PRELIMINARY REPORT

PART 2

METALLURGICAL TECHNOLOGY

MEQB REGIONAL COPPER-NICKEL STUDY

PRELIMINARY REPORT
METALLURGICAL TECHNOLOGY
SMELTER-REFINERY MODEL

BY

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REGIONAL COPPER-NICKEL STUDY
MINNESOTA ENVIRONMENTAL QUALITY BOARD

TABLE OF CONTENTS

TABLE OF CONTENTS (contd.)

TABLE OF CONTENTS (contd.)

LIST OF FIGURES

FIGURES

FIGURES

FIGURES

FIGURES

FIGURES

FIGURES

FIGURES

LIST OF FIGURES

LIST OF FIGURES

LIST OF TABLES

TABLES

TABLES

EXECUTIVE SUMMARY

perhaps should say → 15 since our conc = 14.3

The smelting and refining models developed in this report have been designed to treat concentrates produced from mineral processing techniques, ranging in chemical analysis of 12 to 14 percent Cu and 2 to 3 percent Ni. For the purposes of the material balances found in this report, the concentrate analyzes 14.3 percent Cu and 2.14 percent Ni, and the feed as input to the smelter averages 14 percent Cu and 3 percent Ni. Despite the fact that no detailed engineering data is available on the smelting and refining of Minnesota Cu-Ni ores, it can be safely assumed that pyrometallurgical techniques (drying, smelting, converting, and refining) will be used. The following general and specific assumptions have been made prior to any mathematical calculations.

- A "material balance" is assumed to represent the quantitative flow of constituents through a process flowsheet. A "complete material balance" represents the quantitative flow of all the constituents of an input material through a process flowsheet. (This assumes that a complete and total chemical analysis is known with regards to the input material.) A "smelter/refinery material balance" represents the quantitative flow of constituents through a process flowsheet which involves the smelting and refining of said constituents. A "complete smelter/refinery material balance" represents the quantitative flow of all the constituents of an input material through a process flowsheet which involves the smelting and refining of said input material. (This assumes that a complete and total chemical analysis is known with regards to the input material.) A "fully" integrated smelter/refinery material balance" represents the quantitative flow of all input constituents through a process flowsheet which involves the smelting and refining of said input constituents to a final product (this assumes that no intermediate products exist in the balance which could be further treated, stored, or discarded).

Confusin

- All models developed from the various material balances contained in this report were without the aid of specific and detailed data relating to the treatment of Minnesota copper-nickel concentrates. Hence, all quantitative data have minimum acceptable accuracies of ±50 percent. In many cases the true accuracy of individual numbers will exceed ±100 percent.

Basis?

In some cases this may be true, however, for the majority of elements I believe the ± is smaller - Suggest you place accuracies on specific elements to show that those of major concern are known

But it is our best est. based on data . . .

- The flowsheet as outlined in Figure 1, which is the basis for the material balances found in this report, is itself, an assumption. Flowpaths have been hypothetically selected and are used as a tool in order to predict end-use quantities of metals, acid, concentrates, etc. The particular flowsheet chosen here may or may not be real, and hence, may or may not be indicative of treatment of Minnesota copper-nickel ores. But it is a reasonable approach

why →

- The material balances are based on the production of ≈ 100,000 MTY of high grade copper-nickel matte. The smelter/refinery model is based on the production of ≈ 100,000 MTY of metal (copper + nickel). This, of course, means that the quantities of any one particular constituent appearing in the material balances will have to be multiplied by a "multiplying factor" to yield the quantities as would be indicated by the smelter/refinery model.

- The output products as indicated in the smelter/refinery material balance are:

    metallics
    Fe-Cu-Ni concentrate
    cathode copper
    cathode nickel

Basis?

- To convert the output products to a final metal product basis (copper + nickel), it will be assumed that the metallic fraction and the Fe-Cu-Ni concentrate fraction (both of these fractions contain copper and nickel) will be treated for copper and nickel recovery with no losses of either element. Assume an overall copper recovery of 96.31 percent and an overall nickel recovery of 91.68 percent. These recoveries yield a Cu/Ni ratio of 7.00, which implies a "multiplying factor of" 1.37394 to be used in scaling up the numbers found in Table 1 to yield a smelter/refinery model production of 100,000 MTY of metal (copper + nickel). Refer to Figure 2 to determine the "multiplication factor" for other size operations. Note how the "mulciplication factor" will change as the copper and nickel recoveries change.

t

This not necessary for the Semin

- The multiplication factors to be used in the scaling of the quantities found in Table 1 are determined by the equation below:

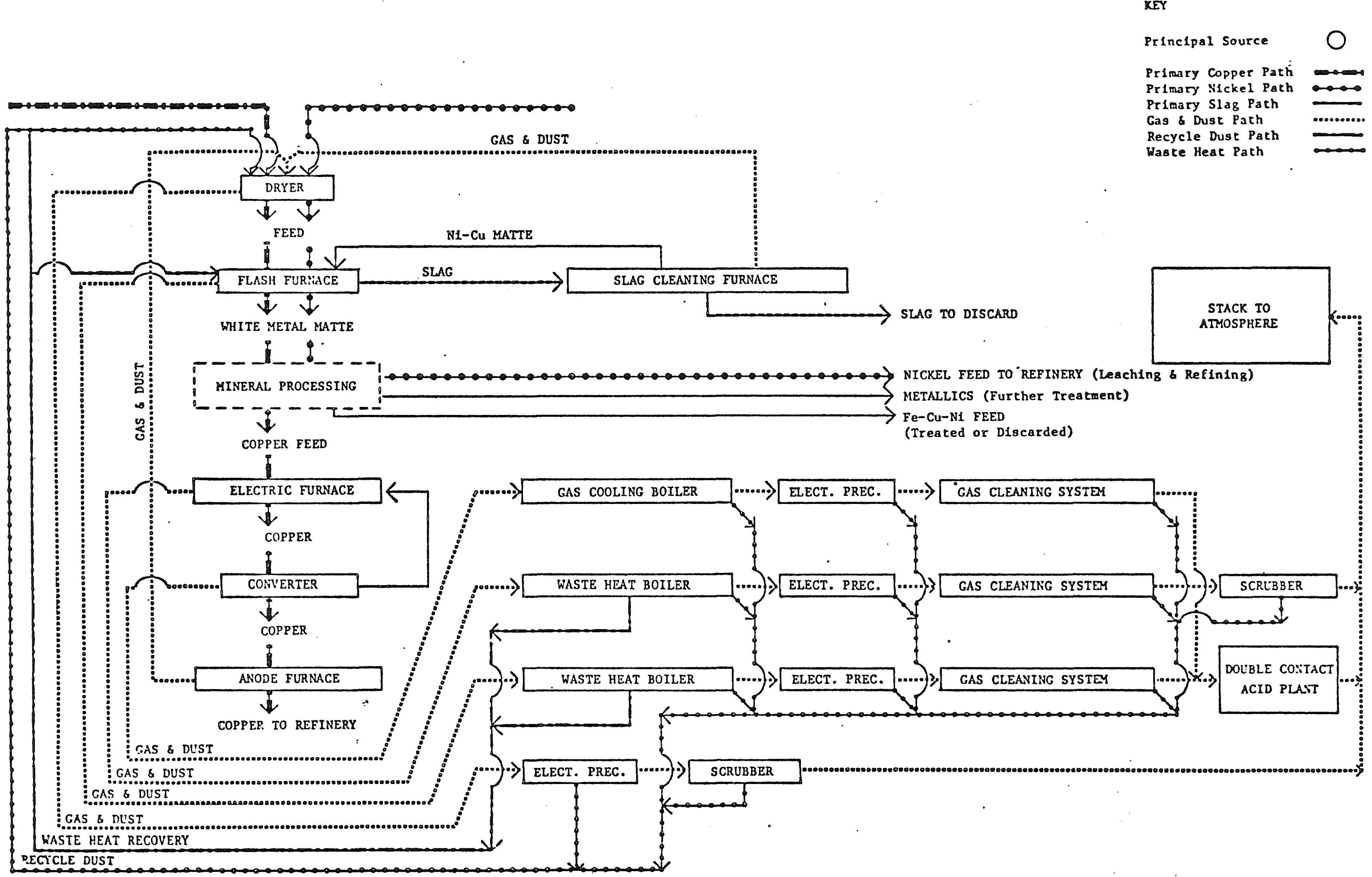
$$MF = \frac{A}{(x \cdot B + y \cdot C)}$$

where
MF = the multiplication factor
A = the desired size of operation (metric tons of metal (copper + nickel) per year)
X = the copper in the concentrate feed to the smelter (metric tons of copper per year)
B = the overall percent copper recovered
Y = the nickel in the concentrate feed to the smelter (metric tons of nickel per year)
C = the overall percent nickel recovered

N SR?

Fig 2 can be put in, but explanation not necessary

Figure 1. Generalized Flowsheet for Concentrate Smelting in the Production of Commercial Grade Copper and Nickel.



If put on a % basis then can apply to any size

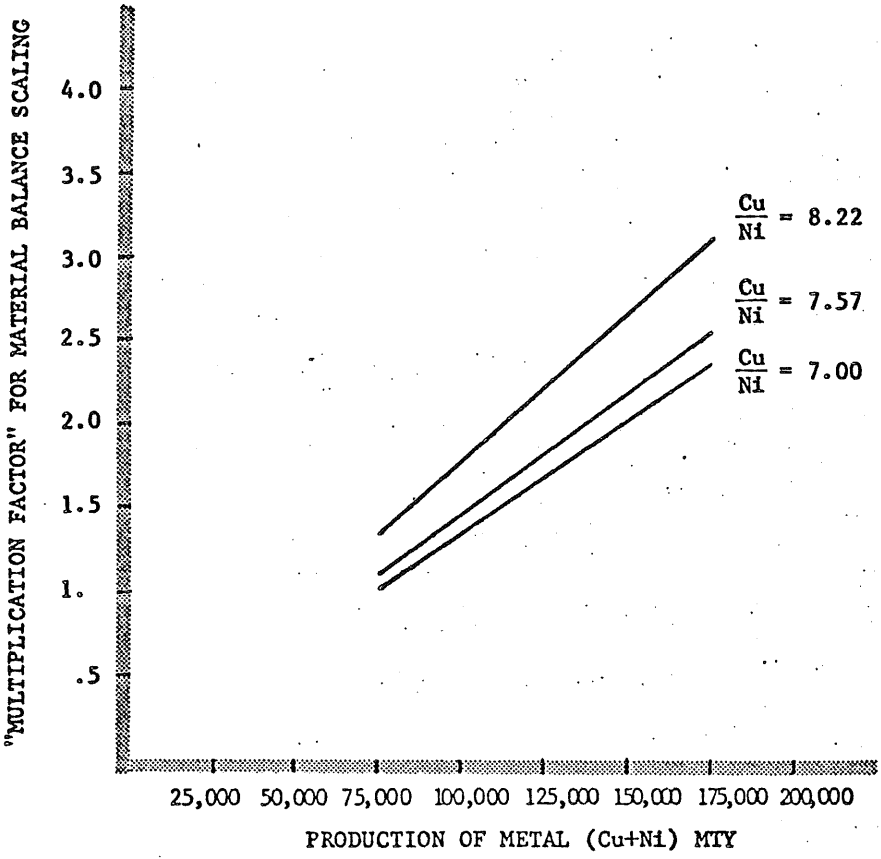
	% Cu	Ni	Fe	S	SiO$_2$	CaO	MgO	Al$_2$O$_3$	As %	Pb	Zn	Cd	Ag	Au	Rh	Pt	Pd
INPUT	100.00								100.00								
Concentrate	66,118	9,929	153,911	119,613	66,586	6,546	23,436	12,879	13.52	41.62	1,965	17.72	23.26	0.72	0.0088	0.89	2.1
Recyclables	13,667 20.67	7,163	11,369	4,094	69,714				2.84 21.01	0.40	37	3.29					
OUTPUT																	
Cu Cathodes	96.31 63,678	38	0.62	0.54					0.44 0.06	0.32			0.64	0.01			
Ni Cathodes	<0.01 0.27	9,065	0.09	0.09					0.05 0.37	0.05							
S for Conversion to acid				102,363													
Others (i.e. leach, residue, dust for use or sale, gas, etc.)	<0.01 0.73	0	3,139.29	7,711.37					80.99 10.95	29.19	899.8	15.47	21.715	0.69	0.0081	0.86	2.0
Discard Slag	1.83 1,207	604	148,337	1,730	126,203	6,488	23,221	12,766	7.18 0.97	11.32	1,021	0.42	0.7	0.01	0.0006	0.02	0.0
Fugitive Emissions	1.06								5.84								
Particulate	702	127	1,383	734	838	33	117	64	0.79	0.42	24.82	1.04	0.12	0.004	0.00006	0.005	0.0
Gas				1,087													
Stack Emissions	0.80								4.44								
Particulate	530	95.2	1,051	559	637	25	98	49	0.60	0.32	19.28	0.79	0.08	0.006	0.00004	0.005	0.0
Gas				5,425													
Discharge Water	<0.01								0.74								
Cu Refinery	0.045	0.08	0.02						0.1		0.1		0.005				
Recyclables	13,667 20.67	7,163	11,369	4,094	69,714				21.01 2.84	0.40	37	3.29					
Element Recovered	96.31	91.68	2.04	92.03	0	0	0	0	81.80	71.02	45.79	87.30	96.11	97.22	92.05	96.63	95.7
Element Lost	3.69	8.32	97.96	7.97	100	100	100	100	18.20	28.98	54.21	12.70	3.89	2.78	7.95	3.37	4.2

BASIS: ≈100,000 Metric Tons Per Year Production of White Metal Matte.

Table 1. Metallurgical Material Balance Summary (Smelter/Refinery Model (Quantities not Scaled) (Quantities are in Metric Tons Per Year).

What does this mean?

Figure 2. The Effects of the Change in the Cu/Ni Ratio on the "Multiplication Factors" for Material Balance Scaling (Yearly Metal Production).



Cu RECOVERY	Ni RECOVERY	Cu/Ni RATIO	MULTIPLICATION FACTOR (100,000 Metric Tons)
96.31	91.68	7.00	1.37394
89.87	79.07	7.57	1.48652
75.79	61.38	8.22	1.77921

?

- Material balances found in this report do not represent "complete material balances."
- Smelter/refinery material balances found in this report do not represent "complete smelter/refinery material balances."
- Smelter/refinery material balances found in this report do not represent "fully integrated smelter/refinery material balances."
- Oxygen is not considered in the material balances. It is not possible to predict oxygen or air pathways, efficiencies, or air dilution values. Therefore, it is assumed that sufficient oxygen as air and/or pure oxygen will be available to carry out the required reactions in the smelting and converting of copper-nickel ores. Total gas flow volumes are not known; however, estimates of gas flow, based on historical information, will be attempted.

3 supposed to be on site model

- All models are considered to be "off site" models with respect to all parameters and variables (i.e. capital and operating costs, manpower requirements, surface land use, water needs, etc.). The only exception to the above assumption will be with respect to transportation costs. Detailed transportation cost data was not generated at the time of the writing of this report. Smelters and/or refineries located "on site" will necessarily incur lower capital and operating costs, reduced manpower requirements, smaller surface land use, etc.

Basis

- Values predicting sulfur removal, fugitive emissions, stack emissions, metal recoveries, and so on, are assumed values.
- Specification sheets relating to quantity, size, and type of equipment used in the smelting and refining operations have not been included in the modeling. Flowsheets have been generalized to include only major types and quantities of equipment needs.

what does this mean?

- Capital costs, operating costs, energy requirements, water requirements, manpower needs, and so forth, are broken down by generalized categories. (These values will exclude any contributions resulting from any intermediate mineral processing which might occur. Mineral processing values will be interpolated from the main mineral processing model.)

SPECIFIC ASSUMPTIONS — Basis

Since you call this section "assumptions", why repeat it in every instance?

Drying

- The moisture content of the concentrate to the smelter is 15 percent.
- Ten percent by weight of the concentrate will become entrained in the dryer gas stream.

↑ 10% of Conc

Dry weight?

- Five percent of the total dust generated will be considered to be fugitive.

- Ninety-six percent of the dust generated (excluding fugitive dust) is treated and removed and returned to the system.

seems all is treated but 96% removed

Smelting

- The input feed concentrate has the following chemical composition:

Element	Wt %	Element	Wt %
Cu	14.3	Pb	≈ 0.009
Ni	2.15	Zn	≈ 0.42
Fe	33.2	Cd	≈ 0.0038
S	25.8	Ag	≈ 0.005
SiO_2	14.3	Au	≈ 0.00015
CaO	1.4	Rh	≈ 0.0000019
MgO	5.0	Pt	≈ 0.00019
Al_2O_3	2.8	Pd	≈ 0.00045
As	≈ 0.0029		

- Converter slag and portions of captured flue dust are recycled and added as feed to smelter.

- Exhaust gases contain some molten or semi-molten dust particles. Volatile compounds tend to be concentrated in the dust. Gases plus dust are assumed to be a homogeneous mixture at the exit port of each emission location.

- Individual dust particles emitted at the various point source locations within a smelting complex assume the same chemical composition as the material from which it is derived. Dusts generated from the fluid bed reactors assume the same chemical analysis as the product calcine, dusts generated from the smelting furnace assume the same chemical analysis as the metal matte, and so on.

- Zinc: major portion of this constituent ~~is oxid~~ized and distributed into the slag. A portion is partly volatilized. meaning √

  85% goes into the slag (true for concentrates high in zinc)
  15% removed as dust (volatilization rate is ≈20-30%)

- Lead: major portion goes into the matte. PbS in the matte is then oxidized partly to PbO in the converting furnace and distributed into the slag.

  90% goes into the matte (true for concentrates high in lead)
  10% removed as dust (volatilization rate is ≈60%)

- CaO, MgO, Al_2O_3: these constituents are principally distributed into the slag.

- Arsenic: 90 percent of the total amount present is removed as dust or particulates (volatilization rate is ≈10%).

- Cadmium: 90 percent of the total amount present is removed as dust or particulates (volatilization rate is ≈10%).

- Cobalt: approximately 70 percent of the total amount present goes into the matte. Thirty percent is removed as dust or particulates (volatilization rate is ≈10%).

- It is assumed that some flue dusts ~~is~~ are more easily collected then others. The ease of collection is dependent upon where dust accumulates in the system. Also, it is assumed that some dusts or particulates are specially treated for removal of certain constituents. It is assumed that ≈73 percent of the As, Pb, Zn, and Cd is either:

basis

  1) contained or isolated in the system where collection occurs less frequently, or

  2) removed by special treatment prior to recirculation.

- Five percent of the total dust generated in the smelting system(s) is considered to be fugitive, and as such, is never treated for removal.

- Ninety-six percent of the dust generated (excluding fugitive dust) is treated and recovered and returned for use in the smelter. The remaining four percent is discharged as a stack emission.

- One percent of the total sulfur generated as SO_2 is considered to be fugitive, and as such, is never treated for removal.

- Ninety-five percent of the sulfur generated as SO_2 (excluding fugitive sulfur emissions as SO_2) is treated for removal as sulfuric acid and sold as a byproduct. The remaining five percent is discharged as a stack emission.

- Ninety-six percent (96%) removal efficiency on particulate cleaners is assumed. Deviations from this efficiency will dramatically change particulate emissions.

- Ninety-five percent (95%) removal efficiency of SO_2 in the sulfuric acid plant is assumed. Deviations from this efficiency will dramatically change gas emissions.

- Assume Cu, Ni, Fe, S in the matte are as Cu_2S, Ni_3S_2, Fe_7S_8, and Fe. Assume in flotation 95 percent recovery of Cu_2S in a Cu-concentrate, 85 percent recovery of Ni_3S_2 in a Ni-concentrate, and 85 percent recovery of Fe_7S_8 in a Fe-concentrate. The remainder of each constituent is split equally among the three concentrates. The analysis of the three concentrates is determined after summation. Assume 12,000 metric tons per year of metallics will form from every 100,000 metric tons of white metal matte averaging 63 to 68 percent Cu. The analysis of the metallics is determined by difference ($\%\ \text{Element}_{\text{metallics}} = \%\ \text{Element}_{\text{white metal}} - \%\ \text{Element}_{\text{Cu conc.}} - \%\ \text{Element}_{\text{Ni conc.}} - \%\ \text{Element}_{\text{Fe conc.}}$) (Figure 3).

what does this have to do with Fig 3

Precious Metals

The metallic phase in sulfur deficient, very slowly cooled matte, over a wide composition range including commercial mattes, contains by direct analysis 80 percent nickel or somewhat less, and about 20 percent copper. The amount of metallics present depends, of course, on the sulfur deficiency of the matte as controlled by the converter finishing procedure. The ratio of concentration of the platinum metals and gold in the metallic grains of the slowly cooled matte, is very roughly 100 to 1. Silver, on the other hand, is strongly concentrated in the copper sulfide, not surprisingly in view of the greater affinity of sulfur for silver than for platinum metals and gold, and the isomorphism of Ag_2S with Cu_2S.

something missing?

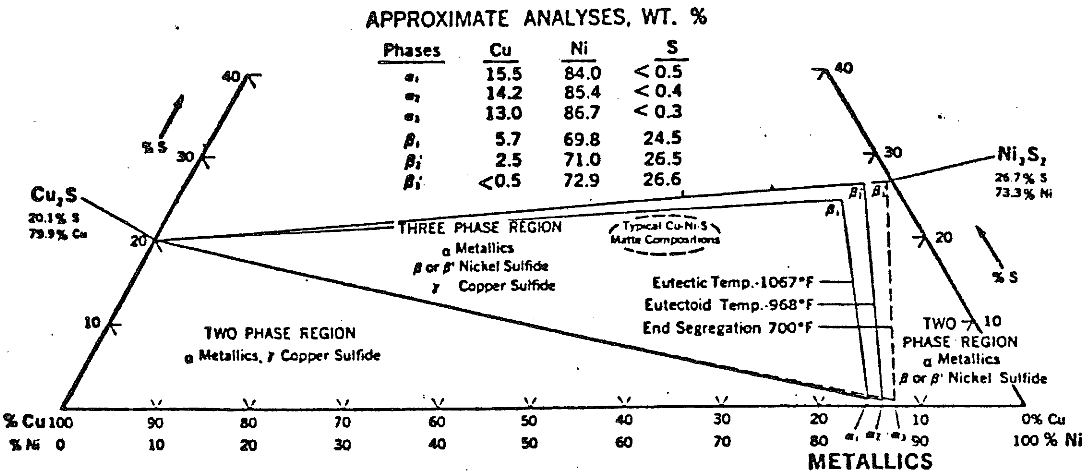
I am Confused !!!

Refining

- Average values for waste effluents from the copper refinery are taken from Table 2. (50,000 metric tons cathode copper/year production is assumed.)
- The volume of copper sulfate production converted to metallic copper amounts to 1.5-1.8 percent of the amount of copper produced. (1.6 percent of the amount of copper produced is assumed for the copper refinery balance.) Source: the electrolytic refining of copper, V.T. Isakov, 1973. — page # or whole Book
- The approximate distribution of the principal elements contained in the anodes between the cathodes, electrolyte, and slime is given below in percent.

where? P12

Figure 3. Cu-Ni Equilibrium Phase Diagram. (Source: Boldt, 1967)



APPROXIMATE ANALYSES, WT. %

Phases	Cu	Ni	S
α_1	15.5	84.0	< 0.5
α_2	14.2	85.4	< 0.4
α_3	13.0	86.7	< 0.3
β_1	5.7	69.8	24.5
β_1'	2.5	71.0	26.5
β_1'	<0.5	72.9	26.6

This equilibrium phase diagram is a concise graphical representation of the amounts and compositions of the solid phases that crystallize from a nickel-copper-sulfur melt held at a particular temperature until all change ceases. All mattes with compositions that fall within the boundaries of the center triangles will contain three separate distinct phases, the compositions of which may be read from the corners of the appropriate triangle. At the ternary eutectic temperature of 1067°F the three solid phases co-existing are: copper sulfide of 20 per cent sulfur and less than 0.5 per cent nickel; metallic phase (α) of about 15.5 per cent copper, less than 0.5 per cent sulfur, and about 84 per cent nickel; and nickel sulfide (β) of about 6 per cent copper. At the eutectoid temperature of 968°F the copper sulfide and metallic phases have changed little, but the nickel sulfide undergoes a transformation to a low temperature form (β') in which the solubility of copper is only about 2.5 per cent. At 700°F, β' contains less than 0.5 per cent copper.

7

10

??

Table 2. Waste Effluents from Electrolytic Copper Plants.

(50,000 MTPY Cu Plant Size)

PARAMETER	PLANT NUMBER 1 NET LOADING kg/kkg	2 NET LOADING kg/kkg	3 NET LOADING kg/kkg	4 NET LOADING kg/kkg	5 NET LOADING kg/kkg	6 NET LOADING kg/kkg	AVERAGE VALUES kg/kkg	metric tons/year
pH								
Alkalinity		0.044	0.04		0.017			
COD	0.006	0.472	0.024	0.119	0.305			
Total Solids	0.074	3.40	0.014	0.310				
Dissolved Solids	0.068	1.63	0.006	0.173	0.94			
Suspended Solids	0.006	1.00	0.008	0.137	0.10			
Oil and Grease					0.05	0.003		
Sulfate (as S)	0.011	0.066	0.002	0.080			0.03975	2.0
Chloride		1.14	0.003	0.037	0.22			
Cyanide								
Aluminum		0.0004			0.010	0.004		
Arsenic		0.0001		0.003			0.002	0.1
Cadmium		<0.0001						
Calcium	0.040	0.28	0.001		0.06			
Chromium					<0.0001	0.0002		
Copper	0.001	0.0005	0.0001		0.002	0.43	0.0009	0.045
Iron	0.0008	0.0003	<0.0001		0.0006		0.00045	0.0225
d	0.0002		<0.0001		<0.0001	0.005	0.00135	0.0675
Magnesium		0.115						
Mercury					0.0001	<0.0001		
Molybdenum		4.72						
Nickel	0.0004	0.0001	0.0001	0.006	0.001		0.00152	0.076
Potassium		0.013						
Selenium	0.0002	<0.0001	0.0002	0.0004		0.042		
Silver		<0.0001				0.0001	0.0001	0.005
Sodium		0.91	0.0001			0.052		
Tellurium			0.001			0.052		
Zinc	0.0002		<0.0001	0.003	<0.0001	0.008	0.00228	0.114

%?

	Cathodes	Solution	Slime
Gold	1.0-1.5	-	98.5-99.0
Silver	2.0-3.0	-	97.0-98.0
Copper	98.0	1.9	0.07
Selenium and tellurium	1.0-2.0	-	98.0-99.0
Lead	1.0-5.0	-	95.0-99.0
Nickel	15.0	75.0	10.0
Antimony and tin	20.0-30.0	20.0	50.0-60.0
Sulphur	3.0-5.0	-	95.0-97.0
Iron	10.0-20.0	80.0	10.0-20.0
Zinc	3.0	93.0	4.0
Aluminium	5.0	75.0	20.0
Silicon	-	-	100
Bismuth	-	21.8	78.2
Arsenic	20.0	60.0	20.0

SOURCE: The Electrolytic Refining of Copper, V.T. Isakov, 1973.

- Assume 95 percent recovery of precious metals.
- Number of starter cells is determined by the following formula:

$$H = Nn_1Mn_3/(2an_2\ K + n_1Mn_3) = 19$$

where N = number of cells in plant (328)
n_1 = number of cathodes in production cell (45)
n_2 = number of blanks in cell (28)
K = yield of usable sheets (95%)
n_3 = number of sheets used for the production of one starting sheet (1.08)
a = average period for growth of cathodes in all the production cells (14)
M = machine time of production cells (93%)
H = number of starter cells

SOURCE: The Electrolytic Refining of Copper, V.T. Isakov, 1973.

what is this here for?

- Weight of anode is 0.35 metric tons.
- Weight of cathode is 0.125 metric tons.
- Number of days per refinery campaign is 28 days.
- Number of anodes per cell is 44.
- Number of cathodes per cell is 45.
- Number of cathodes per campaign is 90.

- Number of cells is 331.
- Number of cells per section is 24.
- Number of sections is 14.

SOURCES: The Electrolytic Refining of Copper, V.T. Isakov, 1973.
Extractive Metallurgy of Copper, A.K. Biswas, 1976.

Why here?

- ok Assume an overall nickel recovery if 98 percent.
- ok Nickel recovery as a nickel cathode is assumed to be 85 percent.
- Weight of nickel cathode is 0.075 metric tons.
- Number of days per refinery campaign is 8 days.
- Number of campaigns per year is 46.
- Number of cells per campaign is 45.
- Number of cathodes per cell is 39.
- Number of anodes per cell is 40.

The pyrometallurgical material balances which summarize "semi-quantitatively" the flow of elements through a system will necessarily have minimum acceptable accuracies of ±50 percent, and in many cases accuracies exceeding ±100 percent. This factor, however, will not detract from the probable values to be anticipated and/or expected in the models (i.e. the order of magnitude for predictable values in most cases is assumed to be correct).

Too high for most cases we need to know about

100% is 2 orders of magnitude -

The smelting/refining model incorporates concentrate drying followed by flash smelting to a bulk copper-nickel matte. Metal separation will be by flotation of the ground copper-nickel matte with the nickel-copper product being refined by hydrometallurgical and electrowinning techniques and the copper-nickel product being pyrometallurgically treated and electrolytically refined.

Figure 4 illustrates, in generalized form, the smelting flowsheet for the processing of copper-nickel concentrates. The flowsheet establishes the principal equipment needs, and indicates the major or primary paths for copper, nickel, slag, gas, dust, and waste heat. Figures 5 through 12 indicate the principal sources of:

Show on flowsheet + list - + recycle Dust
- Gas Em
- Part
- etc

- recycled dust (i.e. the electrostatic precipitators)
- atmospheric gas emissions (i.e. the stack)
- atmospheric particulate emissions (i.e. the stack)
- fugitive gas emissions (i.e. flash furnace, converters, anode furnace)
- fugitive particulate emissions (i.e. flash furnace, converters, anode furnace)
- water discharges (i.e. slag cleaning furnace, mineral processing, anode furnace, double contact acid plant)
- solid wastes (i.e. slag cleaning furnace, mineral processing)

Figure 13 illustrates in generalized form the flowsheet for the refining of anode Cu and Ni-Cu concentrate (intermediate products which are generated in the smelting of Cu-Ni concentrates). The flowsheet establishes the principal equipment needs, and indicates the major or primary paths for copper and nickel. Figure 14 identifies the areas where input water is required, and Figure 15 summarizes the principal sources for water discharges (i.e. anode rinsing, cathode rinsing, product casting, electrolytic bleed, and barometric condenser).

areas

Table 3 collectively summarizes the inputs and outputs for the proposed smelter/refinery model (quantitative information for this table came directly from the material balances found later in this report with no scaling of the numbers). Certain output products identified in Table 3

meaning 12 a 100,000 [illegible] te metal intake?

Figure 4. Generalized Flowsheet for Concentrate Smelting in the Production of Commercial Grade Copper and Nickel.

