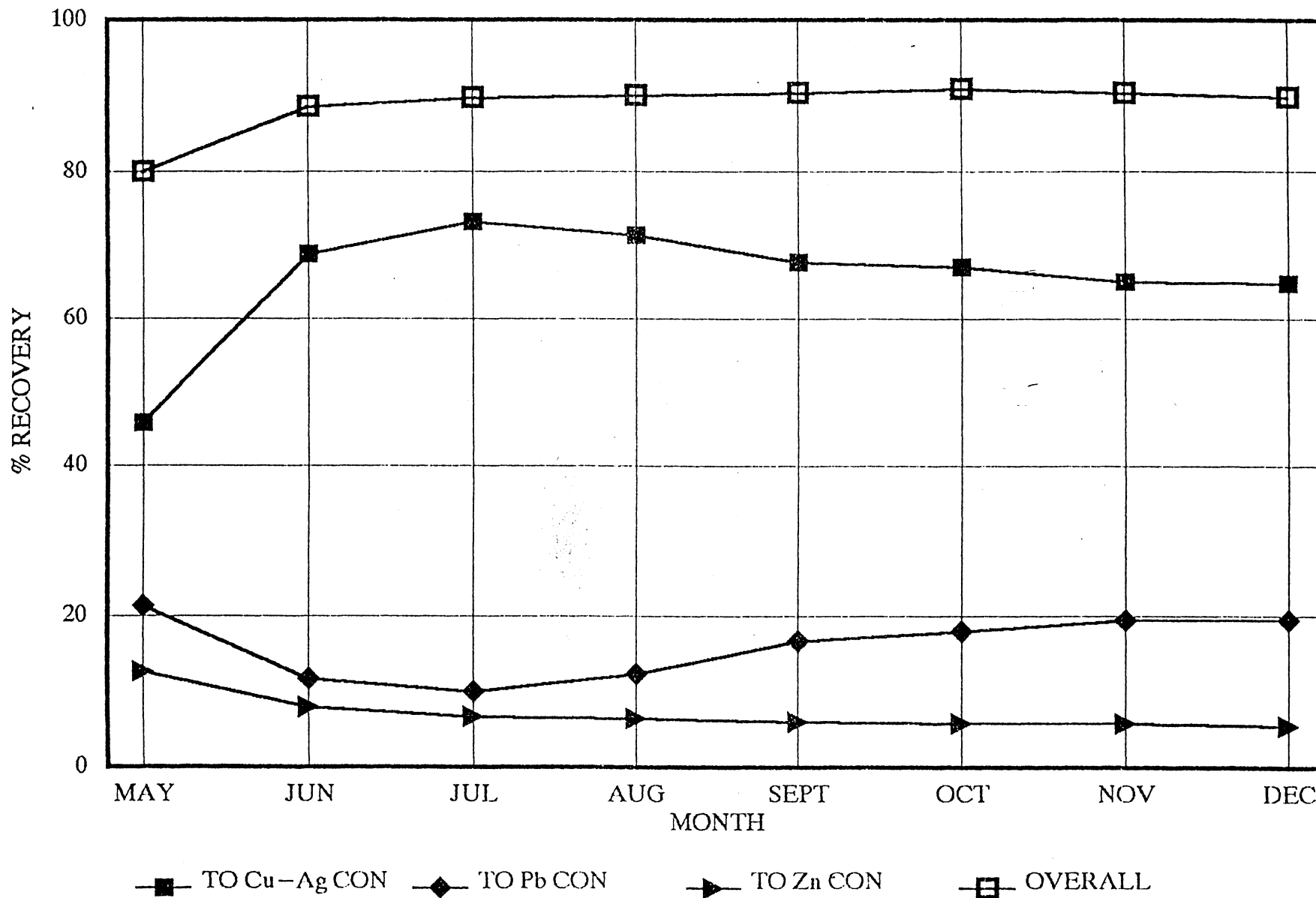


FIGURE 12

# % LEAD RECOVERY

YTD VALUES FOR 1989