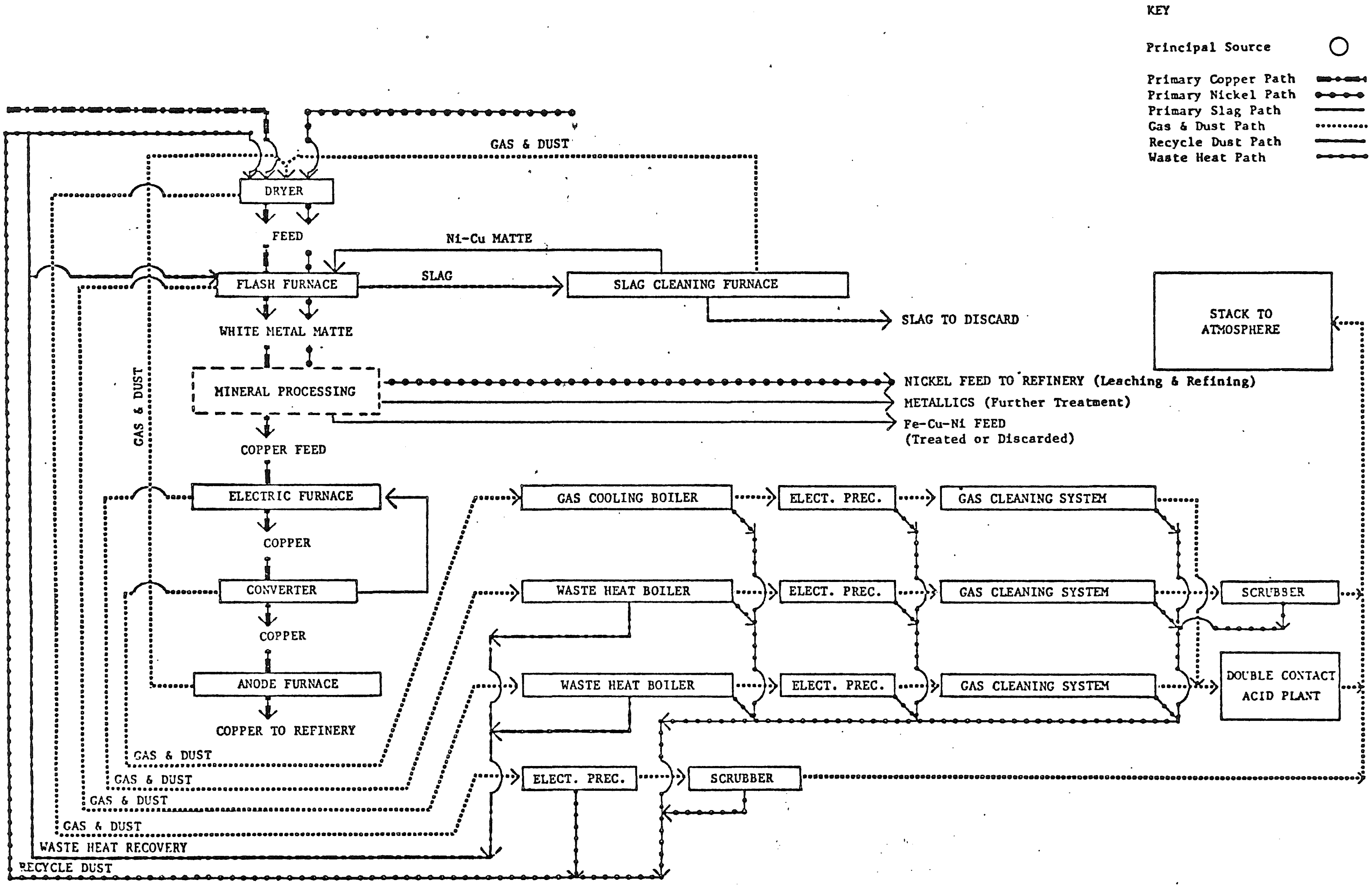


Figure 5. Generalized Flowsheet for Concentrate Smelting in the Production of Commercial Grade Copper and Nickel (Principal Sources for Recycle Dust).

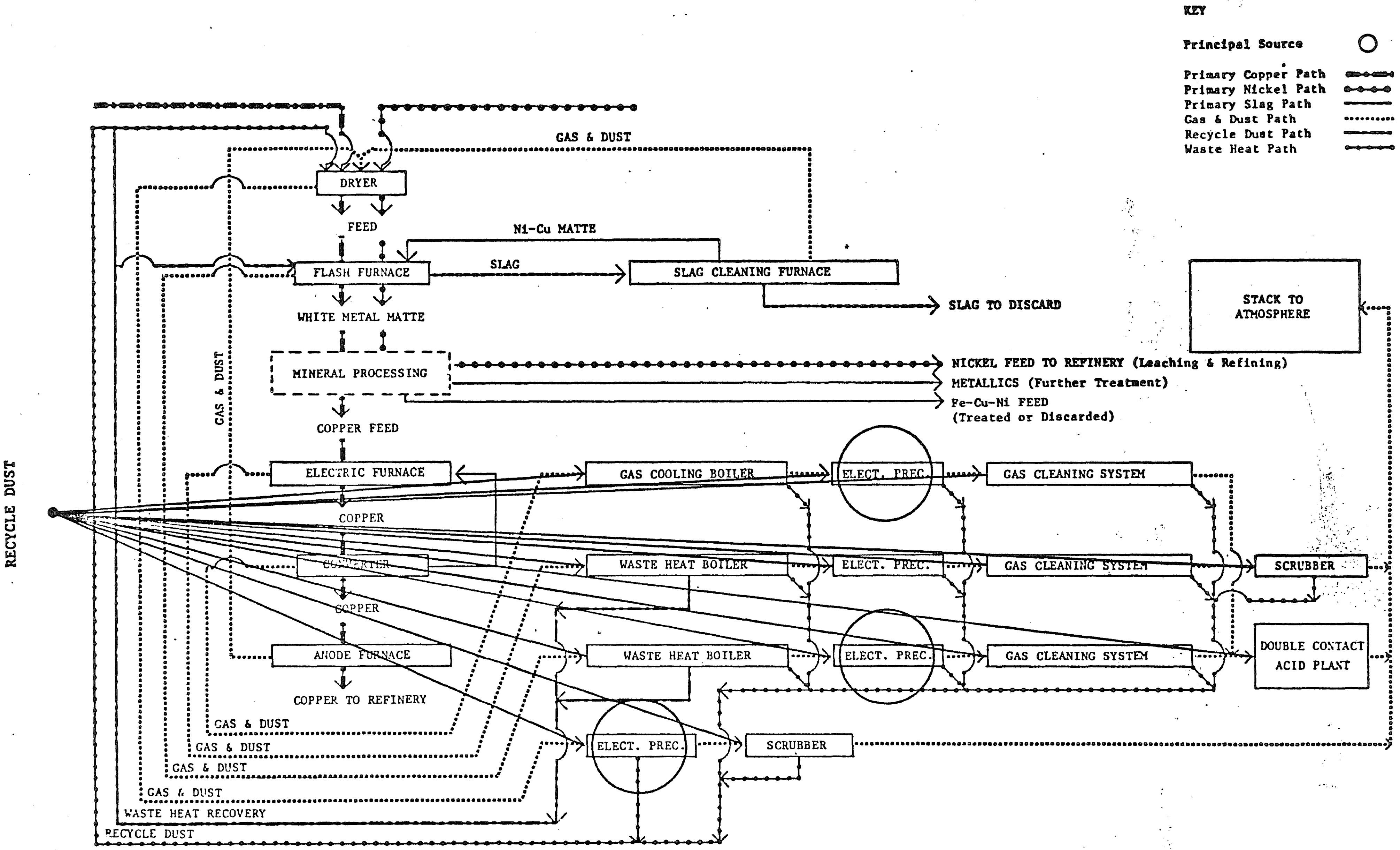


Figure 6. Generalized Flosheet for Concentrate Smelting in the Production of Commercial Grade Copper and Nickel (Principal Sources for Stack Gas Emissions).

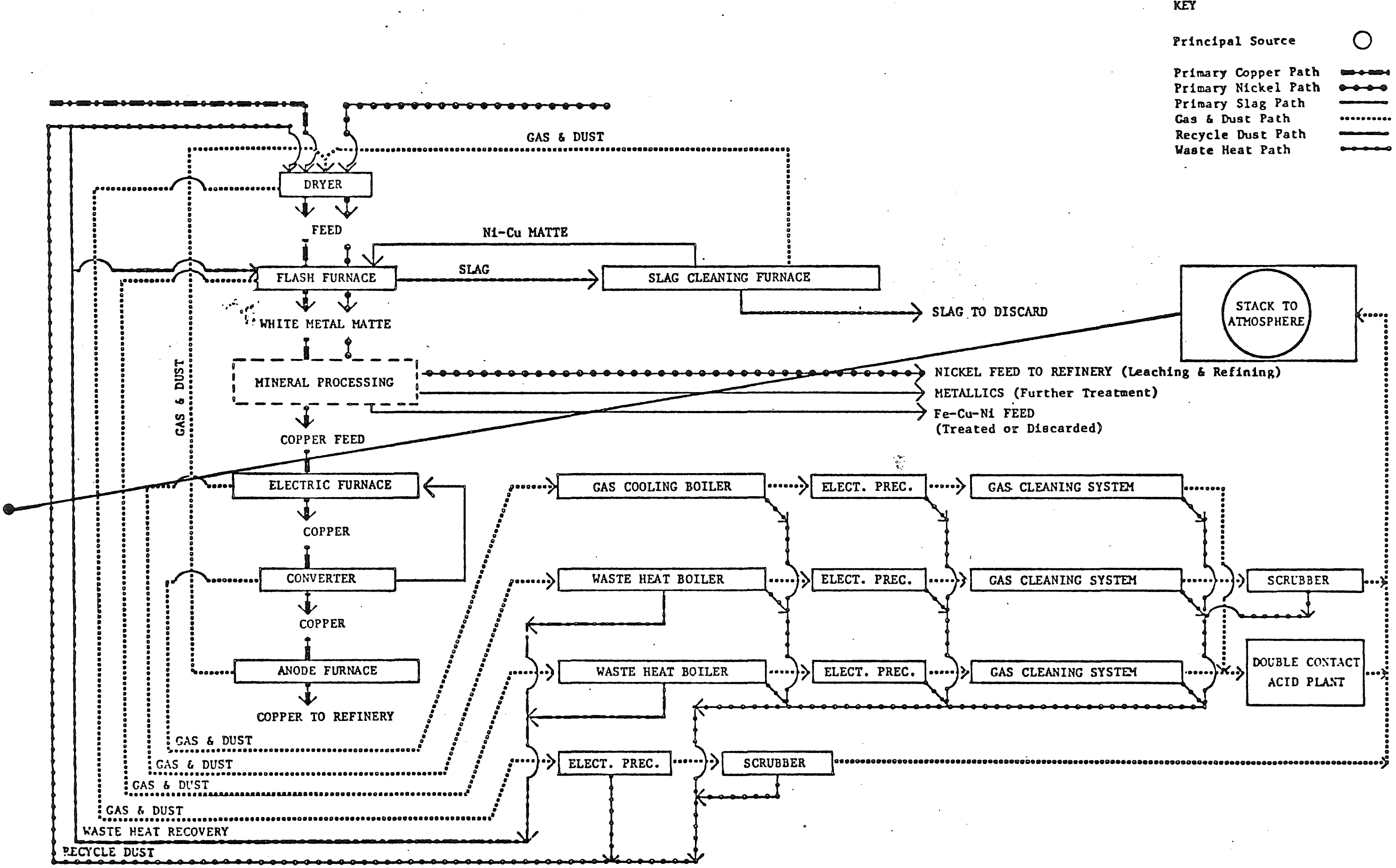


Figure 7. Generalized Flowsheet for Concentrate Smelting in the Production of Commercial Grade Copper and Nickel (Principal Sources for Stack Particulate Emissions).

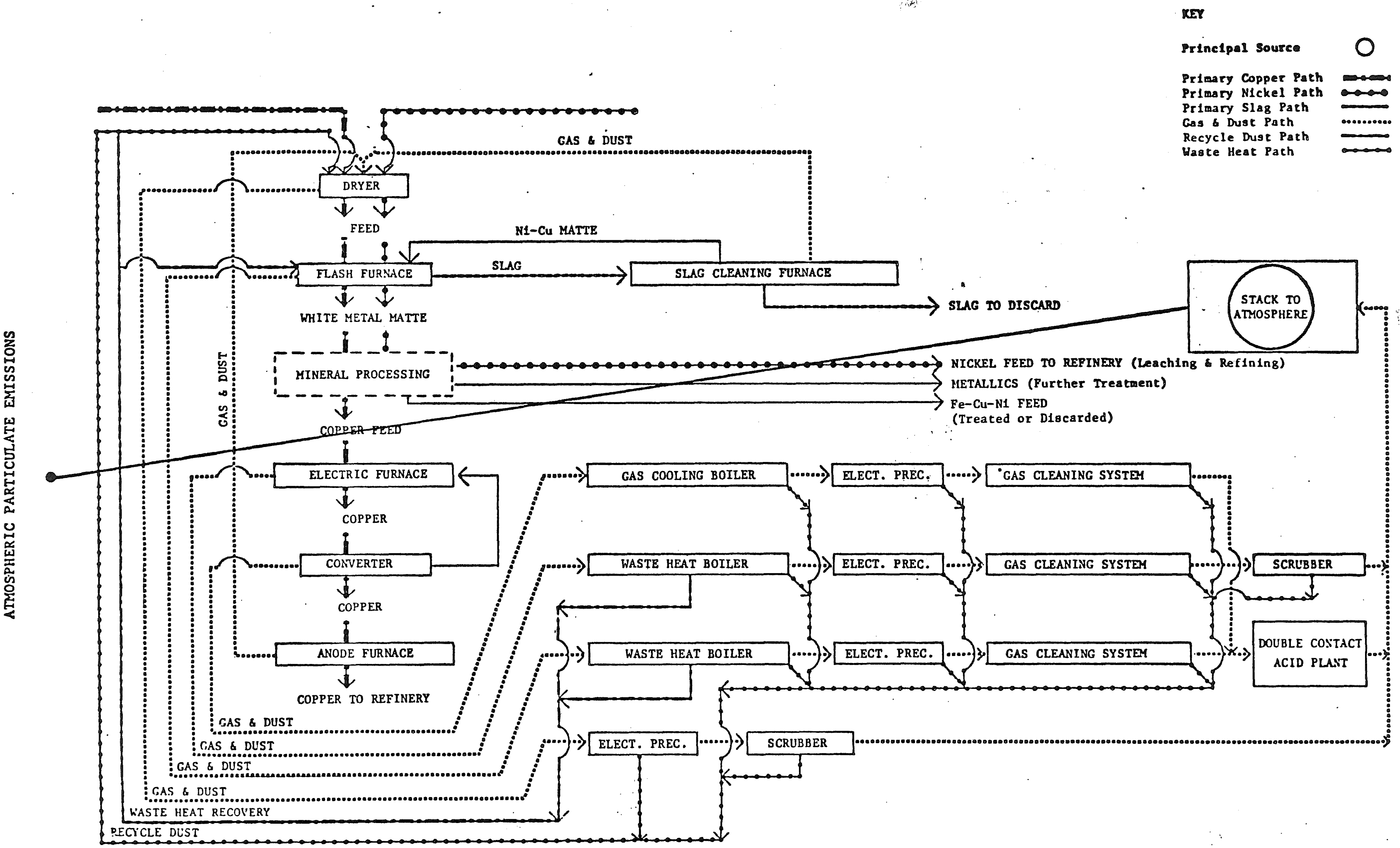


Figure 8. Generalized Flowsheet for Concentrate Smelting in the Production of Commercial Grade Copper and Nickel (Principal Sources for Fugitive Gas Emissions).

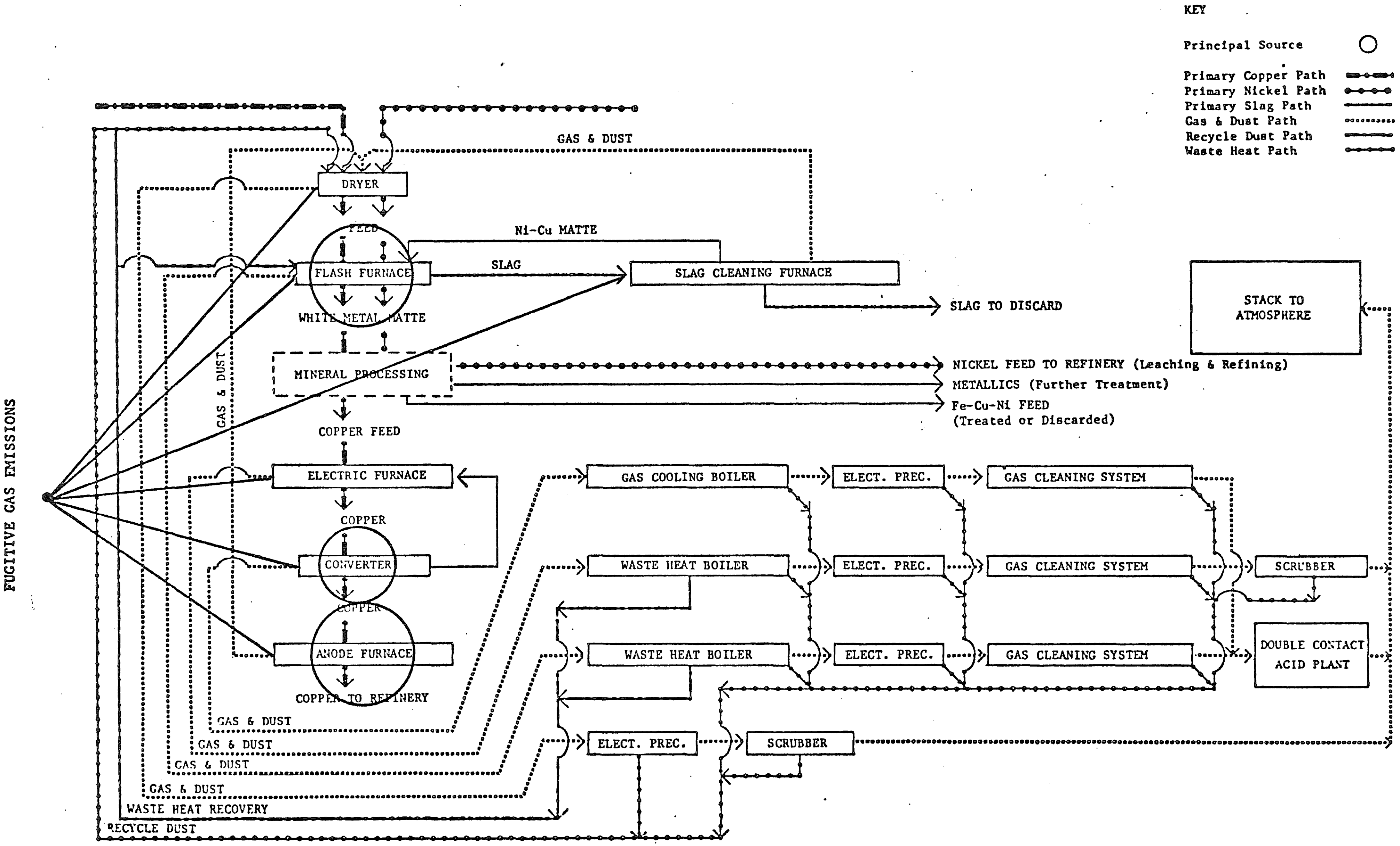


Figure 9. Generalized Flowsheet for Concentrate Smelting in the Production of Commercial Grade Copper and Nickel (Principal Sources for Fugitive Particulate Emissions).

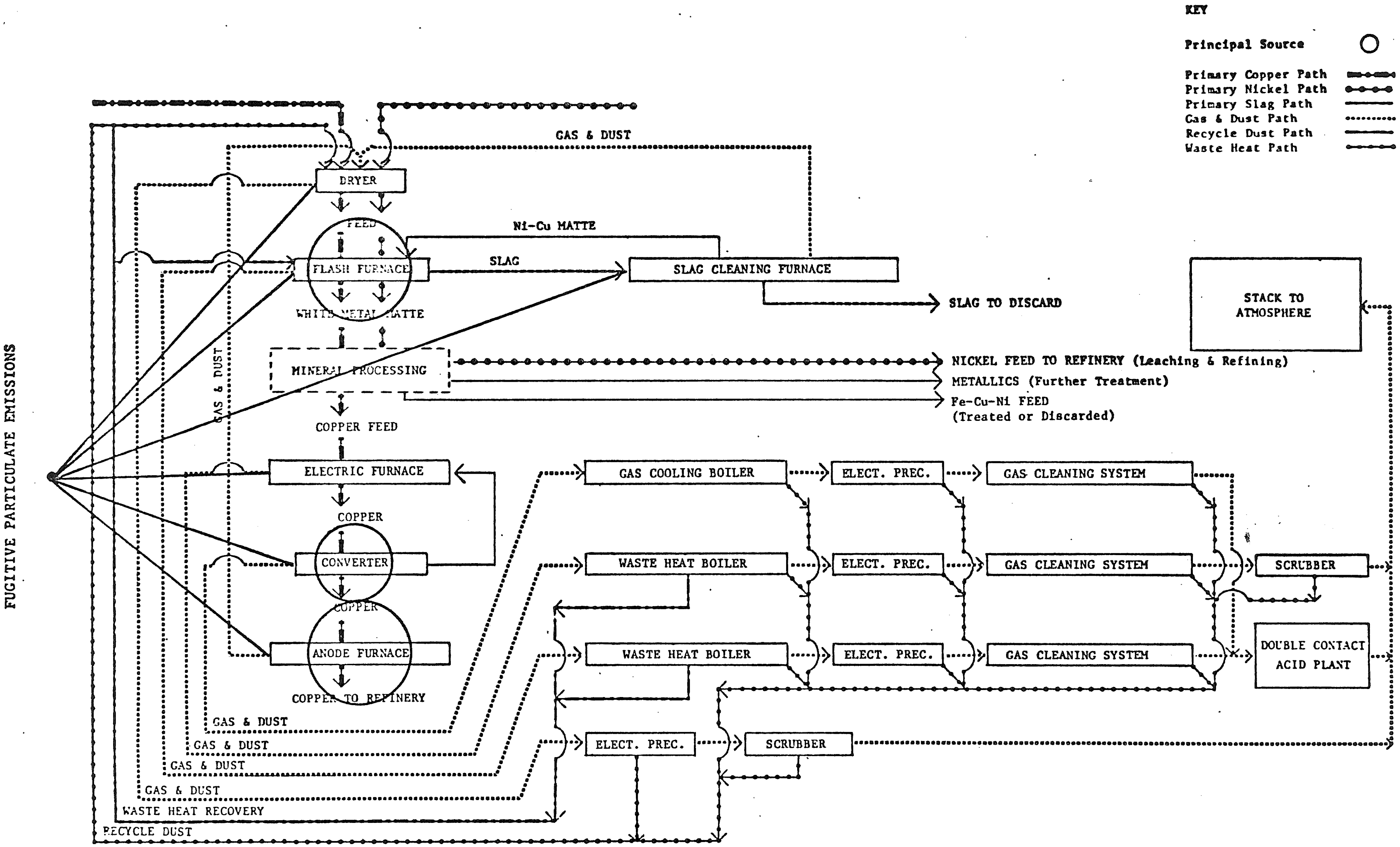


Figure 10. Generalized Flowsheet for Concentrate Smelting in the Production of Commercial Grade Copper and Nickel (Sources for Water Needs).

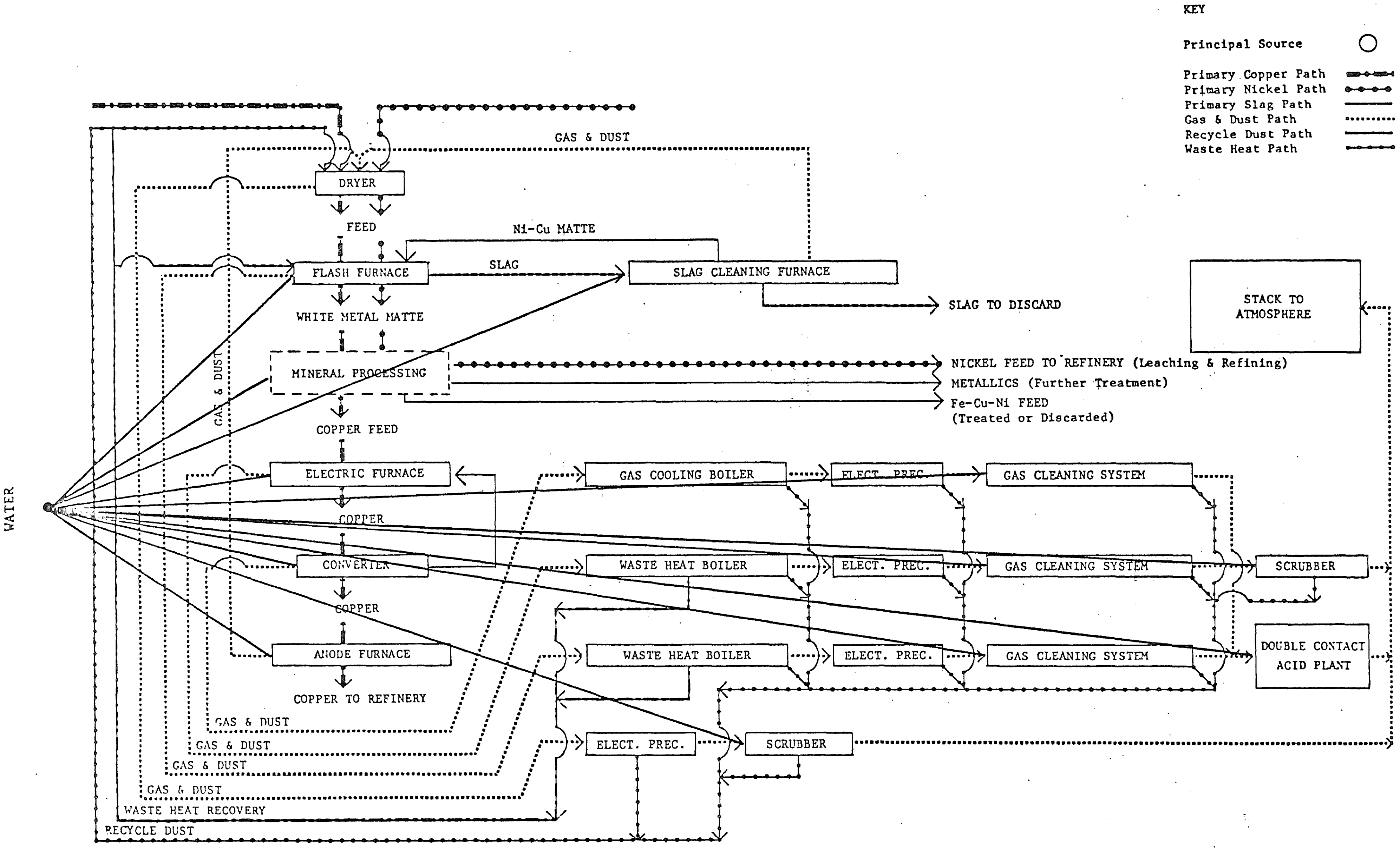


Figure 11. Generalized Flowsheet for Concentrate Smelting in the Production of Commercial Grade Copper and Nickel (Principal Sources for Water Discharges).

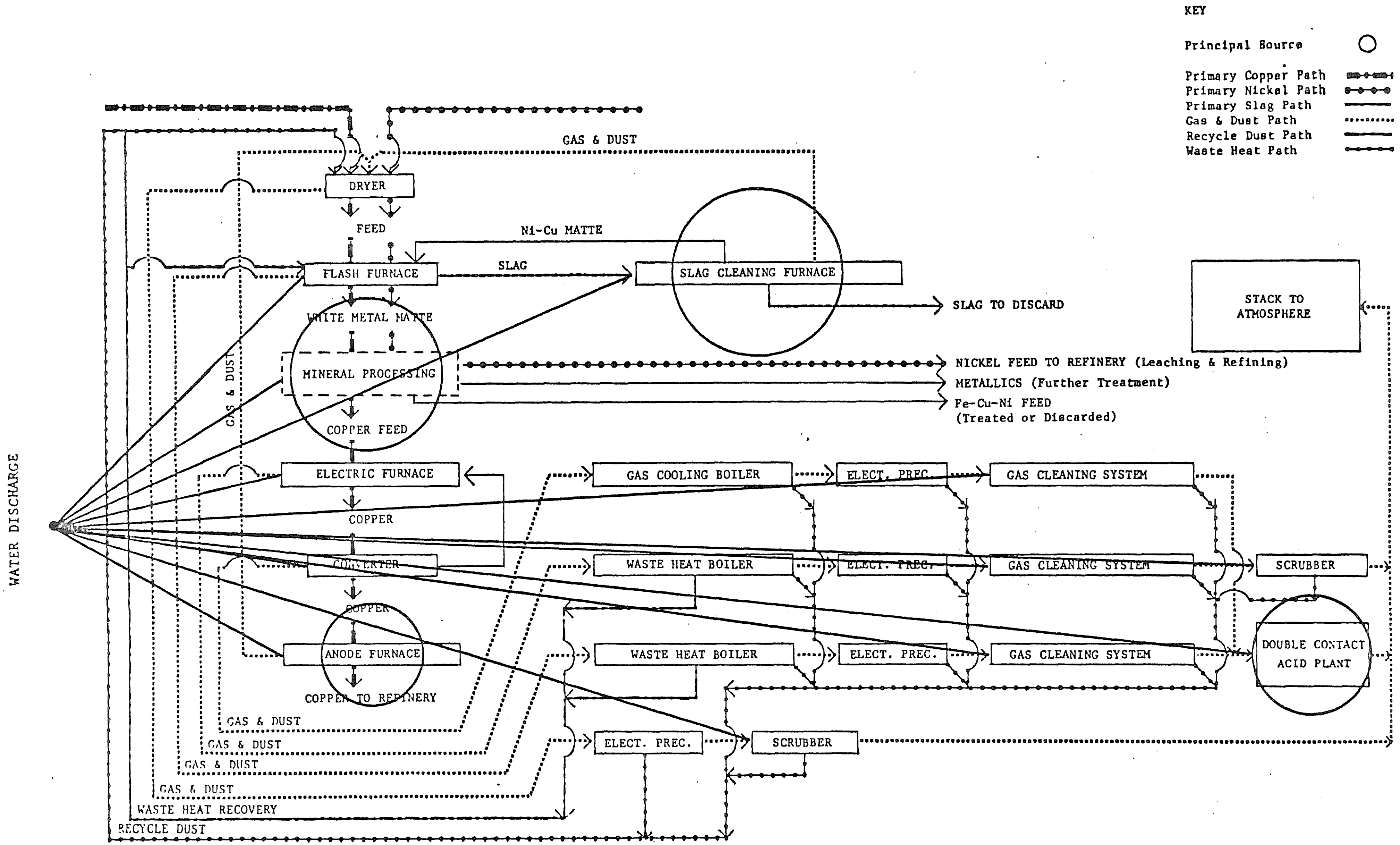


Figure 12. Generalized Flowsheet for Concentrate Smelting in the Production of Commercial Grade Copper and Nickel (Principal Sources for Solid Wastes).

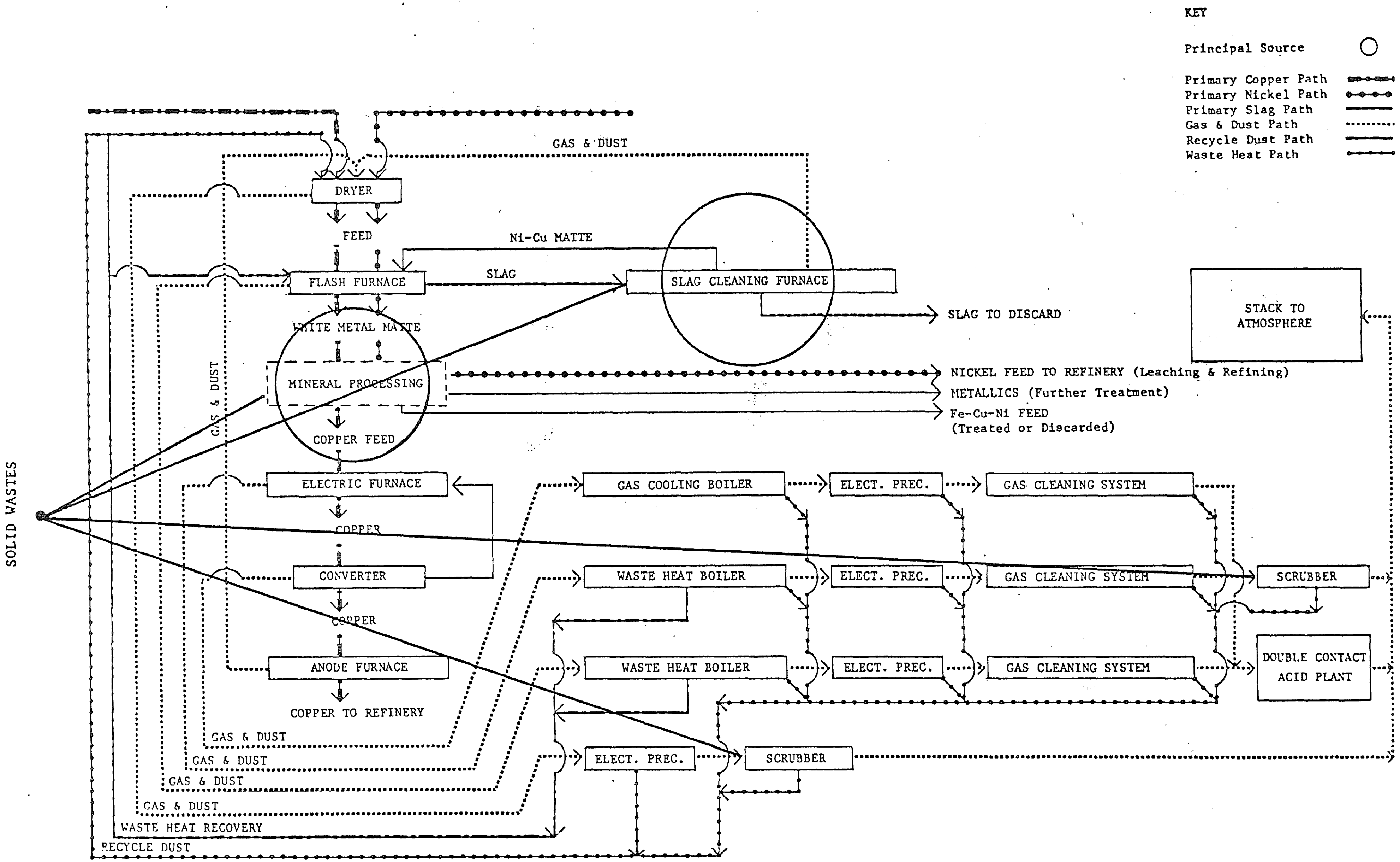


Figure 13. Generalized Flowsheet for a Copper and Nickel Refinery.

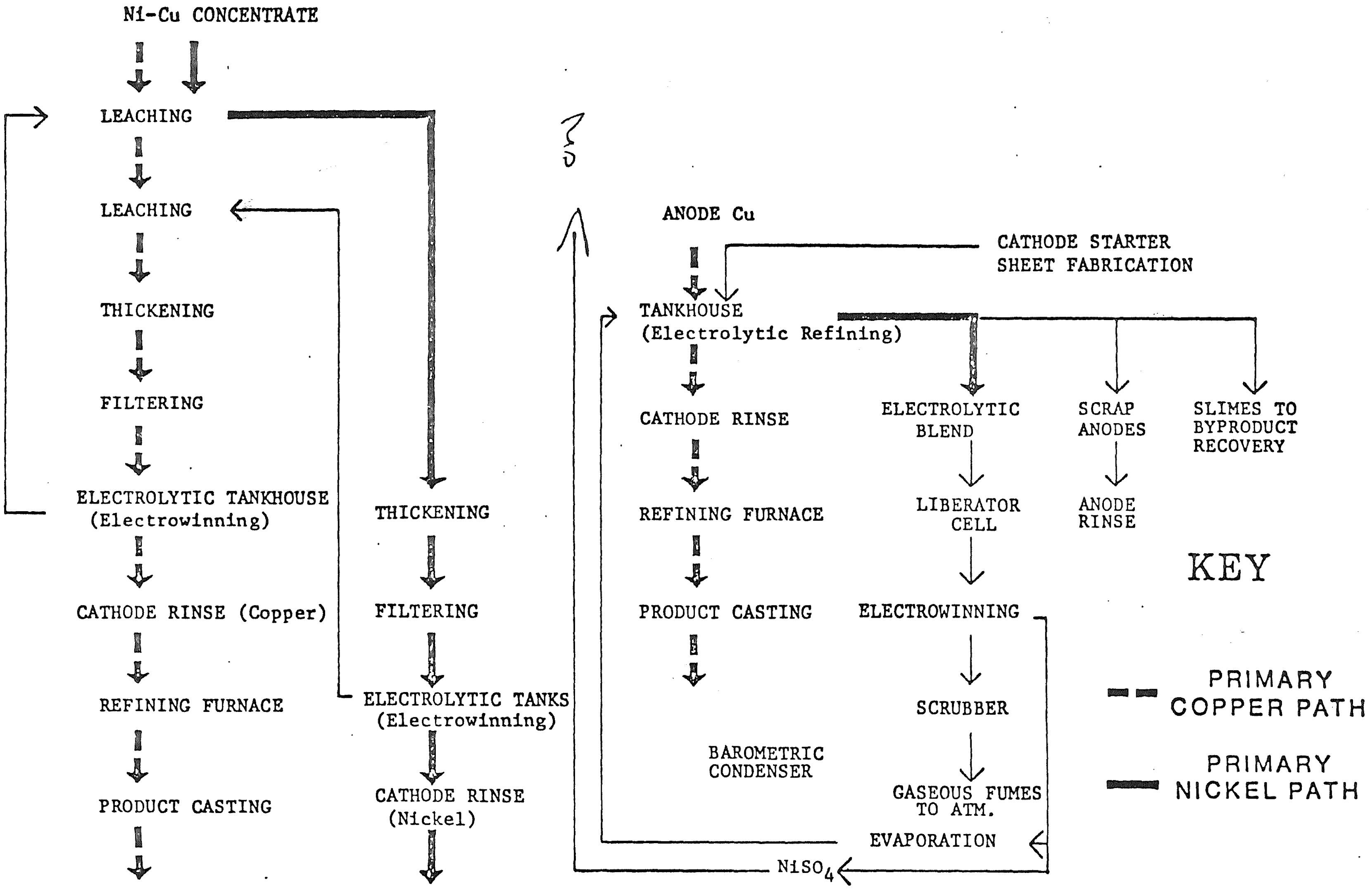


Figure 14. Generalized Flowsheet for a Copper and Nickel Refinery (Sources for Water Needs).

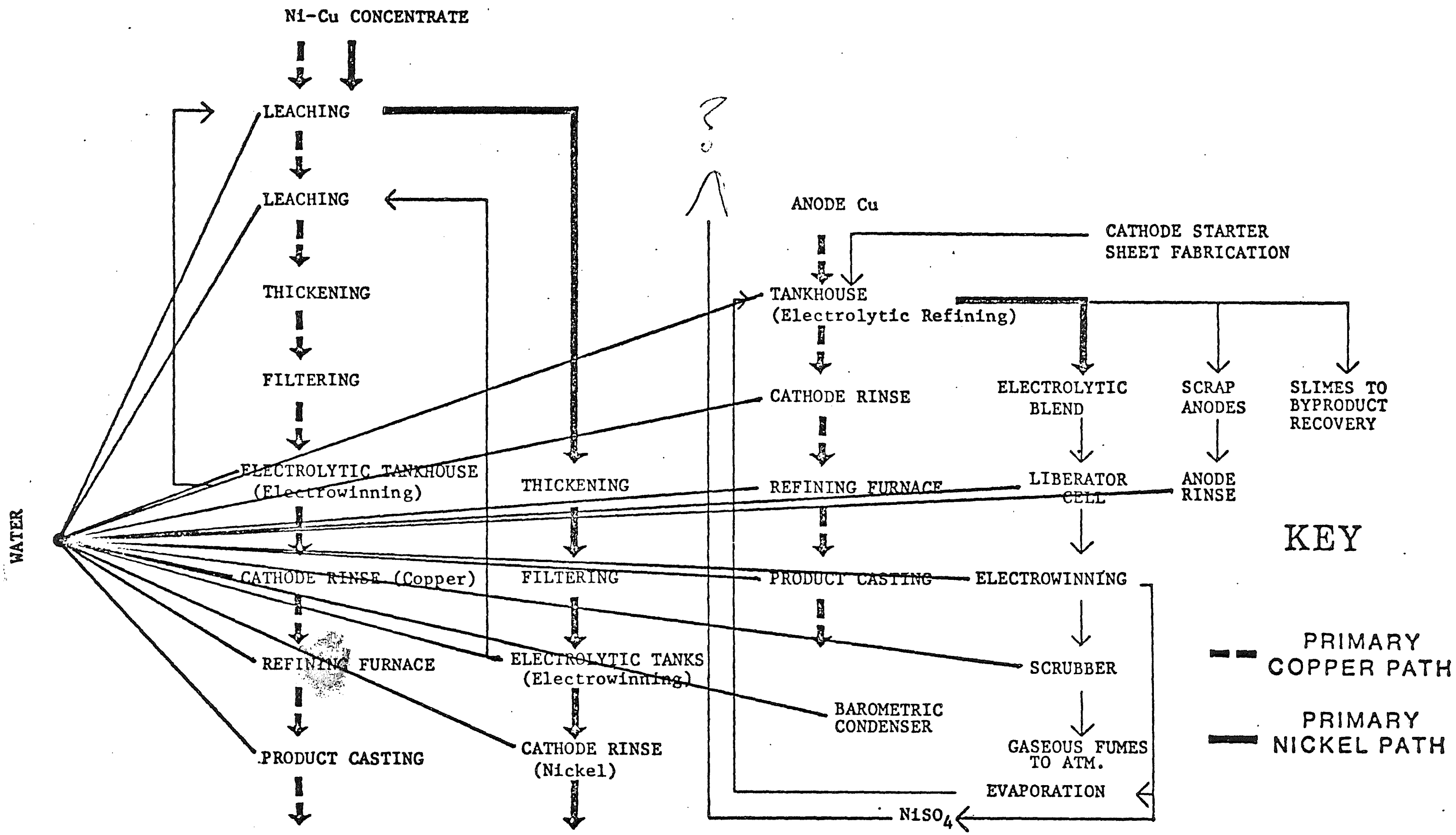


Figure 15. Generalized Flowsheet for a Copper and Nickel Refinery (Principal Sources for Water Discharges).

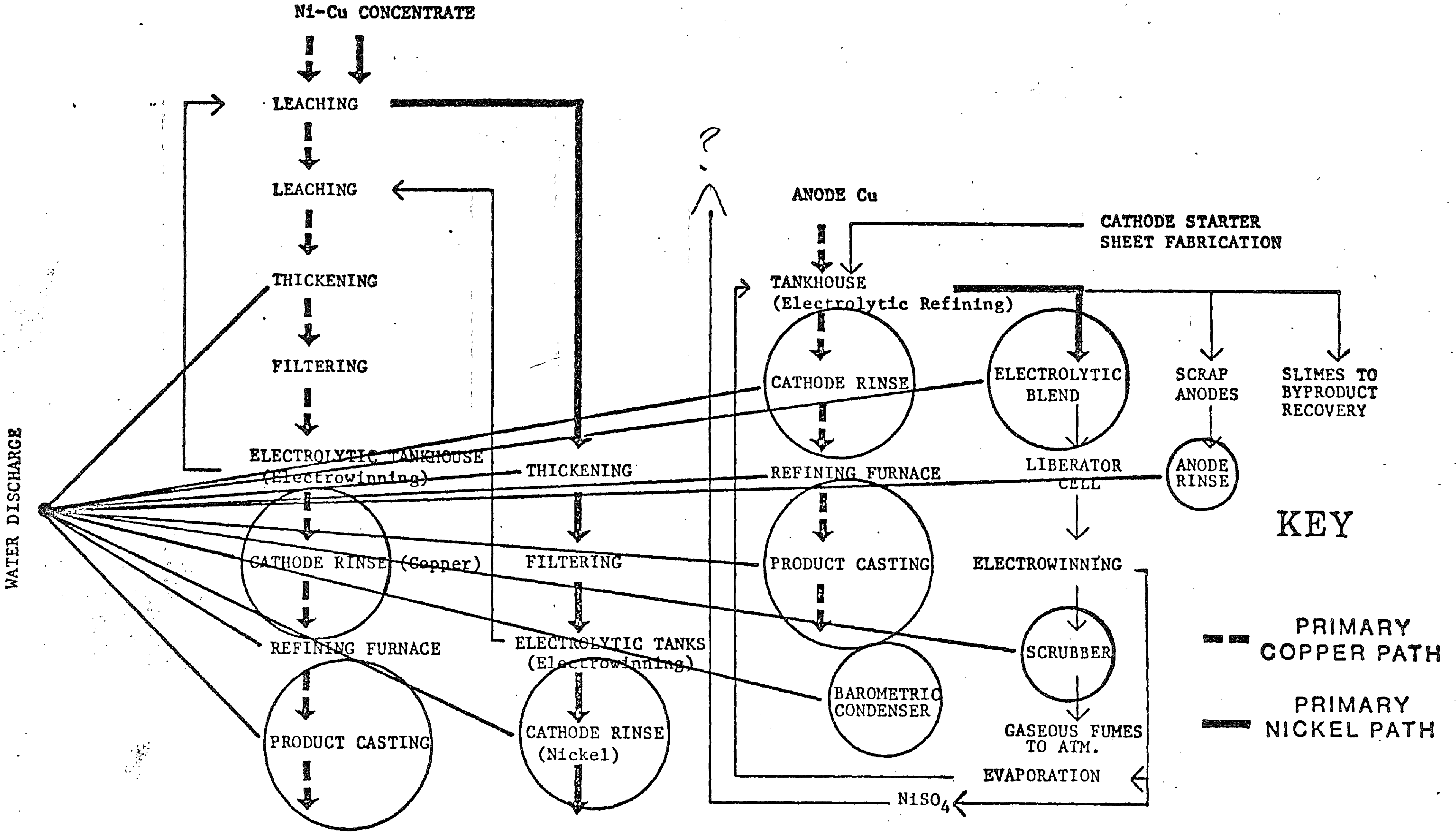


Table 3. Metallurgical Material Balance Summary (Quantities are in Metric Tons Per Year).

Elements

	Cu	Ni	Fe	S	SiO_2	CaO	MgO	Al_2O_3	As	Pb	Zn	Cd	Ag	Au	Rh	Pt	Pd
INPUT																	
1 Concentrate	66,118	9,929	153,911	119,613	66,586	6,546	23,436	12,879	13.52	41.62	1,965	17.72	23.26	0.72	0.0088	0.89	2.18
2 Additions	13,667	7,163	11,369	4,094	69,714				2.84	0.40	37	3.29					
OUTPUT																	
3 Metallics	7,812	1,103	320	2,765							89			0.70	0.0081	0.86	2.02
4 Fe-Cu-Ni Conc.	1,499	633	2,650	2,885						0.5	227	0.08	0.1				
Cathode Cu																	
5 Cu Refinery	49,361	30.4	0.48	0.41					0.122	0.37	3.95		0.26				
6 Ni Refinery	750	0.46	0.01	0.01						0.01	0.06		0.004				
7 Cathode Ni	0.18	6,064	0.06	0.06						0.04							
8 Recyclables	17,923.775	8,414.86	11,538.43	7,158.52	8,622					29.04	616.79	18.68	21.991				
9 Discard Slag	1,207	604	148,337	1,730	126,203	6,488	23,221	12,766	13.778	11.32	1,021	0.42	0.7	0.01	0.0006	0.02	0.07
Fugitive Emis.																	
10 Particulate	702	127	1,383	734	838	33	117	64	0.97	0.42	24.82	1.04	0.12	0.004	0.00006	0.005	0.01
11 Gas				1,087													
Stack Emis.																	
12 Particulate	530	95.2	1,051	559	637	25	98	49	0.79	0.32	19.28	0.79	0.08	0.006	0.00004	0.005	0.01
13 Gas				5,425													
Discharge Water									0.60								
14 Cu Refinery	0.045	0.08	0.02						0.1		0.1		0.005				

BASIS: 100,000 Metric Tons Per Year Production of High Grade Copper-Nickel Matte.

OVERALL COPPER AND NICKEL RECOVERIES all metals
(Numbers are in Percent Form)

CATEGORY	Cu	Ni	Fe	S	SiO_2	CaO	MgO	Al_2O_3	As	Pb	Zn	Cd	Ag	Au	Rh	Pt	Pd	key
% Element Lost	3.69	8.32	97.96	7.97	—	100	100	100	18.20	28.98	54.21	12.70	3.89	2.78	7.95	3.37	4.27	$\frac{(9+10+11+12+13+14)}{1} \times 100$
% Element Recovered	96.31	91.68	2.04	92.03	—	0	0	0	81.80	71.02	45.79	87.30	96.11	97.22	92.05	96.63	95.73	$100-\frac{(9+10+11+12+13+14)}{1} \times 100$
% Element Lost	3.06	4.83	91.22	7.71	93.67	100	100	100	15.34	28.82	53.21	10.71	3.89	2.78	7.95	3.37	4.27	$\frac{(9+10+11+12+13+14)}{1+2} \times 100$
% Element Recovered	96.94	95.17	8.78	92.29	6.33	0	0	0	84.66	71.18	46.79	89.29	96.11	97.22	92.05	96.63	95.73	$100-\frac{(9+10+11+12+13+14)}{1+2} \times 100$
% Element Recovered in Cathode Form	75.79	61.38	0.00036	0.0004	—	—	—	—	0.9	1.01	0.20	—	1.13	—	—	—	—	$\frac{(5+6+7)}{1} \times 100$
% Element Recovered in Cathode Form	62.81	35.66	0.00033	0.00039	—	—	—	—	0.75	1.00	0.20	—	1.13	—	—	—	—	$\frac{(5+6+7)}{1+2} \times 100$
% Element Recovered as "Mineral Processing Product" and "Refinery Product"	89.87	79.07	1.93	3.89	—	—	—	—	0.90	2.21	16.29	0.45	1.56	97.22	92.05	96.63	95.73	$\frac{(3+4+5+6+7)}{1} \times 100$
% Element Recovered as "Mineral Processing Product" and "Refinery Product"	74.48	45.93	1.80	3.76	—	—	—	—	0.75	2.19	15.98	0.38	1.56	97.22	92.05	96.63	95.73	$\frac{(3+4+5+6+7)}{1+2} \times 100$

BASIS: 100,000 Metric Tons Per Year Production of High Grade Matte.

(i.e. the metallics and the Fe-Cu-Ni concentrate) will be further treated for the recovery of individual elements (copper, nickel, precious metals, etc.). Assuming that any additional treatment of a particular output product will not result in significant elemental losses, several types of recoveries (depending upon the particular basis used) can be realized. Calculated recoveries and/or losses are also summarized in Table 3. Refer to the key to determine the basis for each calculated percent. Complete, detailed material balances are not available, and, therefore, it is not possible to calculate the true percent recovery or true percent loss of a particular element or compound. As a result, the stated quantity of a particular output product found in Table 3 is not necessarily the true weight to be produced from the proposed smelter/refinery model. The material balances found in the report predict the quantity of the major product values on the basis of the principal flow of copper and nickel, and do not take into account the products produced from secondary copper and nickel flow paths. Consequently, the amount or quantity of cathode copper as stated in Table 3 will necessarily be less than the actual amount or quantity of cathode copper produced from the proposed smelter/refinery model. This is, likewise, the case for the other elements as products in the balances.

Assuming overall recovery factors of 96.31 percent for copper and 91.68 percent for nickel (refer to Table 3 to see how these values are derived), Table 3 reduces to the input and output quantities as found in Table 4. Table 4 will serve as the basis for scaling either upwards or downwards to a desired size operation (yearly metal production). Figures 16 through 23, which deal with the total capital costs, direct operating costs, energy requirements, water requirements, fugitive particulate emissions, stack

	Cu	Ni	Fe	S	SiO_2	CaO	MgO	Al_2O_3	As	Pb	Zn	Cd	Ag	Au	Rh	Pt	Pd
entrate	66,118	9,929	153,911	119,613	66,586	6,546	23,436	12,879	13.52	41.62	1,965	17.72	23.26	0.72	0.0088	0.89	2.11
clables	13,667	7,163	11,369	4,094	69,714				2.84	0.40	37	3.29					
t																	
Cathodes	63,678	38	0.62	0.54					0.06	0.32			0.64	0.01			
Cathodes	0.27	9,065	0.09	0.09					0.05	0.05							
r Conversion cid				102,363													
ers (i.e. leach, due, dust for or sale, gas,)	0.73	0	3,139.29	7,711.37					10.95	29.19	899.8	15.47	21.715	0.69	0.0081	0.86	2.02
ard Slag	1,207	604	148,337	1,730	126,203	6,488	23,221	12,766	0.97	11.32	1,021	0.42	0.7	0.01	0.0006	0.02	0.07
tive Emissions																	
rticulate	702	127	1,383	734	838	33	117	64	0.79	0.42	24.82	1.04	0.12	0.004	0.00006	0.005	0.01
s				1,087													
k Emissions																	
rticulate	530	95.2	1,051	559	637	25	98	49	0.60	0.32	19.28	0.79	0.08	0.006	0.00004	0.005	0.01
s				5,425													
charge Water																	
Refinery	0.045	0.08	0.02						0.1		0.1		0.005				
clables	13,667	7,163	11,369	4,094	69,714				2.84	0.40	37	3.29					
ent Recovered	96.31	91.68	2.04	92.03	0	0	0	0	81.80	71.02	45.79	87.30	96.11	97.22	92.05	96.63	95.73
ent Lost	3.69	8.32	97.96	7.97	100	100	100	100	18.20	28.98	54.21	12.70	3.89	2.78	7.95	3.37	4.27

ASIS: ≈100,000 Metric Tons Per Year Production of White Metal Matte.

Table 4. Metallurgical Material Balance Summary (Smelter/ Refinery Model, Quantities not Scaled)(Quantities are in Metric Tons Per Year).

Figure 16. Total Capital Costs.

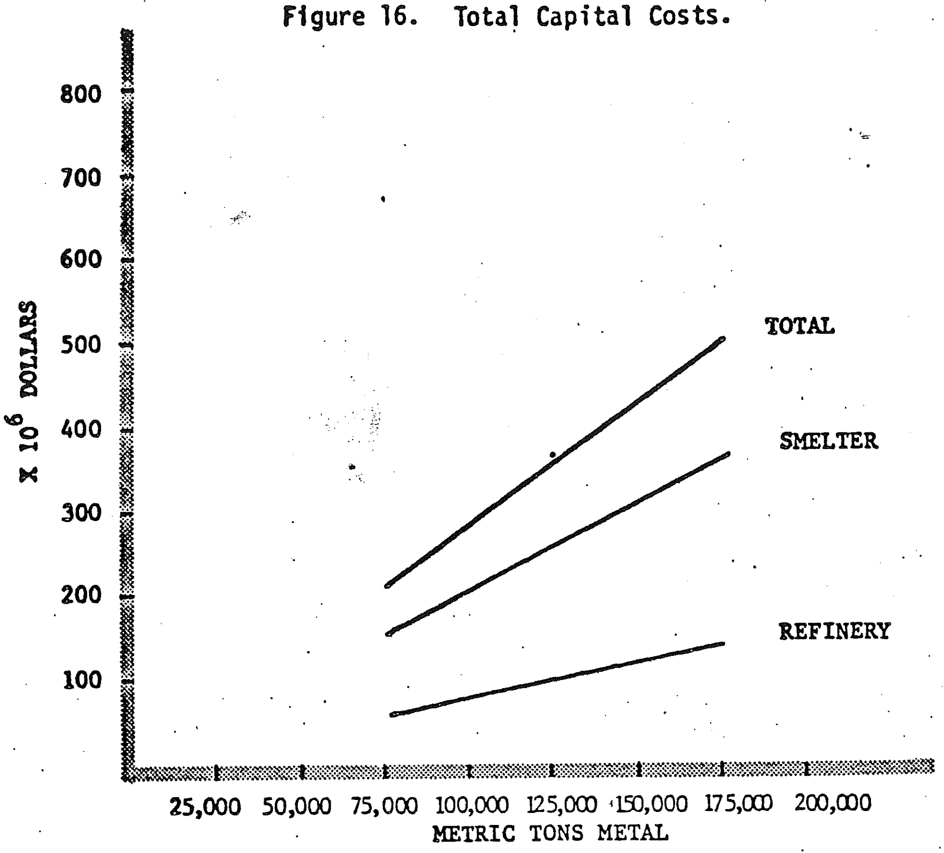


Figure 17. Direct Operating Costs.

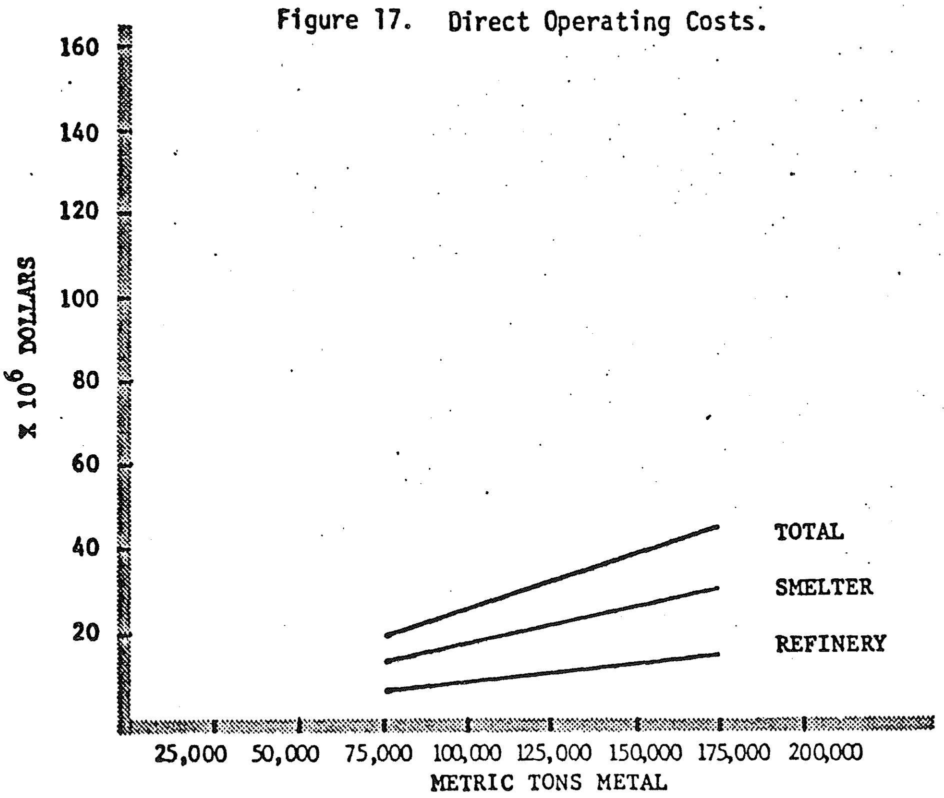


Figure 18. Energy Requirements.

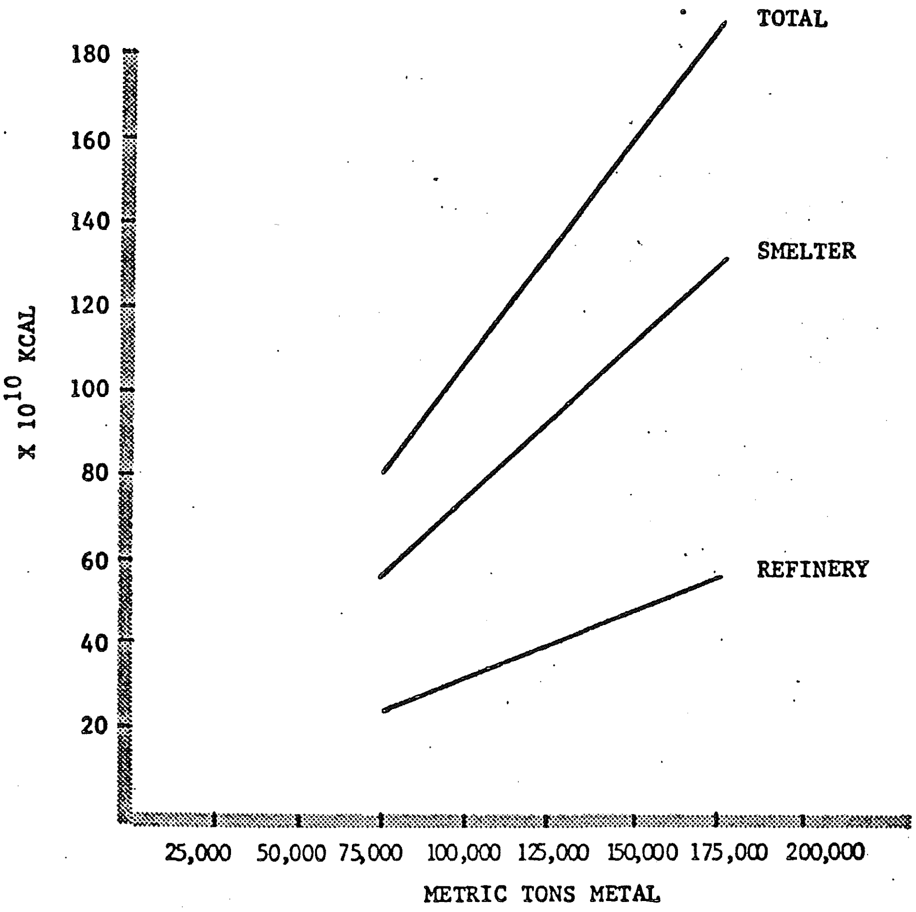


Figure 19. Water Requirements for Smelting.