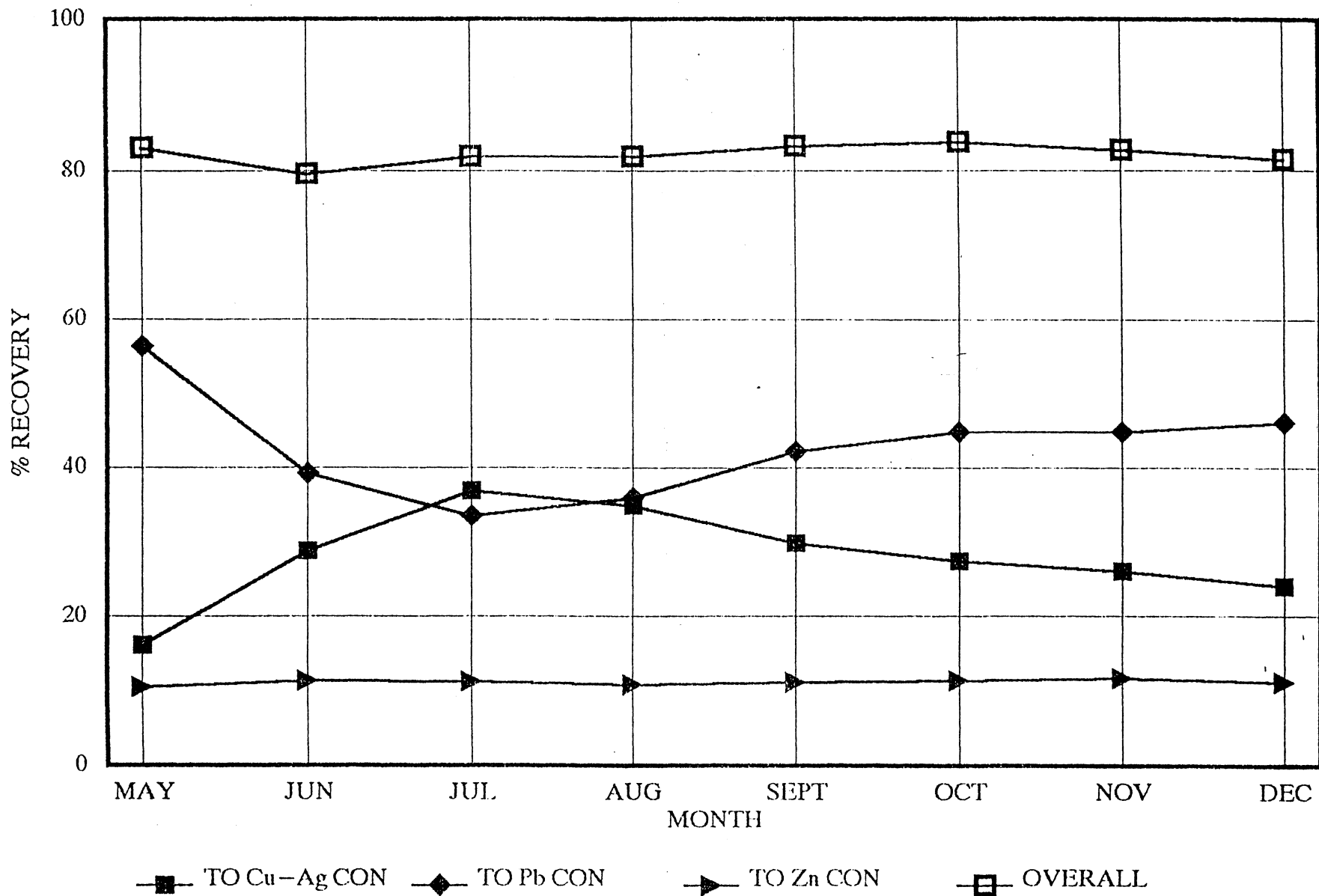


FIGURE 13

# % ZINC RECOVERY

YTD VALUES for 1989