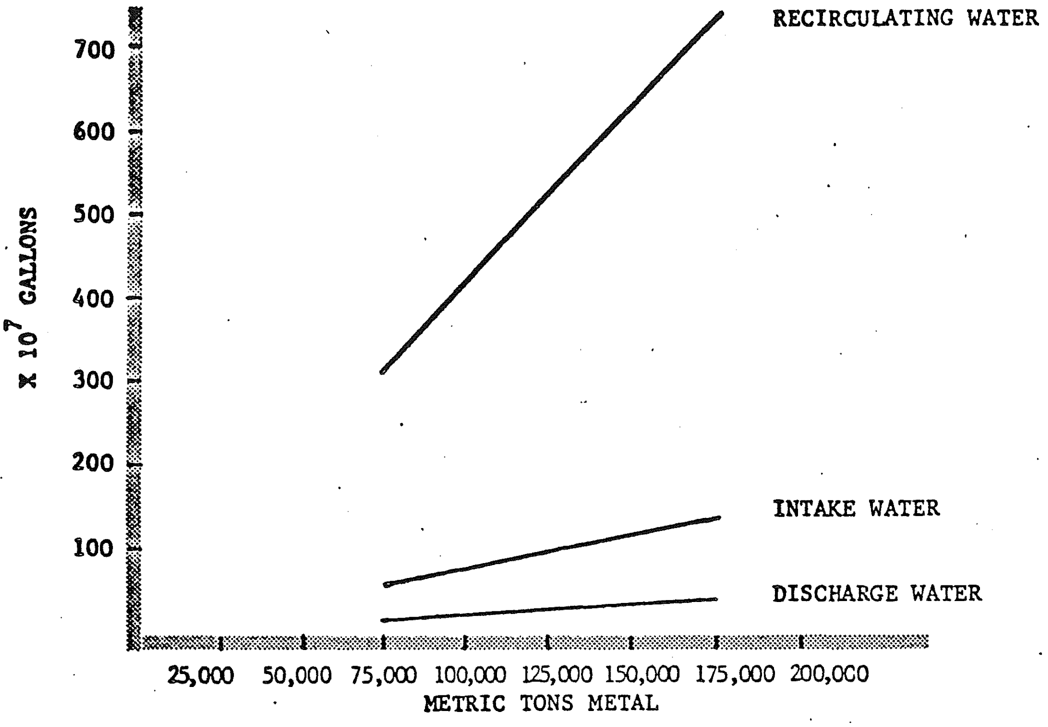


Figure 20. Water Requirements for Refining.

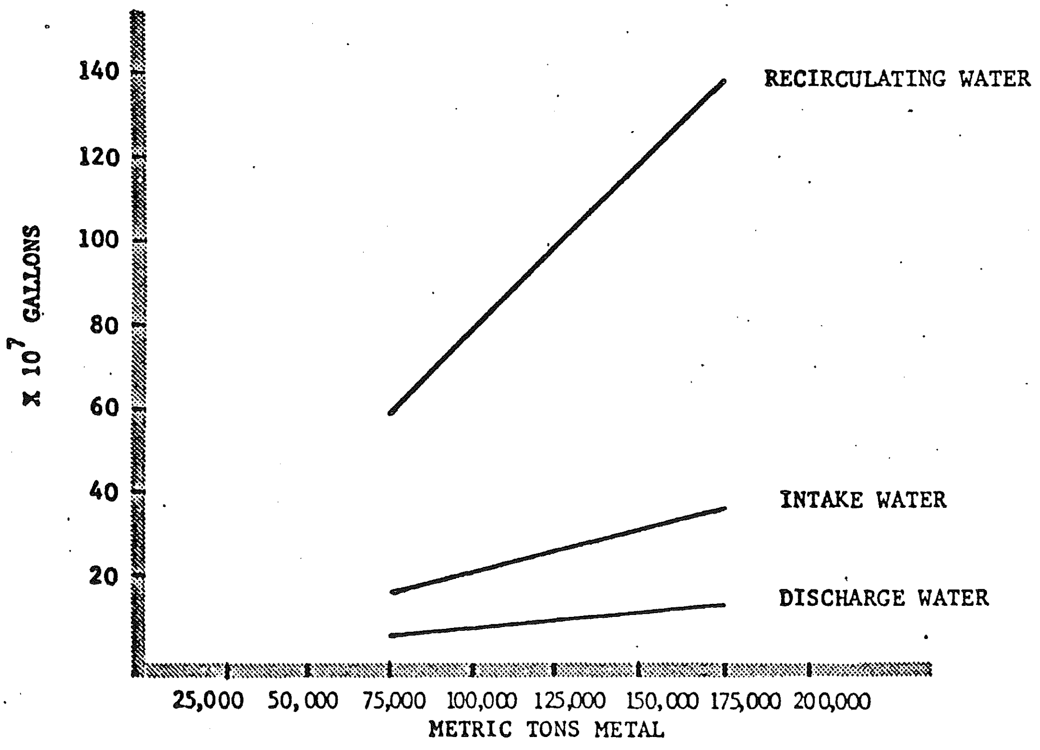


Figure 21. Fugitive Particulate Emissions (Major Elements).

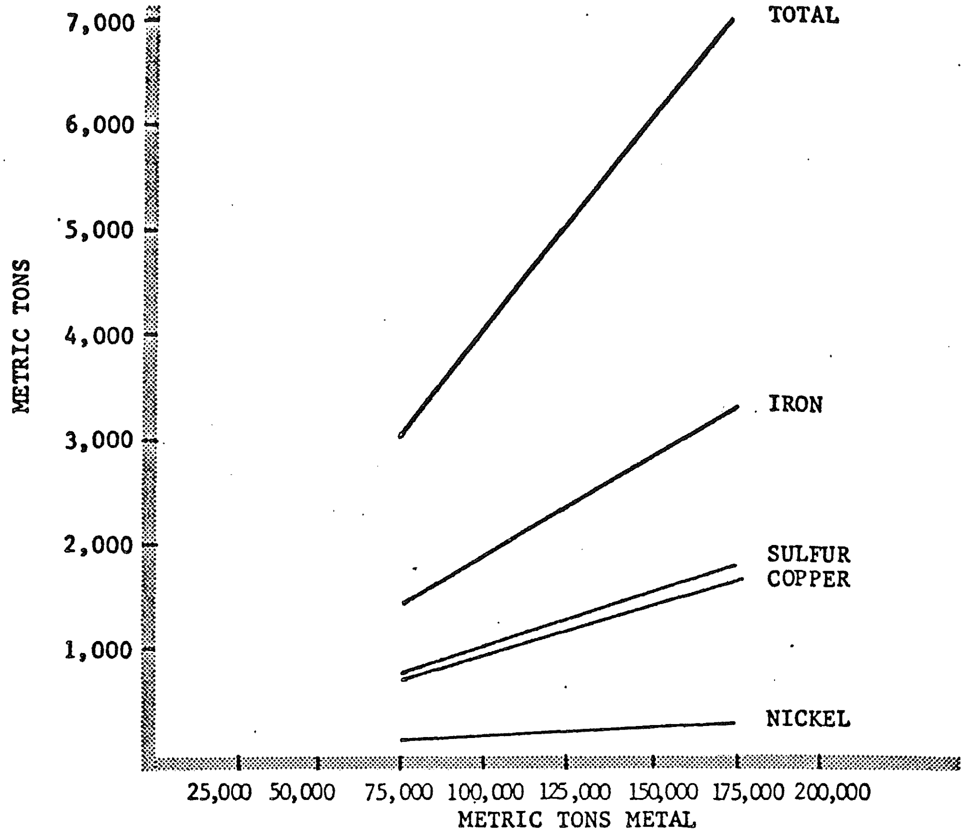


Figure 22. Stack Particulate Emissions (Major Elements).

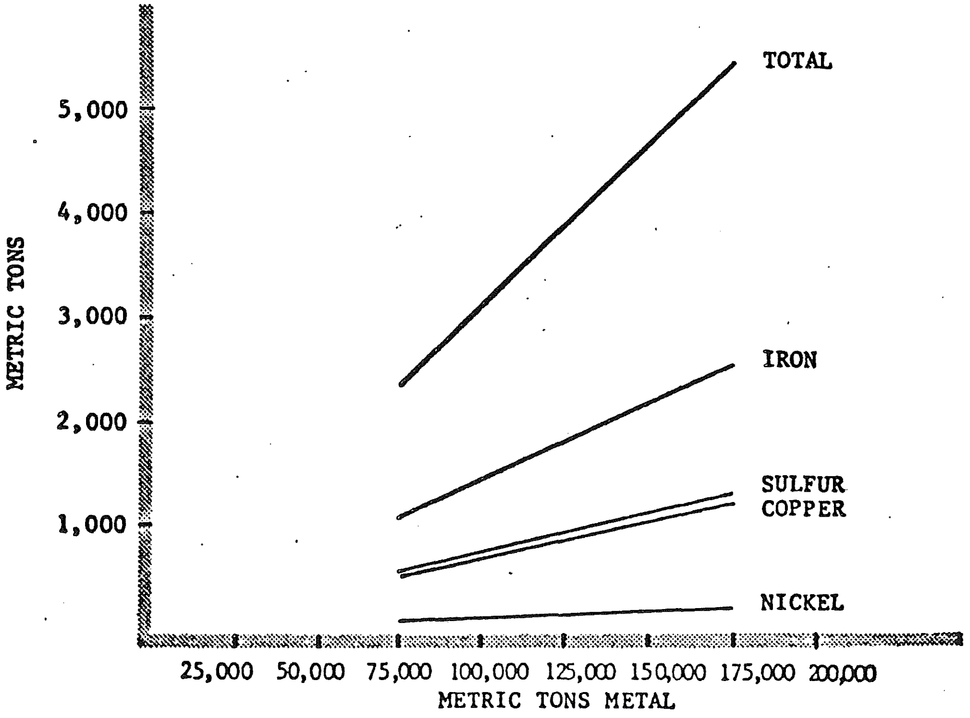
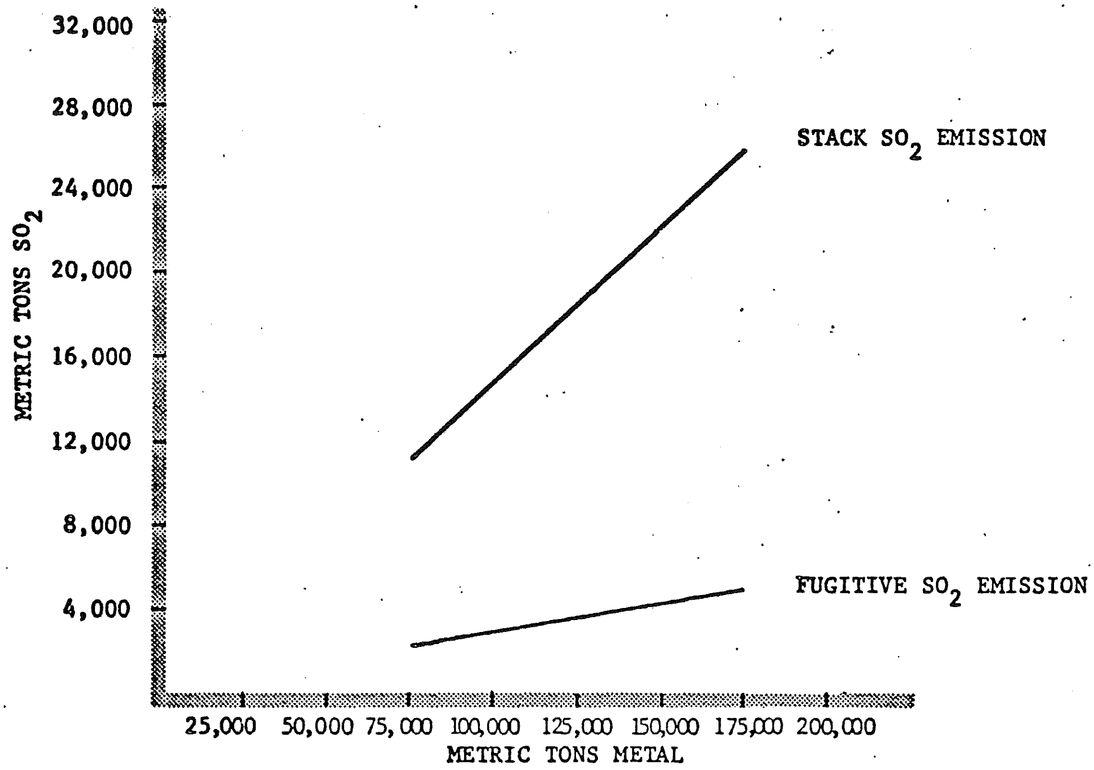


Figure 23. Fugitive and Stack SO_2 Emissions.



particulate emissions, and fugitive and stack SO_2 emissions for a smelter/refinery operation, summarize in graphical form the results of scaling the quantities found in 4 over the range of 75,000 to 175,000 metric tons per year of metal production. Cost data, energy values, and water needs are based in terms of unit measurement per metric ton of metal produced, and are most accurately defined for a 100,000 metric ton per year metal production. Consequently, the smelter/refinery model will be based on the production of 100,000 metric tons per year of metal, and Table 5 summarizes the cost, energy, water, manpower, land, and air emissions for such an operation (87,490 metric tons of copper per year, and 12,507 metric tons of nickel per year). Figure 24 also summarizes the principal final products to be anticipated based on the above recovery factors for copper and nickel.

PREFACE

INTRODUCTION

Ore-dressing is but one of the many vertical stages necessary in the process of isolating metal from its host rock, that is, concentrating or enriching the mineral portion of the ore. Further treatment, however, is required to completely eliminate the remainder of the gangue and to break down the mineral into its component elements for the freeing of metal. Just as different minerals require special methods to effect their concentration, so the extraction of various metals requires a diversity of treatment. Pyrometallurgy is one such treatment in which the transformation reactions are carried out by the application of heat supplied by the burning of fuel.

Table 5. Cost, Energy, Manpower, Land, Fugitive, and Stack Particulate and Gaseous Emissions For a Smelter/Refinery Operation Producing 100,000 Metric Tons of Metal Per Year.

TOTAL CAPITAL COST	
Smelter	20.46 X 10^7('77 dollars)
Refinery	7.96 X 10^7('77 dollars)
Total	28.42 X 10^7('77 dollars)
DIRECT OPERATINC COST	
Smelter	17.2 X 10^7('77 dollars)
Refinery	7.8 X 10^7('77 dollars)
Total	25.0 X 10^7('77 dollars)
ENERGY REQUIREMENTS	
Smelter	7.56 X 10^{11} KCAL
Refinery	3.26 X 10^{11} KCAL
Total	10.82 X 10^{11} KCAL
WATER REQUIREMENTS	
Smelter	
Intake	7.7 X 10^8 Gallons
Recycle	4.23 X 10^9 Gallons
Discharge	2.315 X 10^8 Gallons
Refinery	
Intake	2.1 X 10^8 Gallons
Recycle	7.9 X 10^8 Gallons
Discharge	7.94 X 10^7 Gallons
MANPOWER NEEDS	
Smelter	435 Men
Refinery	186 Men
Total	621 Men

LAND REQUIREMENTS	
Smelter	55 Acres
Refinery	25 Acres
Total	80 Acres
FUGITIVE PARTICULATE EMISSIONS	
Cu	965 Metric tons
Ni	174 Metric tons
Fe	1,900 Metric tons
S	1,008 Metric tons
STACK PARTICULATE EMISSIONS	
Cu	728 Metric tons
Ni	131 Metric tons
Fe	1,444 Metric tons
S	768 Metric tons
FUGITIVE SO_2 EMISSIONS	2,987 Metric tons
STACK SO_2 EMISSIONS	14,907 Metric tons

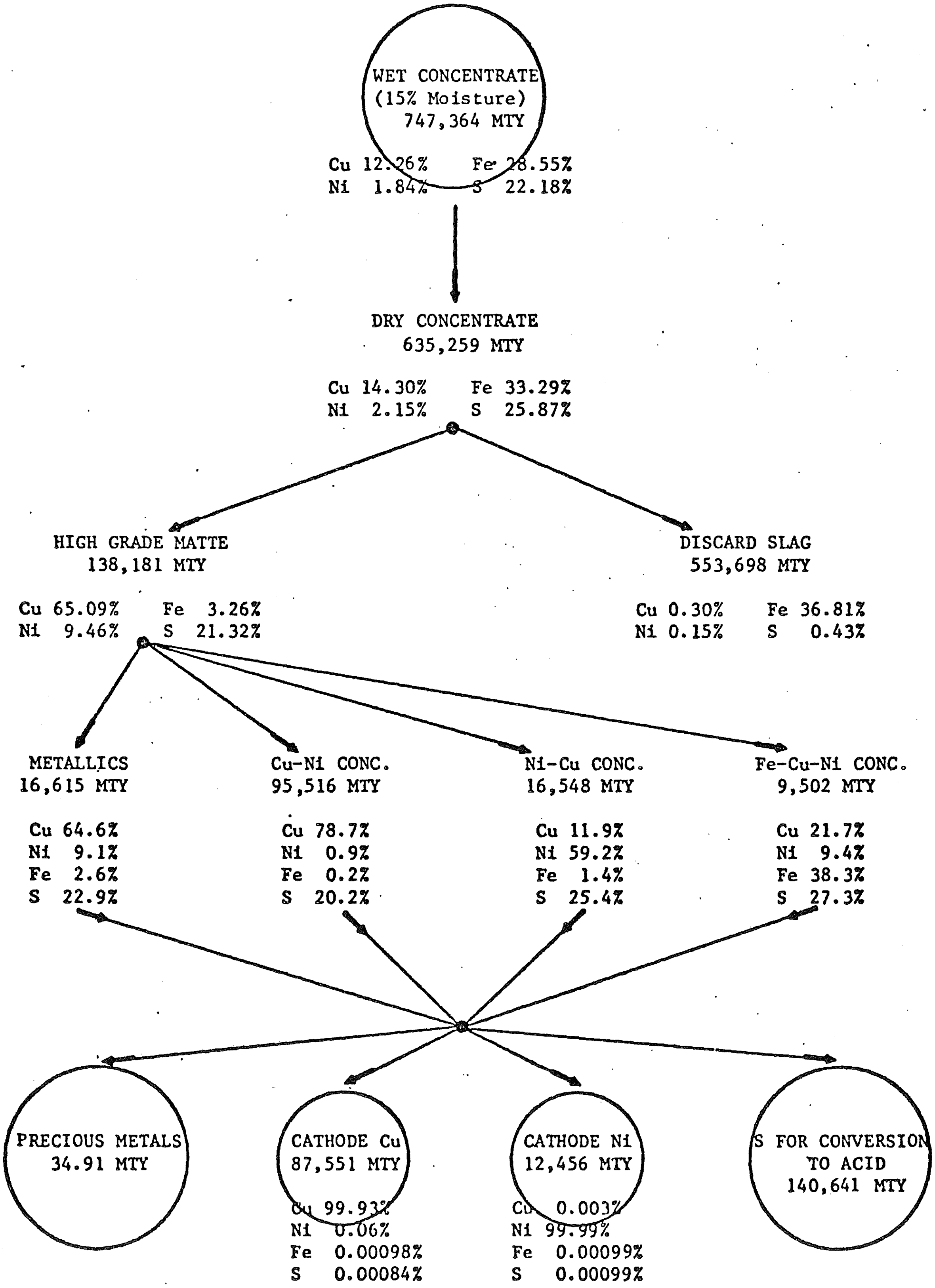
87,493 MTY Cu
12,507 MTY Ni

1.37394 Multiplication Factor

Overall Cu Recovery	96.31%
Overall Ni Recovery	91.68%

Cu/Ni = 7.00

Figure 24. Principal Final Products from a Cu-Ni Smelting/Refining Operation (Units are as Metric Tons Per Year; Elements Should Not Necessarily Balance Out).



Fire treatment usually plays an important role in producing metal from ore or concentrate. Both the quantity of heat as well as the intensity of heat effect metal recovery. Following is a review of the operations of roasting, smelting, and converting as used in the pyrometallurgical treatment of copper-nickel sulfide ores.

ROASTING

Roasting is defined as a metallurgical operation wherein a material is heated to an elevated temperature in a desired atmosphere to create a certain desired chemical change. Stated differently, during roasting, an ore is heated under such conditions and to such a temperature (below that of fusion of its mineral constituents) that the metal components of the ore which is sought for recovery, will be chemically changed from the form in which it occurs in the ore to some other form which will be amenable to some definite subsequent treatment. Roasting processes are generally of the oxidizing, reducing, sulfating, or chloridizing type. Oxidizing roasting involves oxidation of a portion of the sulfur and iron sulfide content of a sulfide ore or concentrate to iron and sulfur oxides.[30]

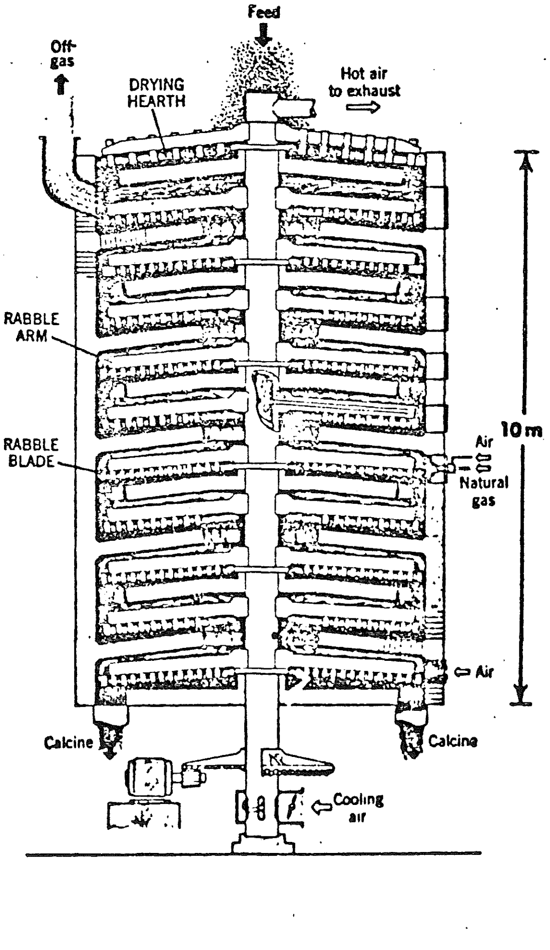
Typical types of roasters include:

1) mechanical roasters
2) agglomeration roasters
3) fluid roasters

Mechanical Roasters

The most widely used mechanical roasters today are of the superimposed multiple-hearth type. The ore is continuously raked by rotating rabble arms from hearth to hearth. During its passage down the furnace, the rabbling action continually exposes fresh surfaces to the oxidizing action of air. In the case of sulfides, provided sufficient sulfur is initially present, once roasting has commenced the heat generated by the reaction is sufficient to run the sulfur down to a low content without the need for extraneous heat. The gases leaving the top of the roaster contain from 2 to 6 percent SO_2 (low for sulfuric acid manufacture) and carry away approximately 6 percent of the roaster calcine product. Sulfur elimination in the roaster is varied by regulation of air flows and charge retention time on the hearths (Figure 25).[30]

Figure 25. Cutaway View of a Multiple Hearth Roaster.
(Source: Biswas, 1976)



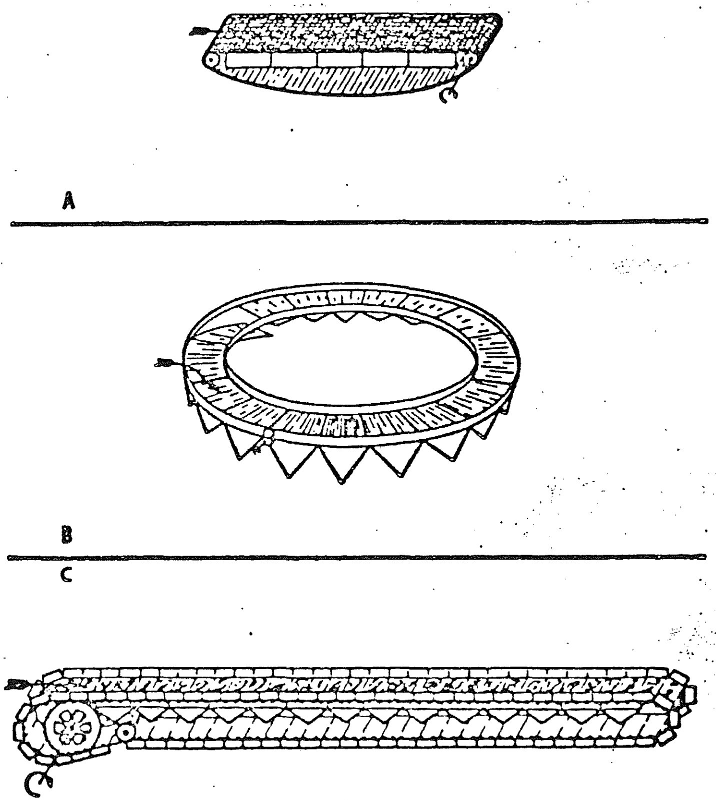
Agglomeration Roasting

The chain, circular, and Dwight-Lloyd types of traveling grates can be considered as proven industrial agglomeration roasting machines. Reactions are usually performed with the beds supported horizontally, though in some unusual cases, grates have been mounted on inclines to elevate the discharge end. Wind boxes and, in some instances, hoods used for confining draft that is either induced or formed by blowers, are used as auxiliaries for traveling grates. External igniting or firing is performed by the use of open torches in cases of sintering, by incandescent refractory heat reflectors in cases of coal combustion with autogenous ignition, or by elaborate combustion chambers mounted in the hoods of the traveling grates when substantial quantities of heated draft are required. In some processes, recirculation and recuperation of draft are used by compartmenting and connecting the wind boxes and hoods with blower arrangements.

The circular type of grate operates by having the grate carriers under compression and tension induced by a friction or gear-driven mechanism outboard of the ring of grate members. The circular grate revolves in a horizontal plan and offers the unique feature of having grate components in continuous use throughout the processing period. Discharge is accomplished by a tripping mechanism which can be operated by gravity or by a separate gear-tilting mechanism. This type of grate was applied very early in the sintering of ores. For this service, relatively heavy-duty components are required.

The Dwight-Lloyd type of traveling-grate is operated by a tail end drive which pushes pallet cars (containing grate bars and side walls) through the processing zone. Pallets made of castings are arranged as a train of cars under compression. Individual pallets are removable to allow the insertion of spare pallets. The relatively heavy narrow pallet and grate members serve as a heat sink and consequently allow processing bed temperatures as high as 3000°F. The Dwight-Lloyd machine has been extensively applied for high-temperature downdraft operations in the production of metallic and nonmetallic sinter and pellets.[3] (Figure 26)

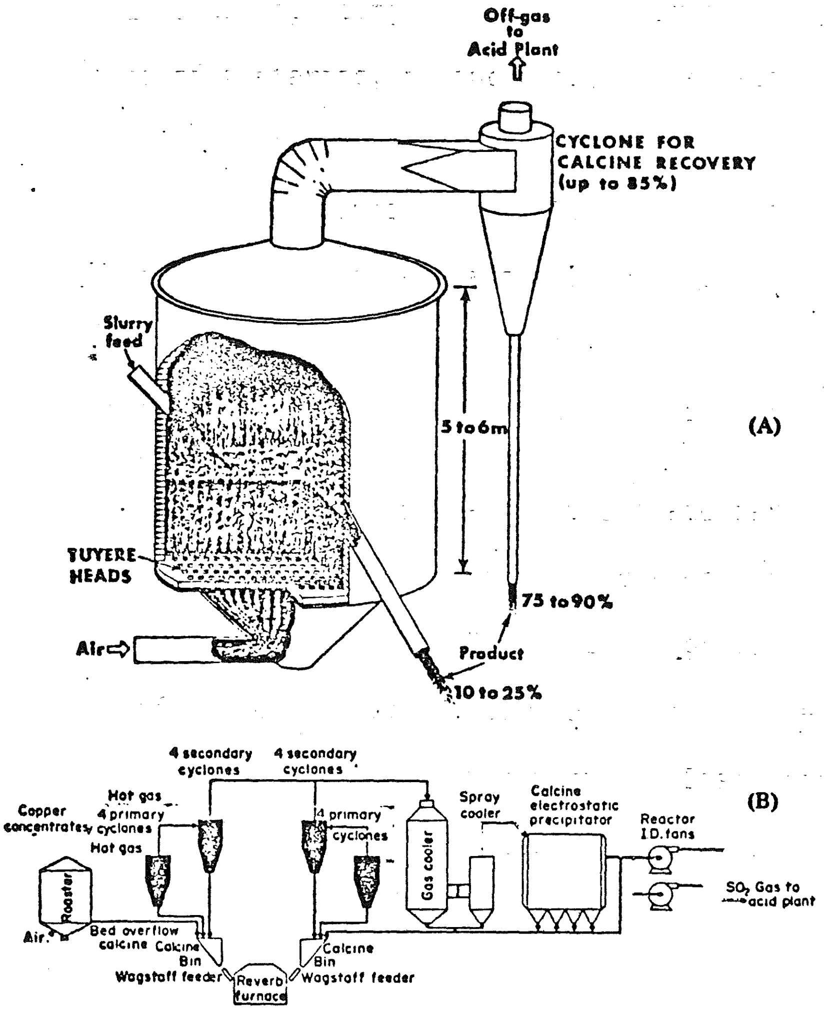
Figure 26. Simplified Illustrations of Various Types of Traveling Grates: Chain or Stroker Type (A), Circular Type (B), Dwight-Lloyd (C). (Source: Ban)



Fluid Roasting

Fluid bed roasting is characterized by a gas-solid reaction in a dense suspension of solids maintained in a turbulent mass by the upward flow of gases that affect the reaction. The roaster is essentially a cylindrical refractory-lined steel shell used to contain the suspended solids (Figure 27).

Figure 27: (A) Cutaway View of a Fluid-Bed Roaster, and (B) Roaster-Reverberatory Furnace Flowsheet. (Source: Biswas, 1976)



Air is forced into the roaster through tuyeres in a refractory-lined steel construction plate that is placed at the bottom of the shell. Calcine is continuously exhausted from the reactor and recovered in cyclone dust collectors. Once started, combustion is self-sustaining. The heated suspension of solids in the reactor diluted by the air/gas streams acts in many ways like that of a fluid. Material will flow like water into and out of the reactor. It will exert a gravity head which is used to effect movement of the mass. Upon the introduction of heat, either in the form of gas or fuel, the fluidized bed, because of its violent motion, quickly reaches a uniform temperature throughout. The temperature of the reactor can be controlled by regulating the volume of combustion air and by the admittance of spray water to the chamber.[21]

In roasting, concentrated sulfide ore is dried and heated until the sulfur begins to combine with oxygen to form sulfur dioxide gas (SO_2) which is removed in a gaseous stream. Ores rich in sulfur are roasted to lower the sulfur content to a level adequate for smelting which is the next operation to follow (ores low in sulfur content generally bypass the roasting stage).

The affinity between copper and sulfur is greater than that between sulfur and any of the other metals present. Consequently, in smelting, the copper will take enough of the sulfur to form Cu_2S, the residual sulfur combining with the iron, nickel, etc. The amount of FeS which will accompany the Cu_2S in the matte is therefore entirely a question of how much sulfur is left to combine with the iron after all the copper has been satisfied, the amount being controllable by the preliminary roast, which in turn controls the grade of the matte. If, therefore, a high-grade (40-70 percent Cu) matte is required, the roasting is carried to a high degree and for a low-grade matte the roast would not be pushed to the same extent. The most desirable grade of matte is a question which is decided on an individual smelter basis. Several factors assist in deciding the question, e.g., relative amounts of copper, nickel, and sulfur initially present in the ore, relative furnace and converter costs, furnace capacity, etc. These factors are by no means uniform as is realized by the fact that the copper content alone of mattes produced by various smelters varies from 20 to 70 percent. A compromise is usually reached on the production under prevailing conditions of the most suitable grade.

Roasting is usually autogenous, however, some hydrocarbon fuel may be required if the concentrates are excessively moist. Temperatures generally range from 500-700°C in the roasting chambers (temperatures of 500-550°C are maintained with iron enriched ores in order to avoid the formation of iron oxides). Roasting is not applicable to blast-furnace, flash, or single-step smelting all of which incorporate the roasting reactions in the smelting step.

Roasting produces SO_2 and volatile oxides of arsenic, antimony, lead and other trace elements. Roasting, therefore, involves a net elimination of sulfur and leads to a smelting product (matte) which is of considerably higher copper grade than the original concentrate.

Dust content generated during the roasting reactions depend principally on:

- characteristics of the copper concentrates
- volume of air aspirated by the roasting furnace
- extent to which concentrates remain continuously in suspension in traveling from hearth to hearth
- size and number of the aperatures in the hearth

SMELTING[4]

Smelting is a process for metal extraction involving the reduction of feed to molten metal or matte (matte is an artificially created sulfide), which easily separates from a siliceous iron slag which also forms in the melt. Gangue material usually melts at higher temperatures than the matte, so it is necessary to add fluxes which allow this gangue to become sufficiently liquid at the furnace temperature to form the slag. The major metals simultaneously combine with sulfur to form the valuable matte.

Treatable materials in the formation of mattes include:

1) naturally occurring sulfides
2) partially roasted sulfides
3) mixtures of sulfides and oxides

In reference to a predominantly Cu-bearing calcine, two theories exist concerning the chemical reactions taking place during smelting. The older version is based on a neutral or slightly oxidizing atmosphere wherein copper has a greater affinity for sulfur than does iron. Little or no sulfur is essentially lost in the system. All of the copper combines with the available sulfur to give the stable Cu_2S form. The remaining sulfur combines with available iron to give the stable FeS form. The remaining iron and gangue combine as a slag. In the newer version, however, the affinities of both copper and iron for sulfur are considered to be approximately equal at 1300°C. It is believed that the difference in the affinity of oxygen by copper as opposed to iron is responsible for the separation of copper into the matte.

1) $2Cu\ (\ell) + FeS\ (\ell) = Fe\ (\gamma) + Cu_2S\ (\ell)$

$$K = 3$$

2) $Fe\ (\gamma) + Cu_2O\ (\ell) = FeO\ (\ell) + 2Cu\ (\ell)$

$$K = 1300$$

3) $Cu_2O\ (\ell) + FeS\ (\ell) = Cu_2S\ (\ell) + FeO\ (\ell)$

$$K = 3900$$

The above chemical equations, with their corresponding equilibrium constants, indicate iron's greater affinity for available oxygen than that of copper for oxygen. The iron tends to combine with oxygen, whereas the copper tends to remain in combination with sulfur.

For copper-nickel bearing calcines, smelting occurs in a similar fashion with copper sulfides and nickel sulfides forming the matte.

With this in mind, a review of available smelting apparatus is in order. Typical types of smelting equipment include:

- Conventional Blast Furnace
- Momoda Blast Furnace
- Reverberatory Furnace
- Electric Furnace

→ Smelting Furnaces

- Outokumpu Furnace
- INCO Furnace

→ Flash Smelting Furnaces

- Noranda Furnace
- WORCRA Furnace
- Mitsubishi Furnace
- Britcosmaco Furnace
- USBM Furnace

→ Continuous Smelting Furnaces

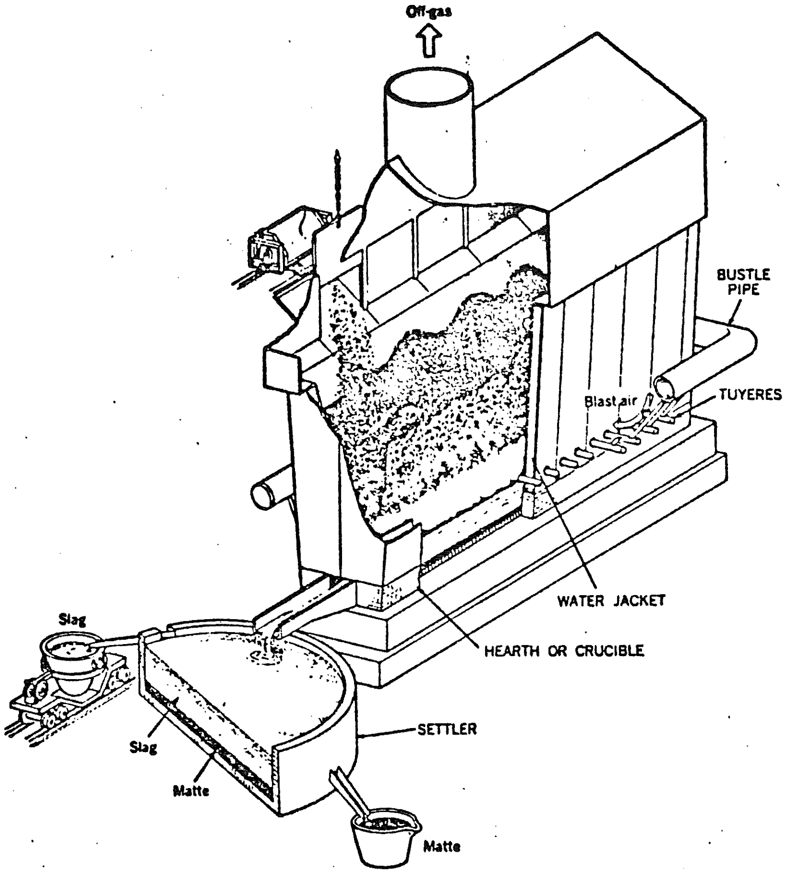
Processing operations for the conventional smelting furnaces include only the smelting phase. Flash smelting equipment combines both phases of roasting and smelting in a single confined unit. Continuous smelting apparatus, on the other hand, combines the phases of roasting, smelting, and converting (with slag cleaning in some instances) in a single contained unit.

Conventional Blast Furnace

The blast furnace consists essentially of a tall vertical hollow shaft, rectangular in cross-section, surmounting a hearth or crucible. Air required for combustion is supplied by a blower; it passes into the blast main surrounding the furnace and from there to the tuyeres, hence into the furnace. Where the air enters the furnace, very high temperatures are generated by the combustion of fuel, and as the products of combustion ascend, they yield up their heat to the charge. Gases and fumes escape from the top of the furnace through a flue located just below the charging floor. Metal and slag separate according to their specific gravities and are tapped from the crucible. The blast furnace, although simple in construction, is elastic in operation and is capable of high efficiencies. Coarse or sintered ore, however, is required for efficient working (Figure 28).[30]

Blast-furnace smelting was used extensively in the past for producing large quantities of matte from lump sulfide ores. It was also used at one time to produce a crude, iron-contaminated "black copper" from oxide ores. However, depletion of rich lump ores and the increasing prevalence of froth flotation concentrates have gradually eliminated the blast furnace from

Figure 28. Cutaway View of a Blast Furnace for Producing Copper Matte from Sulfide Ores. (Source: Biswas, 1976)



matte smelting. The blast furnace is unable to directly treat finely-ground flotation concentrates because they are quickly blown out of the furnace by the rising combustion gases. However, the blast furnace still finds some use (usually with a sintered charge) in Africa where mixed oxide-sulfide ores are treated, in Canada for Cu-Ni concentrates, and in Japan. World-wide there are few than ten matte blast furnaces remaining in operation (1974).

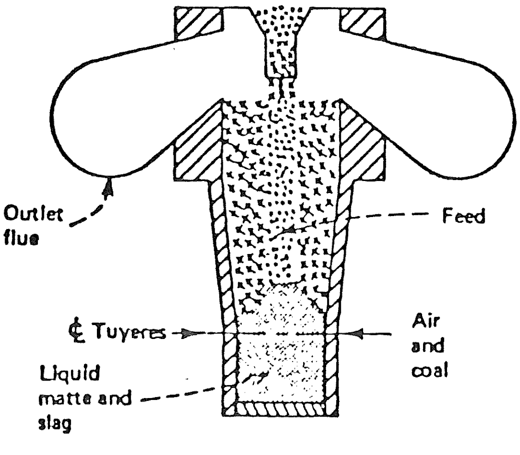
Typical charge to the blast furnace consists of sintered concentrates, lump ores, silica fluxes, converter slags, and metallurgical cokes. The products of the furnace include:

- a copper-rich liquid matte (≈ 50 percent Cu)
- a liquid slag
- gases containing approximately 5 percent SO_2
  (gases are generally laden with dust)

Momoda Blast Furnace

The Momoda process involves the feeding of plasticized concentrates directly into a furnace without prior briquetting or sintering. The plasticizing step consists of thorough kneading of fine copper concentrates, flue dust, and other fine material such as cement copper into a stiff plastic mass containing 10 to 15 percent water. The furnace is charged alternately at approximately one-half hour intervals through a central hopper opening, first with plasticized concentrates, and then with a coarse mixture of coke, silica flux, limestone, crushed converter slag, and crushed skull material. The plasticized concentrate represents 50 to 60 percent of the total charge to the furnace (Figure 29).

Figure 29. Specially-Designed Hopper Feeds Momoda Furnace Without Briquetting or Sintering of Concentrates. (Source: Treilhard, 1973).

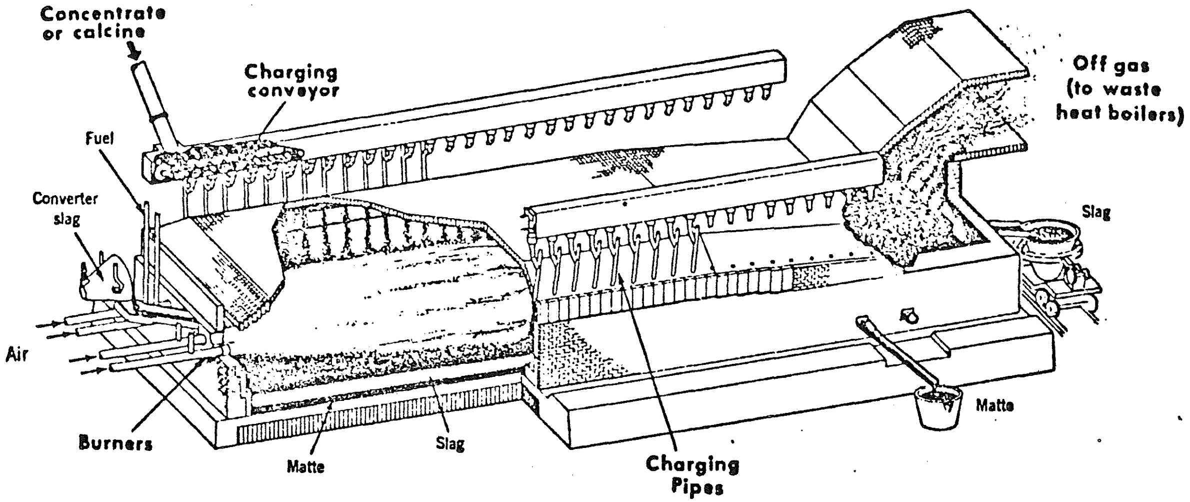


Reverberatory Furnace

The reverberatory furnace is composed of a shallow hearth with side and end walls surmounted by an arch roof. The hearth is well constructed, being built of concentrate overlaid with silica brick and sintered sand. Side walls are of silica brick with frequently two or more courses of magnesite brick at the level of the slag line. The arch, or roof, is composed of a span of silica or magnesite bricks held in place by iron plates which run along the longer sides of the furnace and are bolted to cast-iron uprights set in the foundations. The plates take up the thrust, the weight resting on the side walls. Horizontal tie-rods across the top of the furnace, bolted to the vertical uprights, serve to hold the furnace rigid. Heating is usually by pulverized fuel or gas injected into the furnace via burners inserted in the end wall. The ore is charged either through holes in the roof or through openings in the side walls. Metal is withdrawn through a tap-hole in the side of the furnace, slag either being tapped from the opposite side or, as in the case of copper, running continuously through a tap-hole in the front end (Figure 30).[30]

The reverberatory furnace is the most widely used unit for smelting. The dimensions of reverberatory furnaces vary considerably, but modern furnaces are typically 33 m long (inside), 10 m wide and 4 m high (hearth to roof). Furnaces of these dimensions produce in the order of 500 to 800 metric tons of matte (35-40 percent Cu) and 500 to 900 metric tons of discard slag per day. A smelter usually has from one to three such furnaces. The reverberatory furnace is also used (simultaneously with smelting) to recover copper from recycled, molten converter slag.

Figure 30. Cutaway View of a Reverberatory Furnace for the Production of Copper Matte from Sulfide Concentrates on Roasted Calcines. (Source: Biswas, 1976)



The extensive use of reverberatory furnaces is due to their high degree of versatility. The heat for smelting is provided by burning fuel in the furnace and by passing the hot combustion gases over the charge. All types of material, lumpy or fine, wet or dry, can be readily smelted. There are approximately 100 reverberatory furnaces in use world-wide (1975).

During the melting phase in a reverberatory furnace, approximately 20 to 50 percent of the sulfur in the charge is oxidized. The range of sulfur dioxide concentration in the wet offgas is 0.5 to 2.5 percent depending on the feed and operating conditions. Of the sulfur in the charge, 70 to 80 percent is removed in the matte, 18 to 30 percent in the flue gas, and 1 to 2 percent in the slag.

Dust from the reverberatory furnace is a substantial problem. Heavier particles settle below the waste heat boilers and into the hoppers of the

2) The SO_2 concentrations of its effluent gases are readily controlled by adjusting the amount of air infiltration into the furnace. With a minimum amount of air infiltration, the SO_2 concentration may be as low as 0.4 percent, for venting directly to the atmosphere, while extensive air infiltration and sulfide oxidation leads to SO_2 concentrations in the order of 5 percent. This latter gas is blended with converter gases and the SO_2 is extracted from both as sulfuric acid.

The electric furnace has the same versatility as the reverberatory furnace, and this factor plus the environmental advantages have led to its adoption for new smelting capacity in several environmentally sensitive areas. World-wide there are approximately 15 electric smelting furnaces in operation, including those used for producing Cu-Ni mattes.

Mattes and slags produced in electric furnaces are similar in nature to the mattes and slags from the reverberatory furnace. SO_2 gas emissions generally range from two to four percent with minimal dust losses, a consequence of low gas velocities.

Flash Furnaces

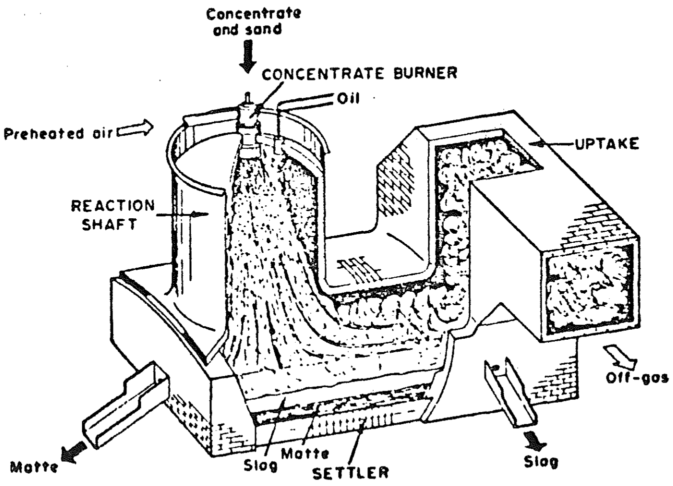
Flash furnaces for smelting are of two types:

1) The Outokumpu Furnace
2) The INCO Furnace

In the flash smelting process, flotation concentrates are injected along with flux and either preheated air or oxygen into a combustion chamber maintained at smelting temperature by "flash" combustion of iron and sulfur while the particles are in suspension. The heat of the exothermic oxidation reactions is used in the smelting, with any heat deficiency augmented by a small addition of fuel, preheating of combustion air, enrichment of the combustion air with oxygen, or a combination of the above.

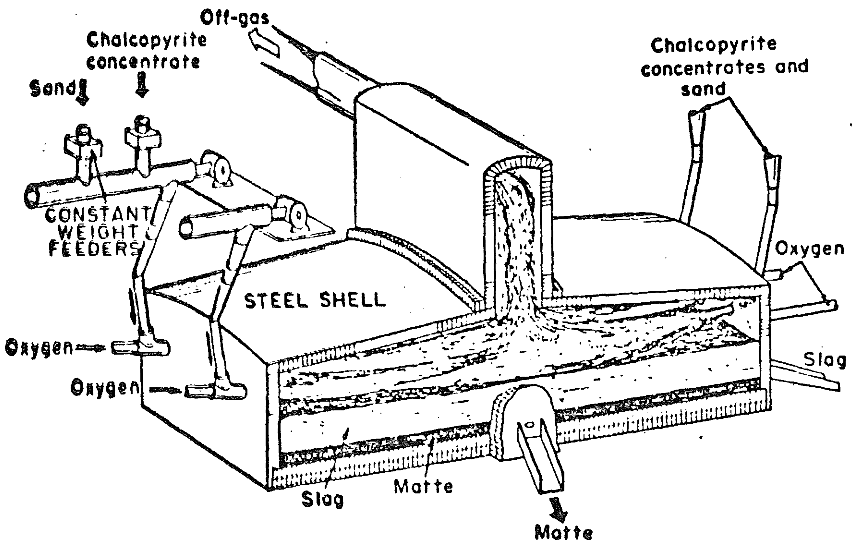
Outokumpu Furnace--Concentrate, dried to less than 0.2 percent moisture is mixed with heated air in a burner at the top of a reaction shaft. The melted droplets collect in the settler and develop the two normal liquid layers—matte and slag. The off gas has a sulfur dioxide content of approximately 14 percent and a dust burden of 6 to 10 percent of the charge. The matte grade is ordinarily 50 to 60 percent in Cu. The slag, which contains too much copper to discard, must be cleaned by settling in an electric furnace, or by casting, slow cooling, and treating by conventional flotation-concentration methods. The high matte grade substantially reduces the amount of converter processing which follows. However, the amount of secondaries which can be smelted as coolant in the converter is limited. The amount of heat available to flash smelting depends upon the iron and sulfur content (Figure 32).[30]

Figure 32. Cutaway View of Outokumpu Preheated Air Flash-Smelting Furnace. (Source: Biswas, 1976)



INCO Furnace--The INCO flash furnace uses commercially-pure oxygen in the flash smelting of sulfide concentrates. The concentrates are dried to approximately 0.1 percent moisture and partially oxidized with the oxygen in burners located in the ends of the furnace. A furnace of modest proportions offers a high throughput. The gas volume is very small, containing approximately 80 percent SO_2 which can be scrubbed and liquified by compression and cooling. The slag developed in the furnace is low enough in value to discard, but the converter slag is returned for cleaning. Thus, slag-cleaning facilities must be included in the smelter design. The oxygen flash furnace is dependent solely upon the oxidation of a portion of the iron and sulfur for the furnace's energy so that its application generally covers the same concentrate grade ranges as the Outokumpu furnace (Figure 33).[30]

Figure 33. Cutaway View of INCO Oxygen Flash Smelting Furnace. (Source: Biswas, 1976)



Flash furnaces utilize the heat evolved from oxidizing part of their sulfide charge to provide much or all of the energy required for smelting. The principal advantage of the flash smelting processes is that their energy costs are considerably lower than those of reverberatory and electric-furnace smelting. Flash furnaces are also excellent from an environmental point of view because they produce SO_2-rich effluent gasses from which the SO_2 can be efficiently removed as sulfuric acid or liquid SO_2. For these reasons, flash furnaces have accounted for most of the new matte smelting capacity since 1965. The principal product of flash-furnace smelting is a high-grade (45 to 50 percent Cu) liquid matte.

Currently (1975), there are twenty Outokumpu flash furnaces in operation (17 built since 1965) and one INCO oxygen flash furnace. Flash furnaces are usually operated singly as the only smelting unit in a smelter.

Continuous Furnaces

Continuous smelting furnaces include the following:

1) The Noranda Furnace
2) The WORCRA Furnace
3) The Mitsubishi Furnace
4) The Britcosmaco Furnace
5) The USBM Furnace

In continuous copper smelting, each metallurgical stage of roasting, smelting, and converting, with slag cleaning, is accomplished in an individual unit with continuous tapping and transfer of matte and metal.

Noranda Furnace--The Noranda process is based on two considerations: by combining smelting and converting in the same furnace, the fuel consumption can be minimized, and dynamic conditions can be created in the bath which will allow the production of copper while charging copper concentrates in the furnace.

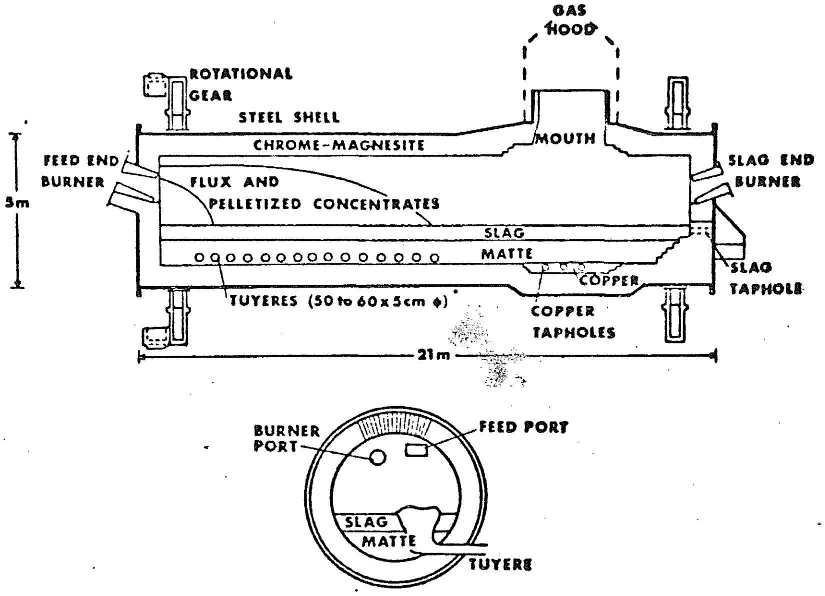
Pelletized concentrates and flux are fed through a slinger belt into a cylindrical furnace of the same diameter but twice the length of a large converter. As oxygen-enriched air is introduced through tuyeres, three liquid layers are produced and coexist in the furnace: slag, matte, and copper.

Matte and slag flows are controlled as they move slowly to the tapping ports. Oxidizing gas is introduced into the matte to oxidize the FeS. Continued injection of the gas into the resulting white metal gradually oxidizes Cu_2S to metallic copper, which is tapped periodically after it separates by settling.

Slag tapped from the reactor is a low-silica, high-copper product, which is treated by milling to yield a high-grade concentrate, which is recycled, and a low-copper (about 0.5 percent) tailing which is discarded (Figure 34).[30]

WORCRA Furnace--In the WORCRA process, concentrate, and flux are introduced in a mildly oxidizing smelting zone by pneumatic or mechanical injection at an appropriate angle to ensure particle penetration into the liquid and to aid the continuous circulation of matte and slag in the "bowl." Some concentrates may be added in the converter zone closer to the slag exit, where they help to control magnetite formation in the slag.

Figure 34. Schematic Longitudinal and End View of the Industrial Noranda Single-Step Reactor. (Source: Biswas, 1976)

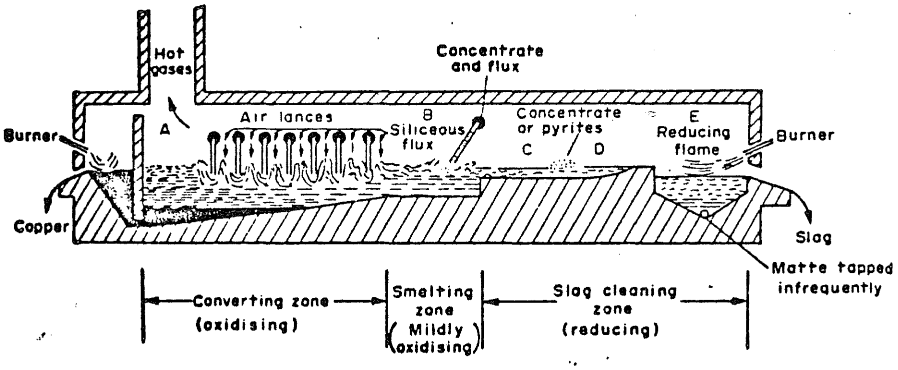


A form of hot solvent-extraction is achieved by forcing the slag to move generally countercurrent to the matte. Unwanted nonvolatile components, particularly iron, are thus continuously being transferred to the slag-after oxidation. Conversely, and particularly in the smelting and slag cleanup zones, valuable copper in the slag can be caused to revert to the matte phase by interaction with ferrous sulfide in the matte. In this separate, but connected, slag-cleaning zone, additions of concentrates or pyrites are made to cause both separation or settling of entrained matte, which is continuously returned by gravity to the smelting zone via a sloping hearth.

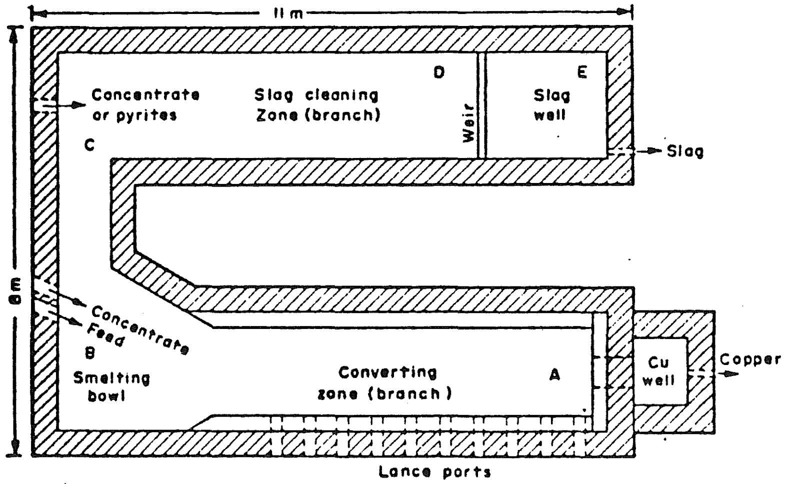
As the matte moves slowly through the smelting and converting zones, it is sequentially lanced with air (or enriched air), causing conversion to white metal and then to copper. Significant differences in specific gravity of these phases aid in the separations. The hearth in the converting zone continues to slope downward to an underpass through which copper passes continuously to a "copper well" which overflows with the blister copper product.

Furnace gases rich in SO_2 can be treated for waste-heat utilization and dust recovery in conventional equipment prior either to venting or to conversion in a sulfuric acid plant. In a commercial-scale plant, the low SO_2 gases generated by the burners that heat the slag-cleaning zone could be used separately for concentrate drying or combined with the rich smelter gas (Figure 35).[30]

Figure 35. Plan and Elevation Views of the WORCRA Process Pilot Reactor. (Source: Biswas, 1976)



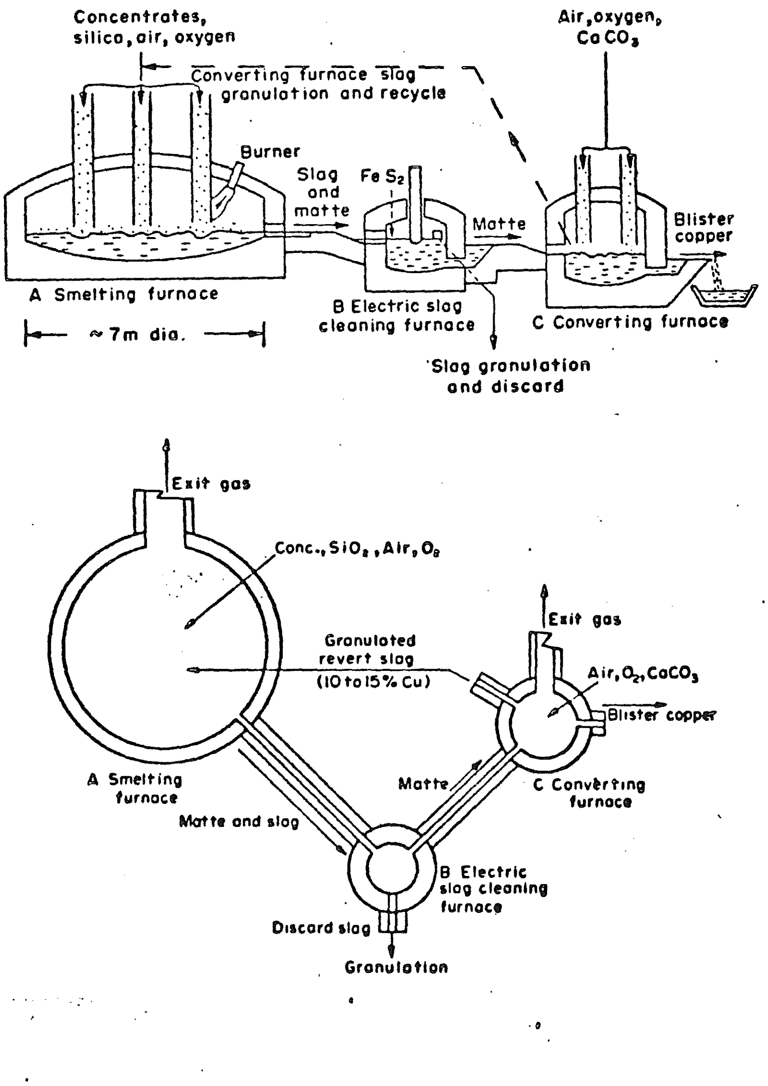
(a) Elevation view†



(b) Plan view

Mitsubishi Furnace--The Mitsubishi process is a relatively recent development and utilizes three separate furnaces. In the first furnace, concentrates are dispersed in air jets which impinge against the surface of the melt, and are smelted and partially converted to approximately a 60 percent copper matte. This matte then flows into a second furnace and is converted by air lances, while the slag is treated pyrometallurgically in an electric furnace. The basic elements of the Mitsubishi process are similar to those of the flash smelting process, with the exception that the three furnaces are interconnected and transfer of slag and matte between them is by gravity flow (Figure 36).[30]

Figure 36. Schematic Plan and Elevation Views of the Mitsubishi Continuous Smelting System. (Source: Biswas, 1976)



Britcosmaco Furnace--This process is intended to bring together the best features currently available in pyrometallurgical methods for the treatment of sulfide concentrates. It is based on minimizing the use of oxygen to not only oxidize sulfur, but also produce a slag of high enough oxygen potential to allow metallic copper to be made directly in the process.