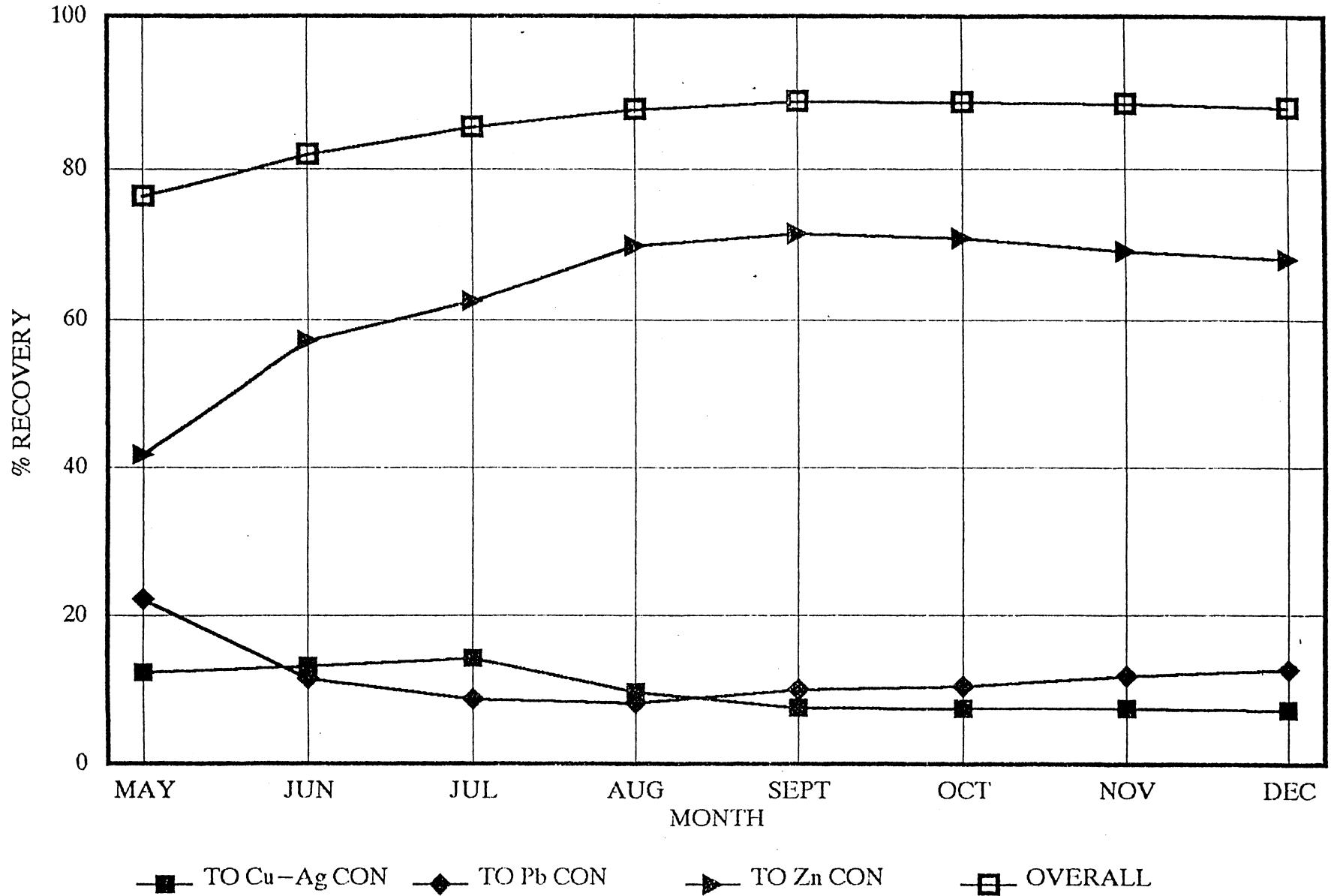




FIGURE 14

# % SILVER RECOVERY

YTD VALUES FOR 1989