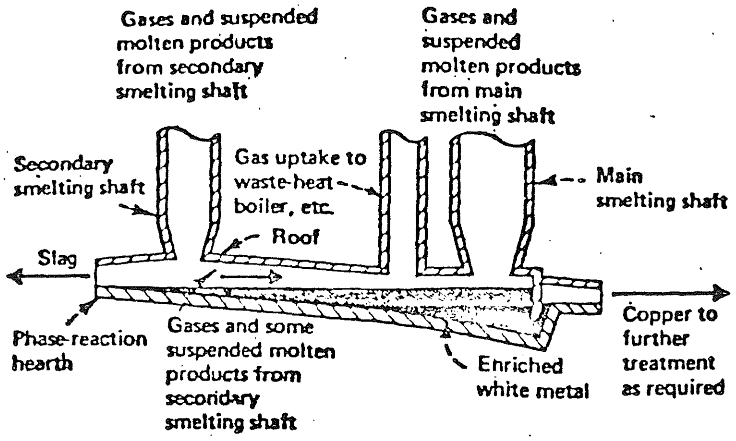
Dry concentrate and flux are fed into the main smelting shaft with sufficient preheated or oxygen-enriched air to provide for autogenous smelting, producing enriched white-metal and slag phases. These collect on the hearth in two layers.

As the volume of slag increases it flows along the phase-reaction section of the hearth. Contact with matte and low-grade matte causes more copper to be rejected. As the slag progresses toward the top hole, it is subjected to even greater reducing action immediately under the secondary smelting shaft. In the final stretch between the shaft and the tap hole, fine particles of matte disperse through the slag, reducing oxygen potential even further and causing additional rejection of copper.

Meanwhile, at the other end of the hearth, copper falling down the main smelting shaft is oxidized up to the state of metal, with variations in oxidation level evening out as the particles work their way through the slag. The copper dissolves at the top of the enriched white-metal layer, is precipitated from the bottom as metal, when it is removed via a bottom-tapping xyphon arrangement (Figure 37).[30]

USBM Furnace--The USBM process consists of a furnace with a sloping hearth. Sulfide concentrate and flux are blown into the furnace through a burner. Flash smelting produces approximately a 50 percent copper matte, which flows down the hearth countercurrent to slag flow.

Figure 37. Britcosmaco Process Needs Low Energy Input.
(Source: Treilhard, 1973)



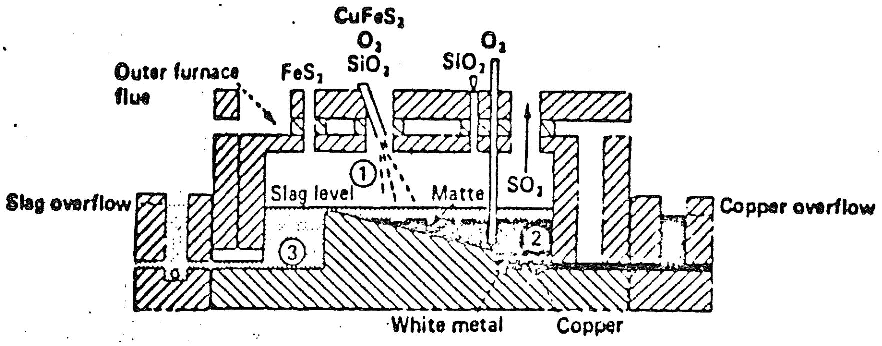
At the converting end of the furnace, oxygen is introduced via a watercooled lance immersed in the matte layer, where conversion to blister copper proceeds. Injection of oxygen into the matte by a lance passing through the slag layer concentrates heat in the matte, where it is needed, and keeps the slag at lower oxygen potential, thus lowering its copper content.

Slag formed in the smelting and converting reactions flow to the slag well, undergoing cleaning enroute by:

1) contact with lower-grade matte
2) the roast reaction of copper sulfide and copper oxide
3) the reduction of copper oxide by sulfur and ferrous sulfide
4) the reduction of magnetite (Figure 38).[30]

All of the continuous smelting processes combine the steps of roasting, smelting, and converting into a continuous (and perhaps autogenous) single-step operation for producing blister copper directly from concentrates. Input materials to the processes are concentrates, fluxes, and air; and the products are blister copper, a gas of high SO_2 strength, and a slag sufficiently low in copper to be directly discarded.

Figure 38. Bumines Autogenous Scheme Combines Multiple Processing in Single Furnace. (Source: Treilhard, 1973)



Verticle elevation section through center

Common reactions by zone

① Flash roasting and smelting

$2CuFeS_2 + O_2 \rightarrow Cu_2S + 2FeS + SO_2$
$2FeS + 3O_2 \rightarrow 2FeO + 2SO_2$
(Fe_2O_3 and Fe_3O_4 also formed)
$3Fe_2O_3 + FeS \rightarrow 7FeO + SO_2$
$3Fe_3O_4 + FeS \rightarrow 10FeO + SO_2$
$2FeO + SiO_2 \rightarrow Fe_2SiO_4$

② Converting

$2FeS + 3O_2 \rightarrow 2FeO + 2SO_2$
$2Cu_2S + 3O_2 \rightarrow 2Cu_2O + 2SO_2$
$Cu_2S + 2Cu_2O \rightarrow 6Cu + SO_2$
$2FeO + SiO_2 \rightarrow Fe_2 \cdot SiO_4$
$6FeO + O_2 \rightarrow 2Fe_3O_4$

③ Slag cleaning

$2Cu_2O + Cu_2S \rightarrow 6Cu + SO_2$
$Cu_2O + FeS \rightarrow Cu_2S + FeO$
$2Cu_2O + S \rightarrow 4Cu + SO_2$
$FeS + 3Fe_3O_4 \rightarrow 10FeO + SO_2$

The potential advantages of such a single-step process for producing blister copper would be:

1) a reduced amount of material handling due to the absence of intermediate steps.

2) a low or zero energy requirement, due to efficient use of the energy obtained by oxidizing sulfides continuously in a single vessel

3) the production of a single stream of high SO_2 strength gas suitable for sulfuric acid or elemental sulfur recovery

4) the ability to apply on-line, automatic computer control to the entire copper-making process

5) low capital cost requirements of a single unit as compared to multiple-unit operations

The Noranda and Mitsubishi processes are operating on an industrial scale and further installations of both are planned. The WORCRA pilot plant was shut down in 1970 and it has not operated since.

Generally speaking, smelting produces SO_2 and volatile oxides of arsenic, antimony, lead, selenium, tellurium, and other trace elements. Depending on the material being processed, and the particular smelting system employed, the dusts emitted in smelting may have commercial value when collected or may pose pollution problems when emitted to the environment.

The behavior of metals other than copper and iron during smelting is estimated in Table 6. These data serve only as a guide and the precise distribution of minor elements depends upon the smelting conditions and the type of process. The most important points are:

1) Gold, silver, the platinum metals, cobalt, and nickel enter the matte almost completely, these metals are carried forward to the converting operation and they are finally recovered as by-products during the electrorefining of anode copper.

2) Significant quantities of impurities harmful to copper also enter the matte; specifically antimony, arsenic, bismuth, lead, selenium, and tellurium. Some of these are also recovered as by-products during subsequent converting and refining operations.

3) Much of the zinc reports to the slag from which it can be recovered by "slag fuming" (reduction) if it is present in sufficient quantities.

Table 6. Estimated Distribution of Elements Other Than Copper and Iron During Matte Smelting. (Source: Biswas, 1976)

	Distribution		
Metal	Matte	Slag	Volatilized*
Alkali and alkaline earth metals, aluminum, titanium		100	
Ag, Au, platinum metals	99	1	
Antimony	30	55	15
Arsenic	35	55	10
Bismuth	10	10	80
Cadmium	60	10	30
Cobalt	95	5	
Lead	30	10	60
Nickel	98	2	
Selenium	40		60
Tellurium	40		60
Tin	10	50	40
Zinc	40	50	10

*Not including solids blown from the furnace (dust losses).

CONVERTING

The process of converting is essentially an oxidation process, no extraneous fuel being utilized. Air for oxidation is admitted a foot or so above tuyeres, so when the converter is in the normal operating position, the air is forced by pressure from the blower through the matte.

The conventional-type converter lies in a horizontal plane and is capable of being rotated on its long axis for purposes of charging and emptying. Liquid matte from the smelting phase is charged to the converter together with silica flux. In the case of a copper rich matte, oxidation commences and slag begins to form, the oxidation of the iron producing the heat necessary to maintain the slagging action.

$$2FeS + 3O_2 = 2FeO + 2SO_2 + 224{,}000 \text{ calories}$$

The iron oxide produced combines with the silica flux to form an iron silicate slag which is poured off, more matte and flux added, air readmitted, and the action commences.

$$2FeO + SiO_2 = 2FeO{\cdot}SiO_2 \text{ (slag)}$$

When sufficient copper sulfide has been accumulated in the converter, this in turn is oxidized, the product being blister copper (98 to 99 percent Cu), which is transferred to furnaces for casting into anodes prior to electrolytic refining.

$$\begin{array}{l} 2Cu_2S + 3O_2 = 2Cu_2O + 2SO_2 \\ \underline{Cu_2S + 2CU_2O = 6Cu + SO_2} \\ 3Cu_2S + 3O_2 = 6Cu + 3SO_2 \end{array}$$

In the case of a copper-nickel sulfide matte, separation of the non-magnetic copper-nickel sulfides is generally affected by flotation, the copper sulfide being conveyed to the copper smelter, the nickel sulfide being sintered for the production of nickel oxide.

Typical types of converters include:

1) The Pierce-Smith Sideblown Converter
2) The Hoboken Horizontal Siphon-Type Converter
3) The Kennecott Converter-Smelter Vessel

Pierce-Smith Sideblown Converter

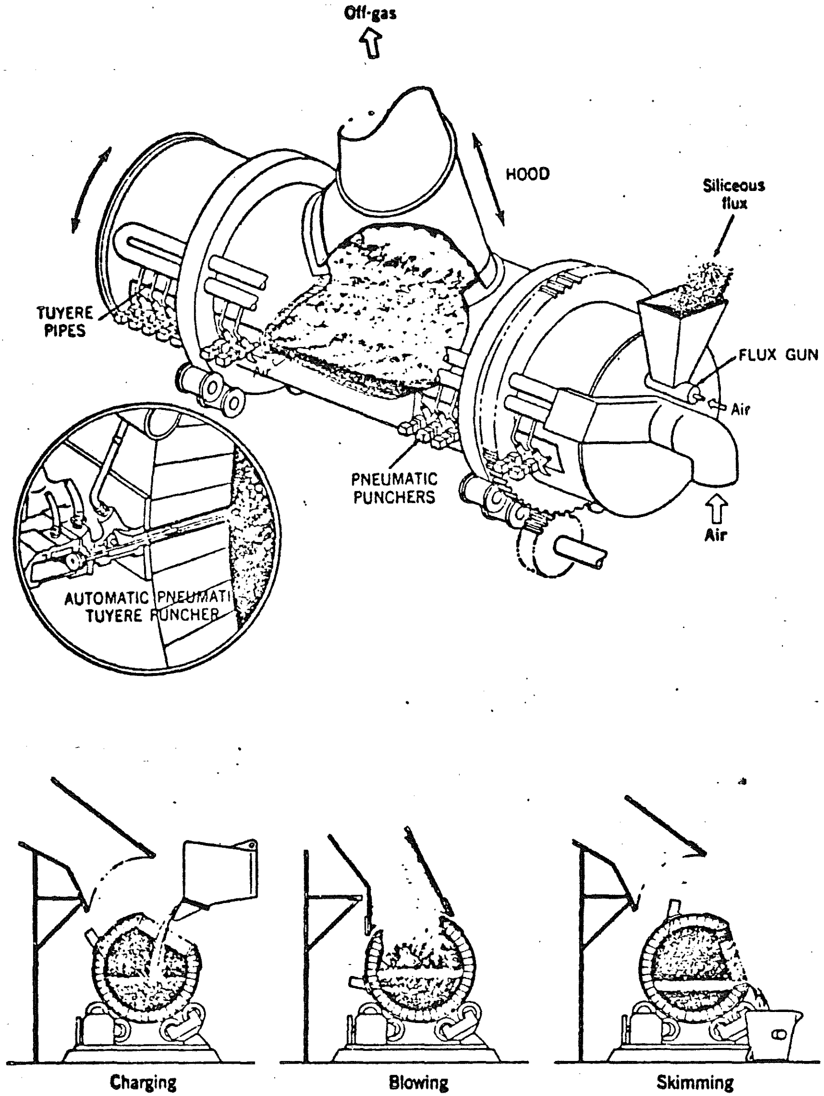
The Pierce-Smith Converter is an efficient machine whose high air-flows allow both large copper throughputs and the smelting of bulky copper-bearing materials and scrap that might otherwise be difficult to dispose of. Outlet gases are generally low in SO_2 concentration, a consequence of excessive air infiltration into the off-take hood over the converter mouth (Figure 39).[30]

Hoboken Horizontal Siphon-Type Converter

In the Hoboken Converter, reaction gas is taken off the vessel constantly through one end, and matte charged to the converter through the small pouring-mouth while the converter is in the upright blowing position. Dilution of the exhaust gas is minimized by controlling the converting draft to give zero suction at the converting mouth.

When the converter is being poured or skimmed of slag, blowing air is discontinued and the weak off-gases are vented to an escape stack rather than to the acid plant. Significantly, an average of eight percent SO_2 gas is consistently obtained during converting operations, and three or

Figure 39. Cutaway View of a Horizontal Side-Blown Pierce Smith Converter for Producing Blister Copper from Matte. (Source: Biswas, 1976)



Charging

Blowing

Skimming

more Hoboken converters are operating on a continuous basis, it should be possible to supply an acid plant with a mixed exhaust gas containing all of the converter off-gases while avoiding excessive fluctuations in gas volume and SO_2 concentration (Figure 40).[30]

Kennecott Converter-Smelter Vessel

This unit is based on the modification of existing converters to allow them to double as smelting and converting vessels. Pelletized concentrates and flux are fed through the mouth of the converter and are smelted and converted to high-grade matte through the use of oxy-fuel burners and oxygen-enriched tuyere air.

In the KCS process a high bath of high-grade matte is maintained in the vessel at all times; slag is skimmed periodically and treated by milling; and at hourly intervals excess high-grade matte is transferred to a finishing converter for blowing to copper. The process may provide a low-capital alternative for existing smelters which for some reason do not want to continue operating their reverberatories (Figure 41).[29]

Figure 40. Hoboken Converter Can Improve Sulfur Recovery but Lacks Pierce-Smith Unit's Smelting Efficiency. (Source: Treilhard, 1973)

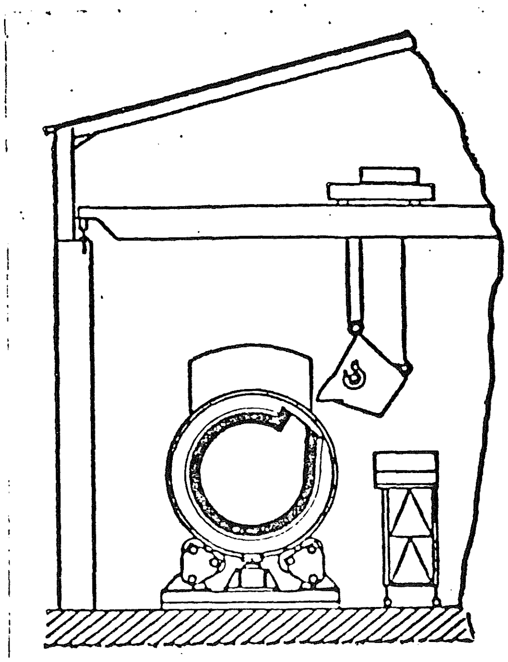
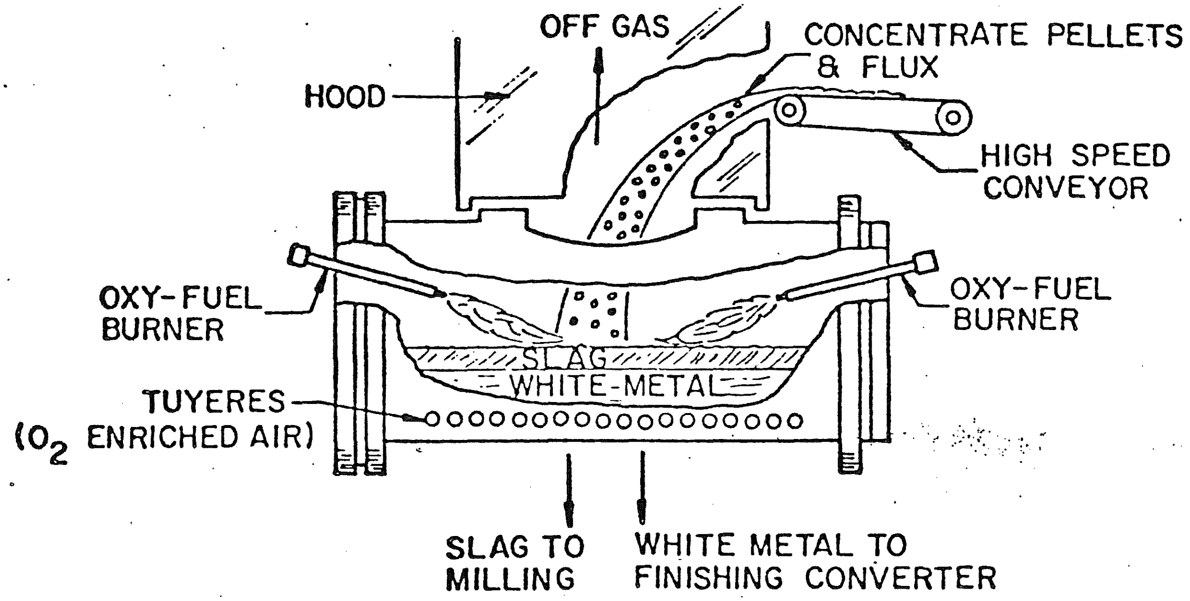


Figure 41. Kennecott's Converter Process Offers an Attractive Low-Cost Alternative for Existing Smelters. (Source: Themelis, 1976)



In converting, the sulfur from the metallic sulfides is driven off as a SO_2 gas and the remaining iron in the system is oxidized. The iron silicate slag which forms is poured off leaving the nearly pure metal or metals in a liquid form. Refining by electrolytic or other means reduces the metal or metals to the required purity.

The converting of copper matte is almost universally carried out in the cylindrical Pierce-Smith type converter. The converting reactions are exothermic and the process is autogenous. Typical converters treat 300 to 400 metric tons of matte per day to produce 100 to 200 metric tons of copper. A smelter will normally have from three to six converters (with one or more of these in the process of being relined or on standby), depending upon the capacity of the smelting furnace.

Converters emit principally SO_2 as a potential gaseous pollutant. Dust loads in converter gases (also considered as a pollutant) may amount to 10 to 20 tons/day/unit. Seventy-five to 85 percent of the solids generated settle in the flue system. The remaining 15 to 25 percent is composed of smaller particles, and for the most part is removed in dust collectors. The dust content depends principally on the chemical composition of the copper matte. An increase in the operating temperature of the converter causes higher volatilization of the metals and, hence, higher dust content in the offgases.

The distribution of elements among blister copper, vapor, and slag is shown in Table 7. This table indicates that, generally speaking, As, Bi, Cd, Ge, Hg, Pb, Sb, and Sn are extensively removed as vapors during converting while most of the Zn is removed along with the iron in the slag. The precious metals, nickel, and cobalt are carried over with the blister copper and are recovered during electrorefining.

Table 7. Estimated Distribution (In Percent) of Impurity Elements During Converting. (Source: Biswas, 1976)

	Blister copper	Vapour[a]	Slag[b]
Ag	90	0	10
Au	90	0	10
Pt metals	90	0	10
As	15	75	10
Bi	5	95	0
Cd	0	80	20
Co	80	0	20
Fe	0	0	100
Ge	0	100	0
Hg	10	90	0
Ni	75	0	25
Pb	5	85	10
Sb	20	60	20
Se	60	10	30
Sn	10	65	25
Te	60	10	30
Zn	0	30	70

[a] Not including ejected droplets of matte and slag.
[b] Including entrained matte.

Converter gases often have entrained droplets of matte and slag which have the approximate composition of their respective liquids at the time of ejection. The solidified droplets are caught in the flue system (generally in the balloon flues and the electrostatic precipitators) and are recycled to the converters. The dusts and vapors from the converters are of sufficient concentration (contain sufficient copper) to be recycled to the smelting furnace or to the converter itself.

POLLUTION AND POLLUTION CONTROL

All of the primary steps in the metallurgical processing of nonferrous minerals emit pollution to the atmosphere either in a gaseous form or in a solid form. The absolute quantity emitted during a prescribed period of time is a direct function of the selection (including the type and the quantity) of auxiliary anti-pollution devices which are capable of minimizing emissions to the atmosphere.

The following broad categories summarize the sources of man-made atmospheric air pollutants:

- transportation
- domestic heating
- electric power generation
- refuse burning
- industrial fuel burning and process emissions

Of the major sources of man-made air pollution, approximately 20 percent is attributable to industrial processes, 30 to 35 percent when pollution from power generating facilities is included. Major industrial polluters are classified and categorized in Table 8 with estimated annual emission levels.

Pollution control in the nonferrous metals industry (which includes copper, lead, zinc, and aluminum) is principally confined to air pollution, water pollution being of secondary consequence. Specific air pollutants include:

- particulate effluents to the atmosphere
- minor constituents and/or volatile fumes
- sulfur dioxide emissions

Particulate Matter

Particulate pollution control must deal basically with two constituents: dust particles micron-size and larger, and metallic fumes (metals or

Table 8. Major Sources of Industrial Pollution. (Source: Ross, 1972)

INDUSTRY	TYPE OF POLLUTION Particulates	Sulfur Oxides	Carbon Monoxide	Fluorides	Hydrocarbons	TOTAL ANNUAL AMOUN (Metric tons)
Petroleum Refining	*	*	*		*	3,809,500
Smelters Aluminum Copper Lead Zinc	*	*				3,764,200
Iron Foundaries	*		*			3,356,000
Kraft, Pulp, and Paper Mills	*	*	*			2,993,000
Coal Cleaning & Refuse	*	*	*			2,131,500
Coke (used in steel manufacturing)	*	*	*			1,995,500
Iron & Steel Mills	*		*			1,632,650
Grain Mills & Grain Handling	*					997,700
Cement Manufacturing	*					771,000
Phosphate Fertilizer Plants	*			*		283,000

metallic compounds that have been volatilized at high furnace temperatures, condensed at lower temperatures, and carried by furnace gases into flues). The coarser dust is easily removed; however, fine dust and metallic fumes are more difficult to treat due to their size range (particles of fume are very fine and do not settle easily, rather have the tendency to pass through the stack with the stack gases). The existance of metallic fumes is characteristic of the nonferrous metals industry. Table 9 summarizes particulate emissions for the primary nonferrous metals industries.

The chemical and physical properties of particulate effluents from primary copper smelters are summarized in Table 10. Particulate emissions from the furnaces are predominantly metallic fumes of submicron size. The fumes are difficult to wet and readily agglomerate. In addition, they are cohesive and will bridge and arch in hoppers and other collection lines.

	Quantity of Material	Emission Factor	Efficiency of Control, C_c	Application of Control, C_t	Net Control, $C_c \cdot C_t$	Emission tons/yr.
I. Aluminum						
A. Preparation of Alumina						
1. Grinding of Bauxite	13,000,000 tons bauxite	6 lb./ton bauxite	-	-	0.80	8,000
2. Calcining of Hydroxide	5,840,000 tons alumina	200 lb./ton alumina	-	-	0.90	58,000
B. Aluminum Mills						
1. Soderberg Cells						
(a) Horizontal stud	800,000 tons aluminum	144 lb./ton aluminum	0.40	1.0	0.40	35,000
(b) Vertical stud	700,000 tons aluminum	84 lb./ton aluminum	0.64	1.0	0.64	10,000
2. Prebake Cells	1,755,000 tons aluminum	63 lb./ton aluminum	0.64	1.0	0.64	20,000
C. Materials Handling	3,300,000 tons aluminum	10 lb./ton aluminum	0.90	0.35	0.32	11,000
			Total from primary aluminum industry			142,000
II. Copper						
A. Ore Crushing	170,000,000 tons ore	2 lb./ton ore	0.0	0.0	0.0	170,000
B. Roasting	40% of 1,437,000 tons copper	168 lb./ton copper	0.85	1.0	0.85	7,000
C. Reverberatory Furnace	1,437,000 tons copper	206 lb./ton copper	0.95	0.85	0.81	28,000
D. Converter	1,437,000 tons copper	235 lb./ton copper	0.95	0.85	0.81	33,000
E. Fire Refining						
F. Slag Furnaces						
G. Materials Handling: Ore, Limestone, Slag, etc.	1,437,000 tons copper	10 lb./ton copper	0.90	0.35	0.32	5,000
			Total from primary copper industry			243,000
III. Zinc						
A. Ore Crushing	18,000,000 tons ore	2 lb./ton ore	0.0	0.0	0.0	18,000
B. Roasting						
1. Fluidized Bed, Suspension	75% of 1,020,000 tons zinc	2,000 lb./ton zinc	0.98	1.0	0.98	15,000
2. Ropp, Multiple Hearth	15% of 1,020,000 tons zinc	333 lb./ton zinc	0.85	1.0	0.85	4,000
C. Sintering and Sinter Crushing	60% of 1,020,000 tons zinc	180 lb./ton zinc	0.95	1.0	0.95	3,000
D. Distillation	60% of 1,020,000 tons zinc		0.0	0.0	0.0	15,000
E. Materials Handling	1,020,000 tons zinc	7 lb./ton zinc	0.90	0.35	0.32	2,000
			Total from primary zinc industry			57,000
Lead						
A. Ore Crushing	4,500,000 tons ore	2 lb./ton ore	0.0	0.0	0.0	4,000
B. Sintering	467,000 tons lead	520 lb./ton lead	0.95	0.90	0.86	17,000
C. Blast Furnace	467,000 tons lead	250 lb./ton lead	0.85	0.98	0.83	10,000
D. Drossing Kettle		-				
E. Softening Furnace		-				
F. Desilvering Kettles		-				
G. Cupeling Furnaces		-				
H. Refining Kettles		-				
I. Dross Reverberatory Furnace	467,000 tons lead	20 lb./ton lead	-	-	0.50	2,000
J. Materials Handling	467,000 tons lead	5 lb./ton lead	0.90	0.35	0.32	1,000
			Total from primary lead industry			34,000
			Total from nonferrous metals industries			476,000

Table 9. Particulate Emissions-Primary Nonferrous Metal Industries. (Source: Jones, 1972)

Particulate

Source	Particle Size	Solids Loading	Chemical Composition
Roasting furnace	15<10	6 - 24	Cu: 9, S: 10, Fe: 26
Blast furnace	--	6.6	Cu: 4.4, Zn: 12.5, S: 7.3
Reverberatory furnace	51<37	1 - 6	Cu: 6.2, Zn: 13.0, S: 13.6 Oxides of arsenic, antimony, aluminum silicon sulfates (traces).
Copper matte converter	--	5.3	Cu: 1.2, Zn: 18.0, S: 10 (also see reverberatory furnace)
Copper refining (coal dust fired)	50<44	--	Cu: 7, S: 1.5

Coding Key for Tables of Effluent Characteristics

I. General Note: All data for uncontrolled sources unless otherwise noted.

II. Specific Notes:

A. Particulates

1. Particle size:
   x < y, x > y.
   x = weight percent, y = particle size (μ).
   Measuring technique noted. If no notation is listed, measuring technique was not reported or is unknown.
2. Solids loading: grains/scf, unless otherwise noted.
3. Chemical Composition: solids - weight percent (unless otherwise noted).

Table 10. Effluent Characteristics-Primary Copper Production. (Source: Jones, 1972)

Technology has been developed to provide equipment which, when operated efficiently, minimizes the negative effects inherent in a particular metallurgical system. Pollutants must be efficiently recovered in a form which is either marketable or inert under atmospheric conditions for regulated disposal.

The selection of suitable control devices requires careful consideration of the characteristics of the pollutants and the equipment capabilities. Poor system performance in many cases is indicative of poor equipment selection.

Control equipment is often classified into two categories:

- equipment capable of removing particulates
- equipment capable of removing gaseous pollutants

Equipment available for each of these two broad categories include:

Particulates
(Including Aerosols)

settling chambers
cyclones
impingement separators
centrifugal separators
bag filters
wet collectors
electrostatic precipitators

Gaseous Pollutants

gas scrubbers
absorbers
incinerators
  direct incineration
  catalytic combustion
condensors

Particulate Matter Removal[16,17]

The evaluation of a pollution problem followed by the design of a control system to meet the particular need is a long and involved process. A well designed program involving pollution evaluation, engineering study, and engineering construction is necessary in all situations where a pollution problem is evident. This means that all potential anti-pollution devices are studied with respect to a particular situation.

There are certain fundamentals which are applicable to all practical collectors of particulate matter. These fundamentals are:

1) All collectors clean the gas by continuously removing the dust from a moving stream of gas.

2) The size of collector and its cost are directly proportional to the gas quantity to be cleaned.

3) The collection efficiency is calculated by the following formula (expressed as a percentage)

$$\% \text{ collection efficiency} = \frac{\text{wt. of input dust} - \text{wt. of output dust}}{\text{wt. of input dust}} \times 100$$

4) Fine dust is considerably harder to collect than coarse dust in all types of particulate collectors. Many dusts are a collection of fine and coarse with the proportion of fines dictating the collector efficiency. Heavier concentrations of dust are usually easier to collect than low-grain loadings.

Table 11 summarizes the basic types and subtypes of particulate collectors.

Table 12 summarizes various characteristics of the control equipment described in Table 11. Included are the efficiency factors attainable by each class of equipment. Efficiency values are based on many factors, some of which directly correlate with the parameters and characteristics of the original ore concentrate being treated.

Table 11. Fundamental Operating Characteristics of Commercial Particulate Collectors. (Source: Ross, 1972)

Basic Type	Specific Type	Basic Operating Force	Basic Measure and Unit of Capacity	Typical Capacity in cfm of Gas per Basic Measure of Capacity	Typical Required Plan Area for Collector to Clean 100,000 cfm[e]	Typical Energy Required and % Collection Efficiency on Fine Dust[c]		COST IN $1000 FOR A COLLECTOR TO CLEAN 100,000 CFM: Cost to Purchase fob Shop and Install		COST IN $1000 FOR A COLLECTOR TO CLEAN 100,000 CFM: Annual Cost for Maintenance and Power		Effect On Collection of Operation at Over or Under Capacity[d]	
					sq. ft.	Inches w.g.	%	fob shop[a]	install	maintenance	power[b]	over	under
Mechanical Collectors	settling chamber	gravity	casing volume (cu ft)	20	2600	0.2	25	10	3	1	0.5	down	up
	baffle	intertia	inlet area (sq ft)	1200–3600	300	0.5	40	13	4	2	1.3	up	down
	high-efficiency cyclones	centrifugal	inlet area (sq ft)	3000–3600	125	4	80	20	5	2	10	up	down
Fabric Filters	manual cleaning	filter cake filtration	fabric area (sq ft)	1–4	1000	4	99	35	20	7	10	down	no change
	automatic shaker cleaning	filter cake filtration	fabric area (sq ft)	1–4	1000	5	99	60	30	10	13	down	no change
	automatic reverse jet cleaning	felt fabric filtration	fabric area (sq ft)	3–8	600	8	99	80	35	15	20	no change	no change
Wet Scrubbers	impingement baffle	liquid capture	baffle cross-sectional area (sq ft)	500–600	300	4	80	30	15	7	10	up	down
	packed tower	liquid capture	bed cross-sectional area (sq ft)	500–700	250	7	90	40	20	9	17	down	up
	venturi	liquid capture	throat area (sq ft)	7,000 to 30,000	100	30	99	50	50	11	75	up	down
Electric Precipitators	single field	electric	collectrode area (sq ft)	5	270	0.5	95	75	55	4	1.3	down	up
	multiple field	electric	collectrode area (sq ft)	3	500	0.5	99	100	70	5	1.3	down	up

[a] Fob shop costs, installation and maintenance costs are based on 1970 mild steel construction costs, and do not include auxiliary equipment such as supporting structure, connecting flues, thermal insulation, foundations, stacks, etc. They are subject to substantial variation due to the specific requirements of each installation including geographic location. Use only for general comparison purposes.

[b] Energy costs are based on electric power at $0.015/kwh.

[c] Fine dust is considered to be 70% by weight minus 10 μ.

[d] Some collectors can be modified to maintain collection efficiency at over or under capacity.

[e] Plan areas do not include connecting flues.

								REQUIRED UTILITIES							
Control Equipment	*Pollutant*	*Optimum[a] Size Particle (microns)*	*Optimum Concentration (grains/cu ft)*	*Temperature Limitations, (°F)*	*Approximate Pressure Drop (in. w.g.)*	*Efficiency*	*Space[b] Requirements*	*Power*	*Water*	*Steam*	*Fuel*	*Solid Waste*	*Liquid Waste*	*Collected Pollutant*	*Remarks*
Mechanical collectors:	Aerosol														
Settling chamber		> 50	> 5	700	< 0.1	< 50	L	X				X		dry dust	good as precleaner
Cyclone		5–25	> 1	700	1–5	50–90	M	X				X		dry dust	low initial cost
Dynamic precipitator		> 10	> 1	700	Fan	< 80	M	X				X		dry dust	
Impingement separator		> 10	> 1	700	< 4	> 80	S	X				X		dry dust	
Bag filter		< 1	> 0.1	500	> 4	> 99	L	X				X		dry dust	bags sensitive to humidity, filter velocity & temperature
Wet collectors:															
Spray tower		25	> 1	40–700[f]	0.5	< 80	L	X	X				X	liquid	1. waste treatment required
Cyclonic		5	> 1	40–700[f]	> 2	< 80	L	X	X				X	liquid	2. visible plume possible
Impingement		5	> 1	40–700[f]	> 2	< 80	L	X	X				X	liquid	3. corrosion
Venturi		< 1	> 0.1	40–700[f]	1–60	< 99	S	X	X				X	liquid	4. high-temperature operation possible
Electrostatic precipitator		< 1	> 0.1	1000	< 1	95–99	L	X				X		dry or wet dust	sensitive to varying conditions & particle properties
Gas scrubber	Gaseous		[c]	40–100	< 10	> 90	M-L	X	X				X	liquid	same as wet collector
Gas adsorber			[d]	40–100	< 10	> 97	L	X		X			X	solid or liquid	adsorbent life critical
Direct incinerator			combustible vapors	2000	< 1	> 98	M	X			X			none	operating costs prime consideration
Catalytic combustion				1400	> 1	> 98	L	X			X			none	contaminants could poison catalyst

[a] Minimum particle size collected at approximately 90% efficiency under usual operating conditions.
[b] Space requirements: S, small; M, moderate; L, large.
[c] Dependent on solubility of pollutant.
[d] Adsorber: concentrations less 2 ppm non regenerative system; greater than 2 ppm regenerative system.
[e] See Chapters 8 and 9 for specific requirements.
[f] Limited by materials of construction when sprays not in operation.

Table 12. Control Equipment Characteristics. (Source: Ross, 1972)

Mechanical Collectors--Mechanical collectors are designed to take advantage of differences in specific gravity between typical industrial dusts and the gases in which they are entrained, separating the heavier dust from the lighter gas. Refer to Table 13 for a summary of the types of equipment which fall into this category.

TYPE	CHARACTERISTICS	UNIQUE DISADVANTAGES
Gravity settling chamber	Physical Size-very large Installed Cost-low Energy Cost-very low Maintenance Cost- low Collection Efficiency-very low Reliability-excellent Efficiency At Low-increases loads Efficiency At -decreases overloads	Low collection efficiencies on fine and moderately fine dusts
Recirculating baffle collector	Physical Size-small Installed Cost-low Energy Cost-low Collection Efficiency-low Reliability-excellent Efficiency At Low-lowers slowly loads Efficiency At -increases slightly overloads	Collection efficiency not sufficient to meet current emission control requirements
High Efficiency Cyclones		Fractional efficiency (efficiency drops off rapidly below a certain size of dust) Draft losses (proportional to the square of the inlet velocity)
Chip trap or cinder tray trap louvre separator Dynamic Precipitator Centrifugal Inertial Separator Ordinary Cyclone Wet Cyclone Cyclonic Drouplet Collector Ultra High-Efficiency Collector		

Table 13. Mechanical Collectors.

Fabric Filters--Fabric filters are versatile collectors used in the removal of dry, particulate matter from a gas stream. Dust bearing gas is passed unidirectionally through a fabric filter medium of woven or felted cloth. The principal function of the fabric filter includes the trapping of dust on the dirty-gas side of the fabric while allowing the gas to pass through the interstices between the woven threads of the fabric. Interstices as large as 100μ in typical commercial bag filters (woven fabric media) trap dust particles as small as 0.5μ.

The capability of the woven fabric filter to collect fine dust is a direct consequence of the buildup of a fragile, porous layer of dust on the dirty-gas side of the cloth. This layer, better known as the filter cake, blocks the larger interstices preventing the finer dust particles from passing through. The thicker felted fabrics are less dependent on the formation of the filter cake. They, however, cannot be cleaned effectively by shaking and must be cleaned frequently with high-pressure reverse flow air jets.

The mechanisms responsible for the formation of filter cake include:

- agglomeration of upstream particles
- impaction of larger particles on the fiber
- electrostatic attraction and repulsion
- thermal effects
- coarse sieving by cloth
- fine sieving by filter cake layer

Table 14 shows a typical operating cycle of one compartment of compartmented, automatic fabric filter. Dust collection efficiency is also indicated.

The major advantage of the fabric filter is its relatively high efficiency at all particle loads from zero to maximum gas flow. Disadvantages include large size and high maintenance due to the cost of bag replacement. Additional problems result from the exposure of the bags to elevated temperatures causing destruction, or exposure to moisture resulting from operation below the dew point causing blinding or plugging.

Cycle Number	Filter Cycle	Cloth Status	Filter Cake Thickness	Air-to-Cloth Ratio or Filtering Velocity (ft/min)	Draft Loss (in./W.G.)	Collection Efficiency
1	new	clean	zero	2–3	< 0.5	no dust
2	precoat	clean to coated	zero to minimum	2–3	< 0.5 to 2–3	low to high
3	filtering	coated	partial	2–3	3–5	high
4	filtering	full coated	maximum	2–3	4–6	high
5	cleaning	shaking	maximum to minimum	dampered off	Zero	not operating
6	filtering	coated	minimum	2–3	2–3	high
7	filtering	full coated	maximum	2–3	4–6	high

The following comments are keyed to cycle numbers:

(1) When new and clean, the draft loss across the fabric filter media is much below 0.5 in. W.G.

(2) During the precoat cycle the draft loss will gradually increase as the layer of filter cake is built up. The efficiency will increase to operating levels of plus 99 percent; the gas discharge will become clear.

(3–4) During the filtering cycle the draft loss will increase to the maximum acceptable level as the layer of filter cake thickens.

(5) The conventional bag house will damper off the compartment being cleaned. Maximum filter cake is shaken off the cloth and drops into a hopper under the influence of gravity. The vigor and duration of shaking must be adequate to reduce filter cake thickness from maximum to minimum. Excessive shaking may reduce collector efficiency by reducing filter cake thickness below minimum.

(6) Compartment is restored to filtering cycle. Cycles (5), (6), and (7) are repeated for each compartment by a predetermined program so that no more than one compartment is removed from service at one time.

Table 14. Operating Cycle of One Compartment of Conventional Compartmented Bag House With Woven Fabric. (Source: Ross, 1972)

Table 15 summarizes the basic types of fabric filters available for commercial use.

Wet Scrubbers--Wet scrubbers are mechanical type devices which use a liquid, usually water, to capture and then remove particulate matter from a moving stream of gas.

The collection efficiency of all well designed wet scrubbers is related to the total energy expended. Low energy collectors have low efficiency,

Basic Type of Baghouse	Type of Bag	NORMAL DIRECTION OF GAS FLOW DURING FILTERING CYCLE		CLEANING TUBES		Air to Cloth Ratios (ft/min)	Space Require-ment	Cost per Square Foot of Filter Media
		Thru Collector	Thru Tube	Method	Cycle			
Inter-mittent	woven cloth tube	up	inside to outside	manual or powered shaking	periodically during shutdown	1-4	very large	low
Conven-tional	woven cloth tube	up	inside to outside	automatic mechanical or pneumatic shaking	intermittent by isolated compartment	1-4	very large	low
Reverse air flow cleaning	woven cloth tube	up	inside to outside	automatic tube collapse by low pressure reverse flow	intermittent by isolated compartment	1-4	very large	low
Reverse ring jet cleaning	felted cloth tube	down	inside to outside	travelling compressed air ring	continuous on heavy dust loadings or intermittent on light dust loadings while gas flows	3-8	large	high
Reverse pulse jet cleaning	felted cloth tube	up	outside to inside	high pressure pulsed jet	programmed by manifolds while gas flows	3-8	large	high

Table 15. Basic Types of Fabric Filters Using Cloth Tubes. (Source: Ross, 1972)

correspondingly, high energy collectors have high efficiency. The energy is introduced either in the water cycle or the gas cycle. For most commercial collectors, the energy is usually introduced in the gas cycle.

The major advantage of the wet scrubber is the great variety of types, allowing selection of a collector suitable for almost any collection problem. Collection efficiencies range as high as 99 percent.

Primary disadvantages include:

- disposal problems associated with wet sludges
- high energy costs associated with the high efficiency scrubber

- increased material costs where chemical corrosion is evident
- potential problems of plugged nozzles
- unavailability of scrubbing liquids of sufficient clarity
- the presence of a visible plume discharging to the atmosphere

Table 16 summarizes the basic types of scrubbers available in this category.

		WATER VS. GAS FLOW	WATER CIRCULATION	DRAFT LOSS	PERCENT COLLECTION EFFICIENCY ON FINE DUST		
Basic Type	*Specific Type*	*Concurrent or counter or cross*	*gpm per 1,000 cfm*	*Inches water gauge*	*Low*	*Moderate*	*High*
Impingement baffle	tangential inlet wet cyclone	concurrent or cross	3–5	1–4	X	X	
	spiral baffle wet cyclone	concurrent	1–2	4–6	X	X	
	single plate	concurrent	2–4	1–8	X	X	
	multiple plate	concurrent	3–5	6–12		X	
Packed tower	fixed bed	concurrent or counter	10–20	2–4	X		
	fluidized bed	counter	15–30	4–12	X	X	
	flooded bed	concurrent	2–4	4–8		X	
	multiple bed	counter	20–40	4–12		X	
Submerged Orifice	wide slot	concurrent	15–30	2–15	X	X	
	circular slot	concurrent	15–30	2–15	X	X	
	multiple slot	concurrent	15–30	2–15	X	X	
Venturi	high-pressure	cross or concurrent	5–7	30–100			X
	medium-pressure		3–5	10–30		X	X
	low-pressure		2–4	3–10	X	X	
	flooded disc		5–6	30–70		X	X
Miscellaneous and combination scrubbers	cross-flow packed	cross	1–4	2–4	X	X	
	centrifugal fan	concurrent	1–2	*	X	X	
	multiple venturi	concurrent	4–6	20–80		X	X
	combination venturi	concurrent	5–7	15–60		X	X
	combination fan type	concurrent	2–3	*		X	

*½ hp per 1,000 cfm.

Table 16. Basic Types of Wet Scrubbers. (Source: Ross, 1972)

Electrostatic Precipitators--The industrial electric precipitator, invented by Frederic Gardner Cottrell in 1910, separates entrained particulate matter from gas streams by:

- charging the dust to a negative voltage (x 50,000 V)
- precipitating the dust onto grounded collecting electrodes
- dropping the agglomerated dust into a hopper

The energy consumption is low and the draft loss is the lowest of all high-efficiency collectors.

In commercial units, the dirty gas is passed horizontally through narrow, vertical gas passages formed by parallel rows of grounded collecting electrodes. Electrically insulated high-voltage wires are spaced precisely on the center lines of each gas passage, thereby causing the dirty gas to pass between the high-voltage wires and the grounded plates.

Dust particles carried through the gas passages by the entraining gas, collide with negative gas ions and are charged negatively. The charged dust particles move rapidly toward the grounded (positive) plates and are held by powerful electric forces.

The dust particles build a thickening layer on the collectors, and the negative charge gradually bleeds from the dust into the grounded electrode. As the layer thickens, the charges on the recently precipitated dust must be conducted through the layer of that previously precipitated. The resistance of the dust layer to this current flow is termed "dust resistivity." Precipitators are successfully operating on dusts whose resistivity is in the range of 10^7 - 10^{11} Ω - cm.

After a 1/16 - 1/4 inch layer of dust has been precipitated, the dust particles next to the collectrode have lost much of their charge to the grounded plate. The electrode attraction has been weakened. The recently arrived dust particles still hold much of their charge, being electrically insulated from the plate by the dust layer, thereby holding the entire layer against the collectrode. A sharp rap causes the dust layer to shear away from the collectrode. The force of rapping is restrained so as to allow the layer to be shattered into relatively large agglomerations of dust particles, but not back into the original submicron particles.

The relatively large agglomerates fall into the hoppers under the influence of gravity just as in the gravity settling chamber. Most electric precipitators are operated at gas velocities in the range of 3-6 ft/sec to allow ample settling time. The high voltage remains on to recharge and reprecipitate any fine particles which may become re-entrained during the rapping cycle.

The major advantage of the electric precipitator is its high collection efficiency with a minimum of operating cost. Other advantages include:

- minimum energy requirements of all high-efficiency collectors
- reduced maintenance requirements
- high reliability at any required collection efficiency from maximum to zero gas flow

Among the disadvantages are:

- high installation costs
- unpredictable collection efficiency on certain high-resistivity dusts
- loss of collection efficiency at gas flows above the designed rating

Gases

As pointed out in the report "Exhaust Gases From Combustion and Industrial Processes" by Engineering Science Incorporated of Washington, D.C., published as report PB 204, 861 by NTIS, (October 2, 1971), the gaseous emissions from copper smelters consist of sulfur oxides (Jones, 1972).

The chemical and physical properties of gaseous effluents from primary copper smelters are summarized in Table 17.

Sulfur Dioxide Removal [4]

Over 90 percent of the world's primary copper originates in sulfide minerals so that sulfur in some form is a by-product of most copper processing. Furthermore, most of the sulfur is emitted as SO_2 gas.

In the past, and even today, SO_2 has been vented directly into the atmosphere; however, this practice is now prohibited in many areas of the world. The standards for clean air vary from country to country, but representative requirements for controlling copper smelter emissions are (U.S. Environmental Protection Agency):

1) not more than 10 percent of the sulfur entering the smelter can be emitted to the atmosphere
2) the yearly average SO_2 content anywhere at ground level must not exceed 0.03 ppm (by volume)
3) the average SO_2 content anywhere at ground level on any one day must not exceed 0.14 ppm

There has been a tendency to build increasingly taller chimney stacks (380 m, INCO, Sudbury) which might possibly meet criteria 2 and 3, but these will not satisfy requirement 1.

Carrier Gas

Source	Flow Rate	Temperature	Moisture Content	Chemical Composition
Roasting furnace	a. 60 - 131 b. 47.3	600 - 890	--	SO_2: 8
Blast furnace	a. 21.2 b. 76.5	248	--	CO_2: 6.5
Reverberatory furnace	a. 50 - 460 b. 71	350 - 750	4 - 10	O_2: 5 - 6, CO_2: 10 - 17, N_2: 72 - 76, CO: 0 - 0.2, SO_2: 1 - 2
Copper matte converter	a. 102 - 750 b. 364	300 - 650	--	SO_2: 4
Copper refining furnace (coal dust fired)	a. 16	410	--	--

B. Carrier Gas

1. Flow rate: flow-rate data presented in two forms:
    a) thousands of standard cubic feet per minute, M scfm, unless otherwise noted.
    b) thousands of standard cubic feet per ton of product processed, M scf/ton, unless otherwise noted.
2. Temperature: °F.
3. Moisture content: volume percent, unless otherwise noted. Dew point is in °F. if listed under moisture content.
4. Chemical composition: volume percent, unless otherwise noted.

Table 17. Effluent Characteristics-Primary Copper Production. (Source: Jones, 1972)

The copper smelter is faced with two major problems:

1) how to capture most of the SO_2 and fix the sulfur in a useful or stable form (e.g. elemental sulfur, liquid SO_2, or sulfuric acid)
2) what to do with the product resulting from 1.

Three basic techniques are used to fix SO_2 (and other oxy-sulfur compounds):

1) reduction to elemental sulfur
2) manufacture of sulfuric acid (or liquid SO_2)
3) direct manufacture of ammonium sulfate (fertilizer) from ammoniacal leach solutions

All of these products are suitable for use in industry or agriculture, but only solid elemental sulfur is suitable for long-term inexpensive storage, possibly in worked-out quarries or mines. It has been suggested that SO_2 gases or sulfate solutions might be treated with limestone or dolomite to form discardable $CaSO_4$ or $MgSO_4$, but these products are of limited usefulness and their long-term stability is open to question.

Process selection is determined by:

- economic conditions for sulfur by-product disposal
- variation in the nature of the offgases
    - sulfur dioxide concentration
    - impurities carried over from smelting operations
    - fluctuations in gas flow rates
- geographic locations
- nearness of markets for sulfurous products
- availability of land for product disposal
- water pollution problems

Contact Sulfuric Acid Process--The contact sulfuric acid process is a well-established chemical process for removing sulfur dioxide from smelter gases.

Smelter gases are first cleaned of particulates, sulfur trioxide removed in a mist cattrell precipitator, the remaining gas stream dried with strong sulfuric acid. The sulfur dioxide in the dry gas is then oxidized to sulfur trioxide in a catalytic converter. The sulfur trioxide is absorbed in strong (98 percent) sulfuric acid to yield the product of the plant. The tail gas is treated to remove droplets of acid and normally is vented to the atmosphere. Tail gases will contain from 0.2 to 0.5 percent SO_2 (Figure 42).[8]

Cominco Absorption Process--This process is an ammonium sulfite-bisulfite absorption process which involves chemical regeneration with sulfuric acid to release SO_2 and form ammonium sulfate (Figure 43).

Sulfur dioxide-bearing gas, free of sulfur trioxide and particulates, is absorbed by an ammonium sulfite-ammonium bisulfite solution. The sulfur dioxide in the gas reacts with ammonium sulfite to form the bisulfite. Ammonia is added to convert part of the bisulfite to sulfite which is recycled to the absorption scrubbers. The remaining bisulfite solution is diverted to the stripper, acidified with sulfuric acid, and stripped with air to produce approximately a 25 percent sulfur dioxide gas and a solution of ammonium sulfate containing approximately 10 percent of the feed sulfur. The process will remove 90 percent of the sulfur dioxide from dilute flue gases, even at concentrations as low as 0.5 percent. Tail gases contain as little as 0.03 percent sulfur dioxide.[8]

Dimethylaniline (DMA) Absorption Process--This process, developed by the American Smelting and Refining Company is used for recovering sulfur dioxide from smelter gases containing 4 to 10 percent SO_2 (Figure 44).

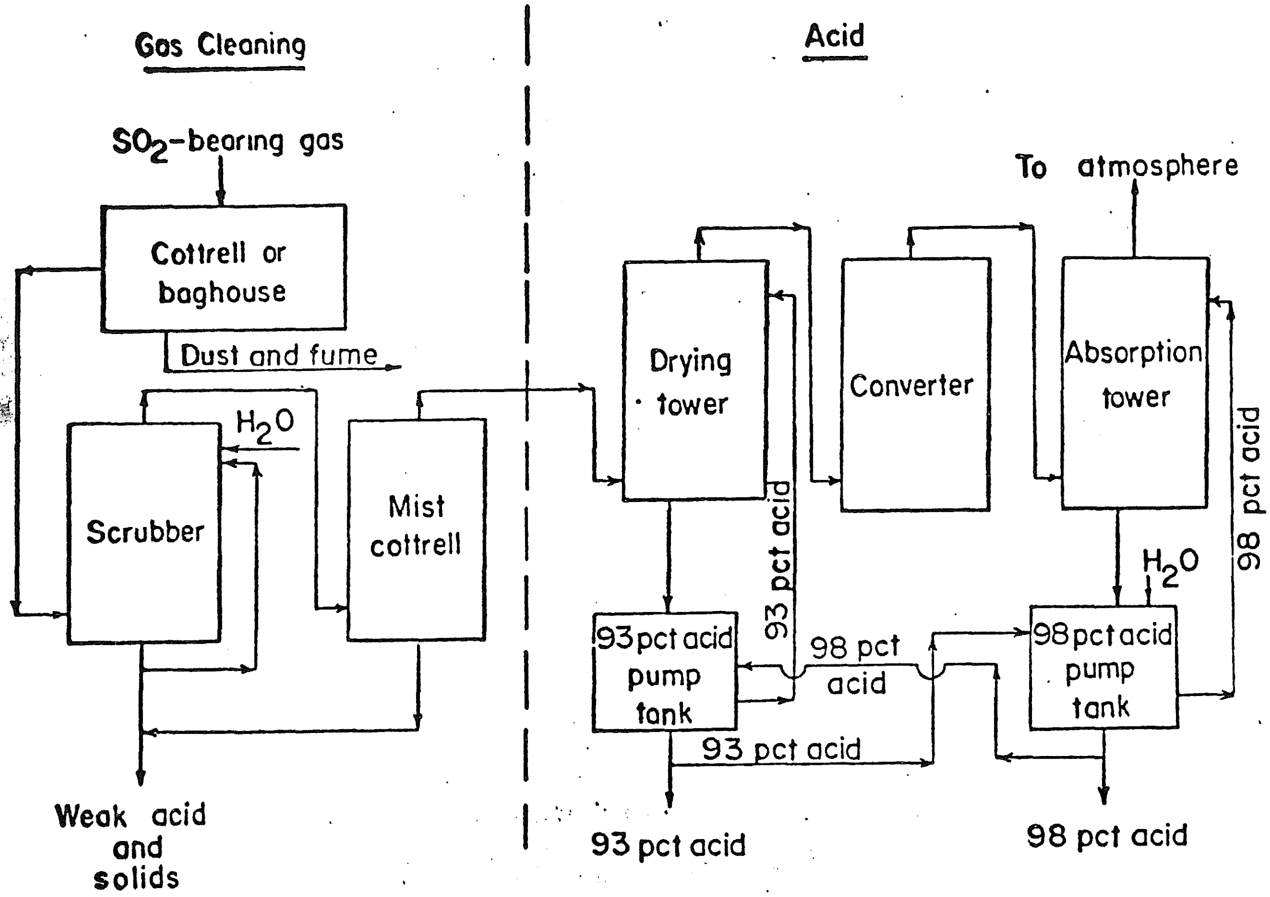


Figure 42. Contact Sulfuric Acid Process. (Source: Jones, 1972)

Comments

The contact sulfuric acid process is the only well established chemical process for removing sulfur dioxide for smelter gases.

Strong gas is a primary consideration for acid production (matter of economics).

The normal economic minimum concentration of sulfur dioxide in acid plant feed gas is 3.5-4.0 percent.

The major factor limiting production of sulfuric acid from smelter gases is the marketability of acid.

Disposal of neutralized acid poses serious problems for some smelters.

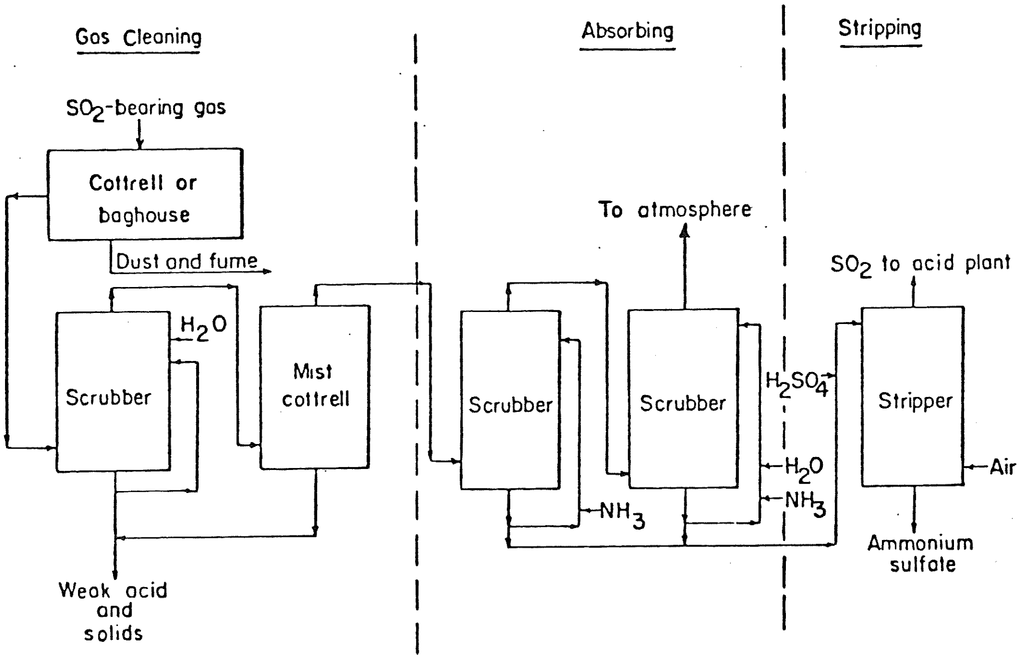


Figure 43. Cominco Absorption Process. (Source: Jones, 1972)

Comments

Developed by the Consolodated Mining and Smelting Company of Canada, Ltd., it has seen only limited application.

The process will recover 90 percent of the sulfur dioxide from dilute flue gases.

A serious disadvantage of process is the high cost of ammonia.

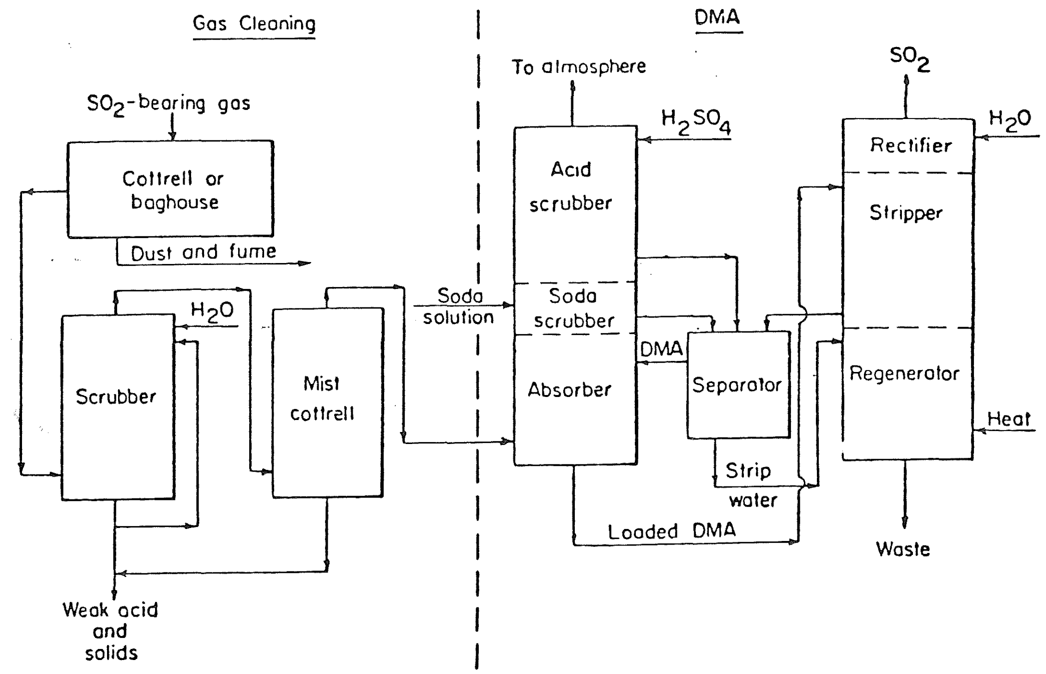


Figure 44. DMA Process. (Source: Jones, 1972)

Comments

Developed by the American Smelting and Refining Company, the process has seen limited use.

The process can recover 90-95 percent of the sulfur dioxide from weak gases and 95-98 percent of the sulfur dioxide from strong gases.

Flue gases are first cleaned, the cleaned gas stream then passing through a bubble-cap tower where the SO_2 is absorbed by dimethylaniline. Tail gases from the DMA absorption section in the bottom of the tower are scrubbed with a dilute sodium carbonate solution in the middle section of the absorption tower, thereby recovering residual sulfur dioxide from the gas stream. The sodium carbonate also neutralizes the sulfuric acid used for DMA vapor recovery as well as any acid formed through sulfur dioxide oxidation. The upper section of the absorption tower scrubs with dilute sulfuric acid to remove any remaining DMA vapor.

The loaded DMA solution is stripped with steam in the stripping section of the stripping column. Dimethylaniline and sulfur dioxide are recovered from the combined aqueous scrubber solutions by steam distillation in the lower section of the stripping tower. The hot gas stream leaving the stripper, containing sulfur dioxide, steam, and dimethylaniline vapor, is cooled in the upper or rectifier section of the stripping column. In the presence of the sulfur dioxide, dimethylaniline vapor is recovered as water soluble dimethylaniline sulfate. This leaves essentially pure sulfur dioxide which can be liquified. The process can recover from 90 to 95 percent of the sulfur dioxide from a gas stream containing 2 to 4 percent sulfur dioxide, and as much as 95 to 98 percent from a gas stream containing 8 to 10 percent sulfur dioxide.[8]

Monsanto Cat-Ox Process--The Cat-Ox process is particularly suited for gas streams lean in sulfur dioxide. As indicated in Figure 45, the incoming lean gas is partially heated by exchange with product gas effluents from the converter. The gas is then preheated to approximately 800 to 900°F, and finally is oxidized to sulfur trioxide in the presence of vanadium pentoxide catalyst. Conversions are on the order of 90 percent.

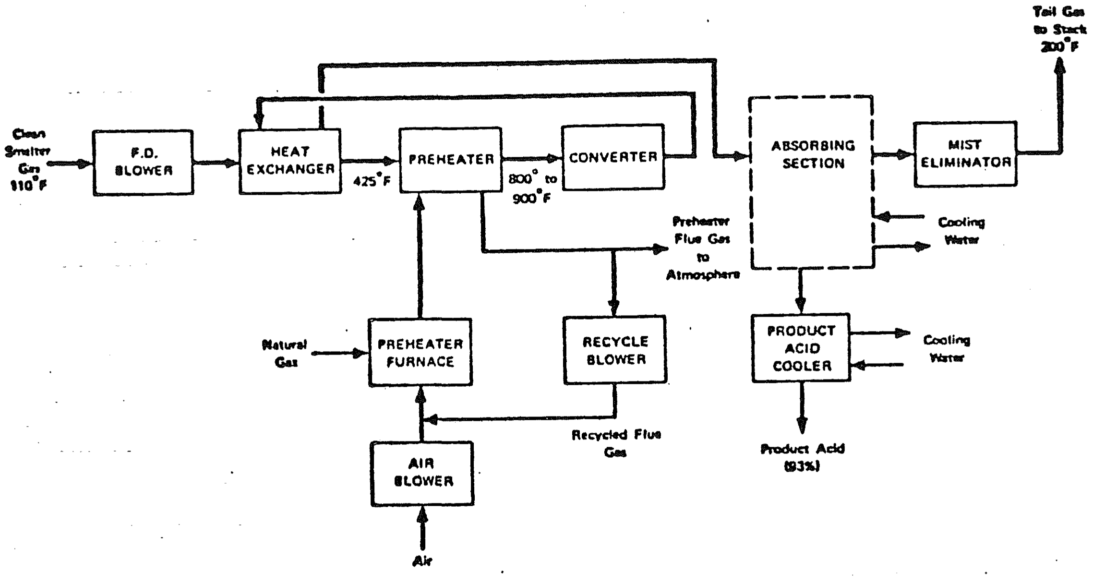


Figure 45. Cat-Ox System for Smelter Gases. (Source: Jones, 1972)

Comments

The Monsanto Cat-Ox system is essentially an adaptation of the contact sulfuric acid process.

Feed gas entering the system must be at a temperature high enough for conversion, otherwise additional heat must be supplied to the system.

The system operates on wet gas.

The heat of the exit gas is used to concentrate the sulfuric acid formed in this final absorption step.

Product gases from the converter are heat-exchanged against feed gases, the sulfur dioxide then absorbed in water. Product acid ranging in concentration from 80 to 93 percent is then recovered in a cooler, with offgases going to the stack after cleansing in a mist eliminator. Critical areas in the process include:

1) fly-ash removal upstream of the unit
2) life and efficiency of the catalyst
3) quality and concentration of the acid product
4) corrosion in the heat exchanger unit[30].

Wellman-Power Gas Process--The Wellman-Power Gas system is based on a cyclical absorption-desorption phase, with sodium sulfite solution as the absorbent (Figure 46). During the absorption cycle, the sulfite reacts with sulfur dioxide to form sodium bisulfite. The reaction is reversed in desorption, with concentrated sulfur dioxide released and sulfite regenerated and recycled back into the absorption system.

Gases coming from the smelter must be cleaned of particulate matter before treatment. Upon entering the absorber the gases come in contact with the sodium sulfite solution and the sulfur dioxide is tripped out by converting the absorbent to the biosulfite form. Exit gases contain approximately 500 ppm when input gas has a sulfur dioxide concentration of 1 to 2 percent. Desorption is accomplished quite easily by stripping the enriched bisulfite solution with steam in a countercurrent tower (7-15 # stream per # SO_2).

Any entrained stream in the sulfur dioxide gas is removed in a two-stage condensation process which removes the bulk of the water vapor at a relatively high temperature, and most of the remainder at approximately 120°F.

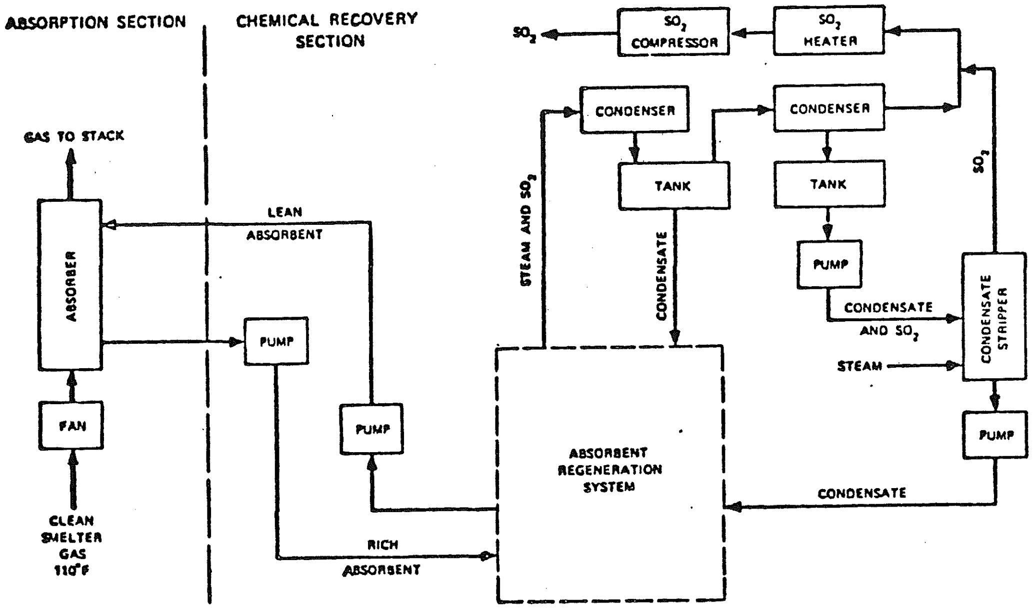


Figure 46. Wellman-Lord SO_2 Recovery System for Smelter Gases. (Source: Jones, 1972)

Comments

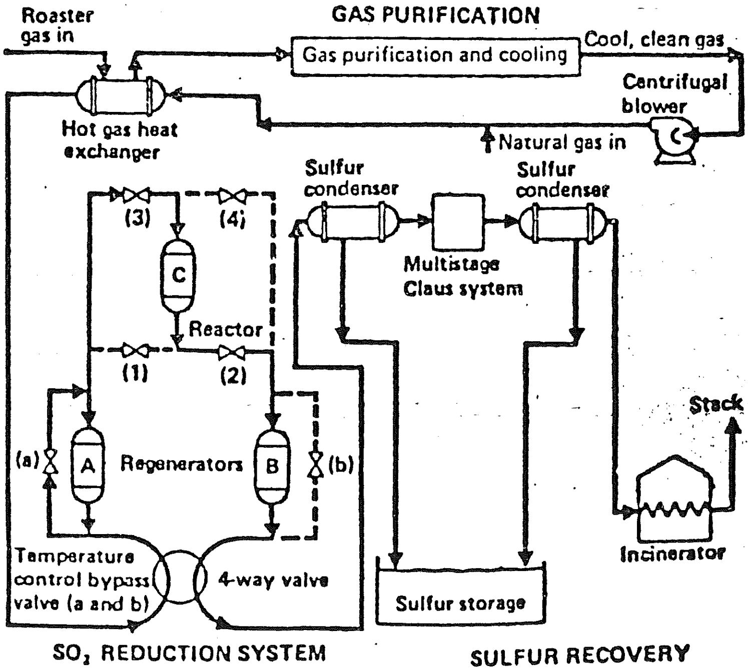
Application of the Wellman-Lord process to smelter gas is only speculative and cost estimates recently published are discouraging.

The product sulfur dioxide gas contains approximately 10 percent water vapor, which makes it suitable as feed to a sulfuric acid plant or a sulfur dioxide reduction plant.[30]

Allied SO$_2$ Process--The Allied SO$_2$ process involves the removal of sulfur dioxide from relatively rich sulfur dioxide gas streams. The process involves direct catalytic reduction of the sulfur dioxide to sulfur using natural gas as the reductant (Figure 47).

Gas purification is optional, but generally is considered necessary in order to remove particulates and excess water vapor. The cool, clean gas leaving this step is preheated by exchange with hot roaster gases before going to the reduction section.

Figure 47. Allied SO$_2$ Process Preserves Reaction Heat. (Source: Treilhard, 1973)



In the reduction section, approximately half of the sulfur dioxide is converted to elemental sulfur by reaction with methane, creating in the process H_2S. In the subsequent sulfur recovery system, sulfur formed in the reactor is condensed and the remaining sulfur dioxide and H_2S are reacted in a multistage claus conversion system to produce additional sulfur via the reaction:

$$2\ H_2S + SO_2 = 2\ H_2O + 3S$$

All sulfur recovered in the condensers is sent to storage and exit gas incinerated to remove traces of H_2S before it is exhausted to the atmosphere.[30]

Bureau of Mines Citrate Process--In this process sulfur dioxide bearing gases are cleansed of dust and fume by precipitation or filtering, followed by water scrubbing. The wet gases have the mist removed by another precipitator, then pass into an absorber system where they are contacted with a partially neutralized solution of citric acid. Ninety to ninety-five percent of the sulfur dioxide is claimed to be removed.

The sulfur dioxide stream next goes to a reactor unit where hydrogen sulfide reacts with the absorbed sulfur dioxide to yield elemental sulfur. This is separated from the citric acid solution by filtration, with the solution recycled to the absorber.

A small portion of the sulfur product is diverted to a H_2S generator, where it is burned with steam and methane to produce H_2S and CO_2, the latter passing through the process and exiting from the reactor (Figure 48).[30]

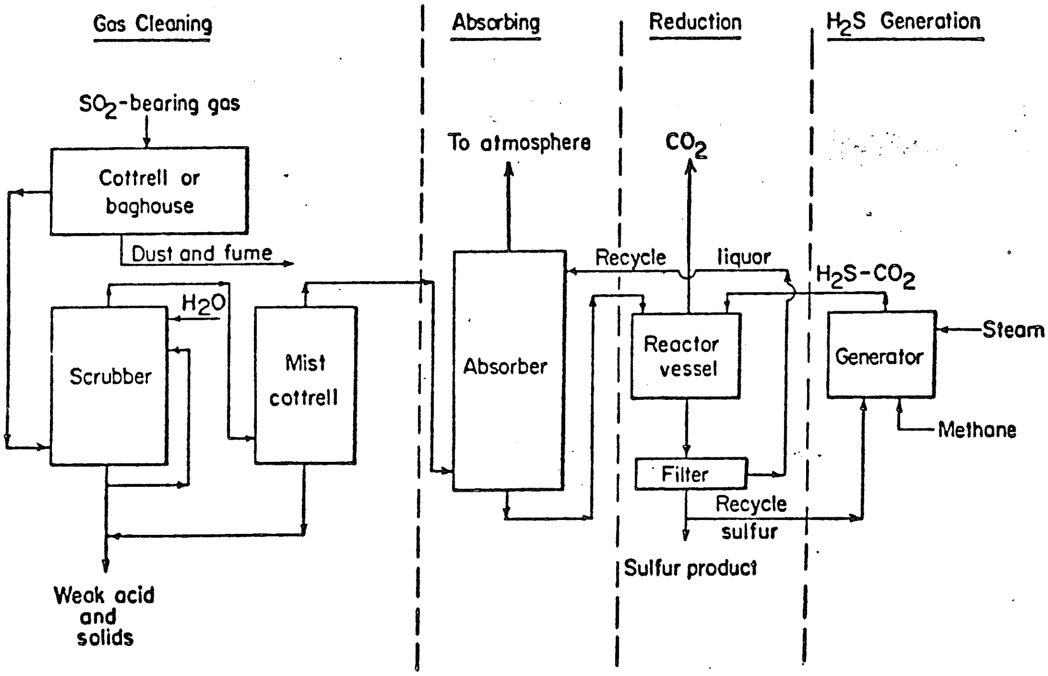


Figure 48. Citrate Process. (Source: Jones, 1972)

Limestone Scrubbing Process--A system for removal of sulfur dioxide from effluent gases by exposure of such gases to materials such as limestone exists. Designed systems convert the sulfur dioxide to an innocuous form (Figure 49).

Flue gas is precleaned of fly ash, then blown into the bottom of a spray tower where it is sprayed with a slurry containing 7 to 10 percent powdered limestone. As contact is made, the limestone reacts with the sulfur dioxide to yield insoluble calcium sulfite and sulfate. Slurry effluent drains by gravity into a delay-and-mixing tank, wherein supersaturated calcium salts settle out before the solution is recycled back to the scrubber. Fresh limestone is added in this tank.

A portion of the recycle stream is directed to a clarifier that returns a fairly clear overflow stream to a wet electrostatic precipitator into which the treated gases flow after scrubbing. Underflow from the classifiers is filtered so that precipitated calcium salts can be eliminated from the system and filtrate returned to the mixing tank.

Other limestone-based processes exist, many in various stages of development.[8]

Closed-Loop Control System--This approach to control of sulfur dioxide emissions involves the use of special automated instrumentation for the continuous measurement of sulfur dioxide in the air. The system includes designed monitoring stations, digital computers which question each monitor and printing out average SO_2 values for each station periodically, a full-time meteorologist,.plus supporting staff with extensive weather data recording and receiving equipment.

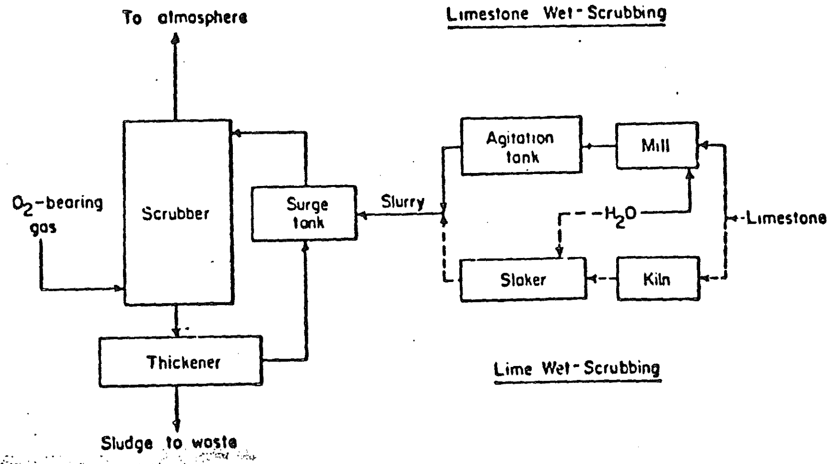


Figure 49. Lime or Limestone Wet-Scrubbing Process. (Source: Jones, 1972)

Comments

In the Limestone Wet-Scrubbing process, a good grade of Limestone is required, hence, substantial costs in mining, hauling, grinding, and calcining will be realized.

Disposal of the final product will involve problems similar to those encountered in acid neutralization.

Is flexible and applicable to variable gas flows and sulfur dioxide concentrations.

Two techniques are used to limit sulfur dioxide emissions. The first involves reaction to increasing concentrations of sulfur dioxide, the second involves utilization of meteorological data in the prediction of low winds and inversions.

A third technique in the developmental stage involves continuous measurement of sulfur dioxide emissions with the simultaneous prediction of downwind SO_2 concentrations at ground level.

SUMMARY

Pyrometallurgy

Table 18 compares the production features attributable to existing smelting type operations. Also, refer to Table 19 for a statistical review of the salient features of each principal piece of pyrometallurgical equipment; refer to Figure 50 which illustrates, in a very generalized combined flowsheet, the various choices in current and developing technology offered to smelter designers.

Figure 51 compares and contrasts the quality of metal product produced from the various pyrometallurgical systems. The reverberatory and electrical furnaces produce a matte of composition A, unless they are preceeded by fluidized or hearth roasters, in which case matte is converted slightly to the composition of point B. The flash smelting furnace and also the converting furnace of the Mitsubishi process produce a matte of approximately 55 to 60 percent copper, near point C. In the Noranda process reactor, the matte can be converted all the way to point E (98 percent Cu), unless there is an electrorefining problem due to the retention of high bismuth or antimony content in the anodes produced. In this case, the conversion in the reactor proper is stopped at the white metal stage of point D. The Kennecott converter-smelter process also converts concentrates to the white metal stage of point D.[29]

Table 18. Production Details of Smelting Furnaces.

TYPE	BLAST	REVERBERATORY	ELECTRIC	FLASH	CONTINUOUS
Hearth Area (M^2)	6.6-11.4	315-360	84-350	133-140	----
Solid Charge Rate (Total Metric tons/day)	250-1300	400-1300	250-1650	1100-1600	115-1075
Matte (Metric tons/day)	110-650	250-800	150-1100	550-850	20-180 (Blister Cu, ≈98% Cu)
Matte Grade (% Cu)	46-63	34-47	22.5-40	47-49	----
Slag (Metric tons/day)	120-700	260-900	160-1360	300-550	65-700
Slag Grade (% Cu)	0.15-1.0	0.38-0.60	0.16-0.63	0.55-1.0	0.50-12.0
Productivity (Metric tons of charge per day/M^2)	35-130	1.2-3.6	3.0-5.8	7.9-11.6	3.0-9.0
Total Energy in Fuel (Kcal/Metric ton of charge)	$6x10^5$-$8x10^5$	$10x10^5$-$16x10^5$	$3.5x10^5$-$5.9x10^5$	$1.1x10^5$- $6.1x10^5$	$6x10^5$-$13x10^5$
Total Energy as an Average (Kcal/metric ton of charge)	$6.75x10^5$	$13.33x10^5$	$4.18x10^5$	$4.45x10^5$	$10x10^5$ (without air enrichment)

Table 19. Salient Features of Pyrometallurgical Equipment and Apparatus.

APPARATUS	DESCRIPTION AND DIMENSIONS	CAPACITY	GAS GRADES	MAJOR DISADVANTAGES	MAJOR ADVANTAGES
Multiple Hearth Roaster	Cylindrical-brick-lined vessel		2-6% SO_2	More expensive to operate in comparison to other calcining operations. Calcine Carryover of 6% in outlet gases.	
Fluid Bed Roaster	Cylindrical-refractory-lined steel shell	1600 TPD Concentrate	12-14% SO_2	Processing difficulties. Excessive calcine carryover of 80% in outlet gases.	No moving parts in the combustion chamber. Maintenance requirements are at a minimum. Uniform bed temperatures and composition are maintained in the fluid bed. Little excess air is required.
Sintering Machine	Traveling straight line grate 25 ft. - 35 ft. long 5 ft. - 8 ft. wide		1.5-2% SO_2	Requires a very careful adjustment of the physical and chemical conditions of the charge. Sulfur control is not very practical.	Excellent hearth efficiency.
Conventional Blast Furnace	Rectangular in plan with vertical sides at the top, inwardly sloping sides in the center section, and a vertically sided hearth at the bottom 1.5 m x 7 m x 5 m	1000 TPD Concentrate	5% SO_2	The blast furnace is unable to treat finely ground flotation concentrates.	The blast furnace has a high productivity rating. The typical blast furnace uses a relatively small amount of fuel.
Momoda Blast Furnace		500 TPD charge	7.3% SO_2 9.5% CO_2 3.1% O_2	Lacks capacity for larger smelter operations.	Lower BTU requirements per ton of charge when compared with the BTU requirements per ton of wet charge in reverberatory.
Reverberatory Furnace	A long horizontal furnace with a roof designed to reflect flame down onto a charge on the hearth 120-150 ft. long; 30-40 ft. wide High temperatures 1400-1700°C Tapping temperature 1100-1300°C Gas temperatures 1100°C	1600 TPD charge	.5-1.0% SO_2 11-20% of sulfur content removed	Thermal efficiency of the reverberatory is low.	Furnace possesses flexibility with respect to feed. Capable of handling large tonnages of hot charge per day.
Electric Furnace	Rectangular in cross-section with a firebrick spring-arch roof and a basic-brick inverted-arch bottom		2-4% SO_2	High hydrostatic heads attributable to deep layered slags constitute a definite furnace operating hazard. Bath runaways are a major hazard.	It is completely versatile and it can be use to smelt any and all materials. It produces small volumes of effluent gas (consisting of N_2 from infiltrated air, CO

Table 19. (contd.)

lectric Furnace (cont.)				Copper matte is more difficult to produce than nickel matte. Furnace makes little use of the energy which is potentially available from oxidizing the sulfide minerals of the charge. The operating costs tend to be high due to the high price of electrical energy.	and CO_2 from electrode-slag reactions and SO_2 from sulfide oxidation). The SO_2 concentration of its effluent gas is readily controlled by adjusting the amount of air which is infiltrated into the furnace. It makes efficient use of electrical energy.
Flash Furnace (Outokumpu) (not autogenous) Requires ½ the fuel of an equivalent (dry) reverberatory smelting process.	20 m x 7 m x 3 m	1200 TPD-1400 TPD Concentrate	14% SO_2	Slags are too high in copper content to be discarded. An additional electric furnace is needed to treat both flash furnace slag and converter slag.	Its small physical size supports a large throughput. It is very economical in its use of fuel, the greater portion used for air heating, and therefore, of low quality. The gas produced is a good feedstock for an acid plant. Direct production of elemental sulfur in the flash-smelting emission system is being developed.
Flash Furnace (INCO) (autogenous)	Small hearth-type furnace 23 m x 6 m x 5 m	1600 TPD dry charge	75-80% SO_2	Slags are too high in copper content to be discarded. Additional treatment is needed for both flash furnace slag and converter slag.	It has a much lower overall energy requirement. Its volume of effluent gas (per ton of charge) is small due to the absence of nitrogen and hydrocarbon combustion products, which means that its gas-collection equipment requirement is small. The SO_2 concentration in its effluent gas is very high (80%) which simplifies SO_2 removal as sulfuric acid, liquid SO_2, or elemental sulfur. Dust losses are low due to a relatively small volumetric gas-flow rate. Its productivity (tons of charge per day/M^2 of hearth area) is approximately 30% higher than that of the Outokumpu process.
Pierce-Smith Sideblown Converter	Cylindrical steel vessel lined with basic refractory bricks. 13 ft. diameter; 30 ft. length		2-6% SO_2	Relatively low concentration of SO_2 in outlet gas.	Very efficient as a smelting machine under certain modified conditions.

Table 19. (contd.)

Hoboken Horizontal Siphon-Type Converter	56.5 TPD blister copper	8% SO_2	Capital costs are high than Pierce-Smith converter Capacity per unit size is lower than comparable units. Efficiency as a smelting machine is low.	
Mitsubishi Process (Continuous)	500 TPD-1500 TPD concentrates	12-15% SO_2	The Mitsubishi process may not be suitable for treating high impurity concentrates. (Concentration of impurities in the blister copper is likely to be high.) The multi-step Mitsubishi process produces SO_2-bearing gas in each of its three furnaces and a more extensive gas-collection system is required.	The system is pollution free, all exit gases have a SO_2 content greater than 10% which will permit recovery of sulfur as sulfuric acid Capital investment is 70% of that needed to build a conventional smelter. Operating costs are reduced due to a smaller work force. Blister copper requires relatively little fire refining due to its relatively low (.1-.9% S) sulfur content. Slag recycle to the melting furnace is kept to a minimum by the small production of slag.
WORCRA Process (Continuous) (Utilizes submerged lances)		9-12% SO_2	The heat from the oxidation reactions is not available to the settling branch. As a consequence, hydrocarbon fuel must be burnt in the settling branch and large volumes of combustion gas, dilute in SO_2 (1 or 2%) are produced. Only a small part of the furnace is devoted to copper-making reactions and hence the productivity of the furnace is low, 2-3 tons of charge per M^2 of hearth area.	Capital costs 20-30% below that of reverberator converter plants. Lower operating costs, with savings dependent upon local costs for fuel, power and labor. Economic viability at low annual throughputs-possible 10,000 to 20,000 tons/year of copper. Efficient recovery of byproduct sulfur from continuous SO_2 emission. Process makes metal rather than matte directly from concentrates. Most of the exothermic oxidation reactions are generated and continued within the liquid bath. Injection of oxygen-containing gases via lances creates turbulence and continuous flow in the smelting and converting zones. In the converting zone, slag moves by gravity, generally countercurrent to matte and metal. Copper content of the slag is reduced to throwaway levels as the slag flows through the smelting and slag-cleaning zones; there is no revert slag.

Table 19. (contd.)

NORCRA Process (cont.)					Offgases generated in the smelting and converting stages are combined and leave the furnace continuously. SO_2 concentration is mostly in the 9 to 12% range.
Noranda Process (Continuous) (Uses submerged tuyeres and a rotatable reactor)	Horizontal cylindrical furnace with a raised hearth at one end, and a depression near the middle in which the copper collects. 13 ft. diameter; 60 ft. length	800 TPD-1600 TPD concentrate		The sulfur content of Noranda process blister copper (1-2% S) is considerably higher than that of conventional converter copper and hence requires more air and a much lower oxidation period in the anode furnace. The Noranda process may not be suitable for treating high impurity concentrates. (Concentration of impurities in the blister copper is likely to be high.)	
Kennecott Converter-Smelter Vessel		400-500 TPD concentrate			

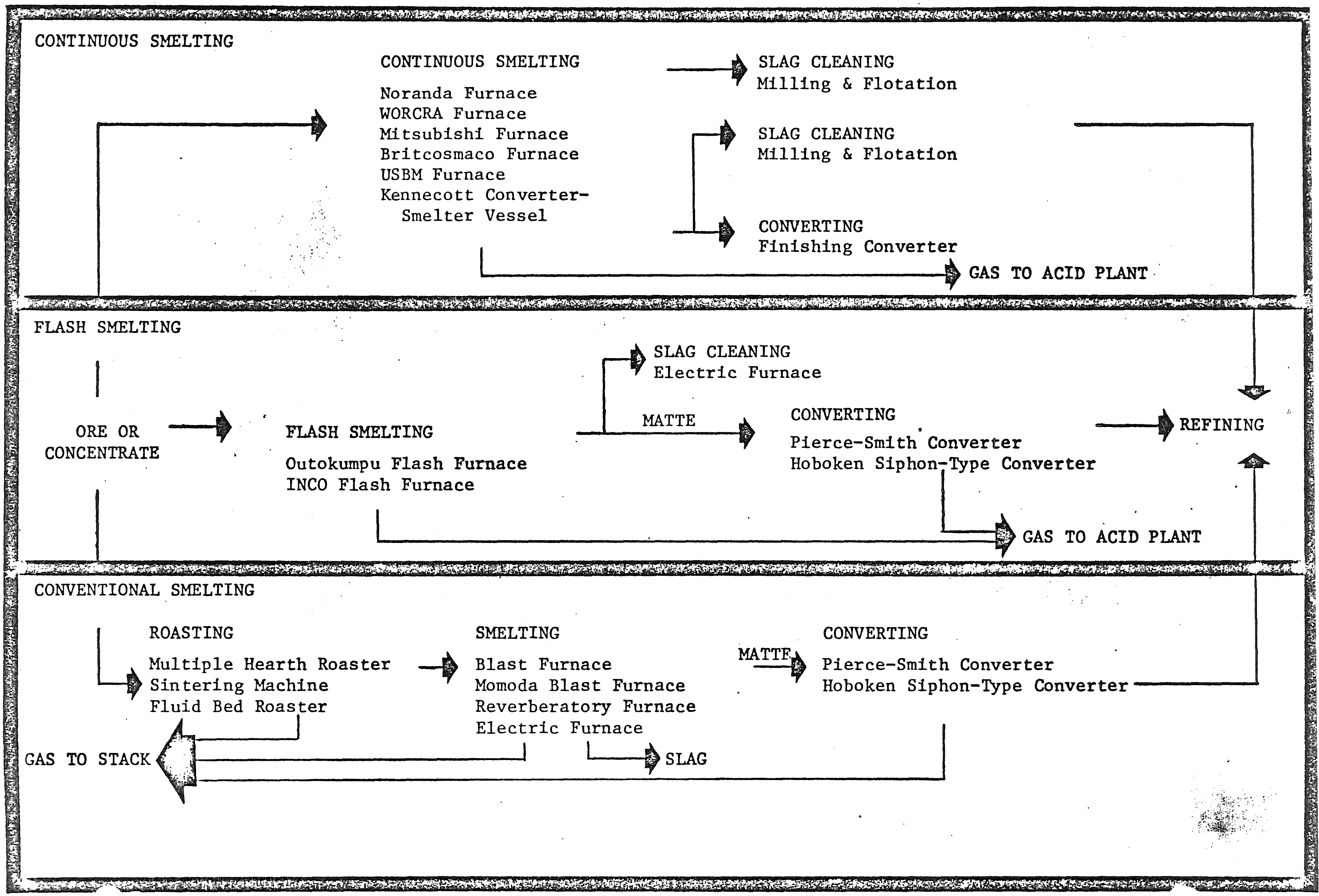
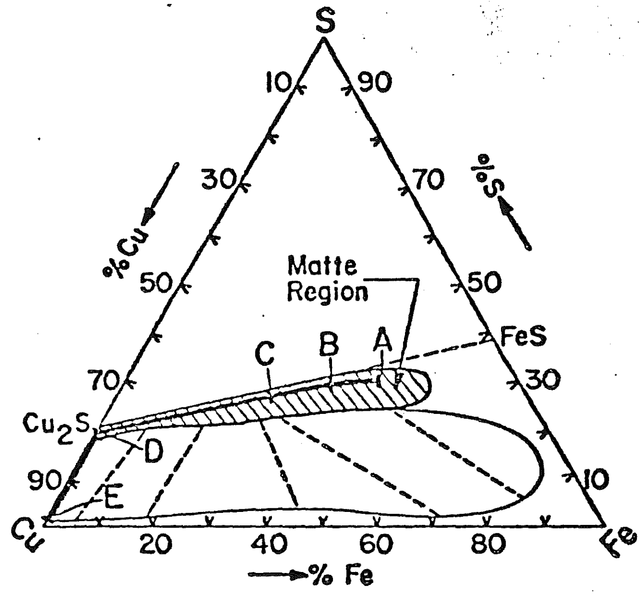


Figure 50. Combined Flowsheet Shows Variety of Choices in Current and Developing Technology Offered to Smelter Designers.

Figure 51. Copper Matte Composition Diagram. (Source: Rutledge, 1975)



Data gathered from existing operations have been tabulated in graphical form and are displayed in Figures 52 through 91. The first family of curves (Figures 52 through 61) represent computer print plots of the distribution of the major elements of the concentrate into the matte, slag, and other phase categories (product categories resulting from the pyrometallurgical treatment of the concentrates). The "other" phase category includes elemental distribution into the particulates, gases, fugitives, etc. All curves relate elemental distribution to the matte grade (percent copper in the matte). Assuming that the curves are truly indicative of the distribution of the elements during smelting, these plots indicate the change in the distribution of said elements with a change in matte grade. As the matte grade increases:

the distribution of copper;

decreases in the matte phase
increases in the slag and "other" phases
reaches a minimum value in the matte phase at approximately a 40 to 50 percent copper matte grade

the distribution of nickel;

- decreases in the matte phase
- increases in the slag phase

the distribution of sulfur;

- decreases in the matte phase
- increases in the "other" (gas) phase

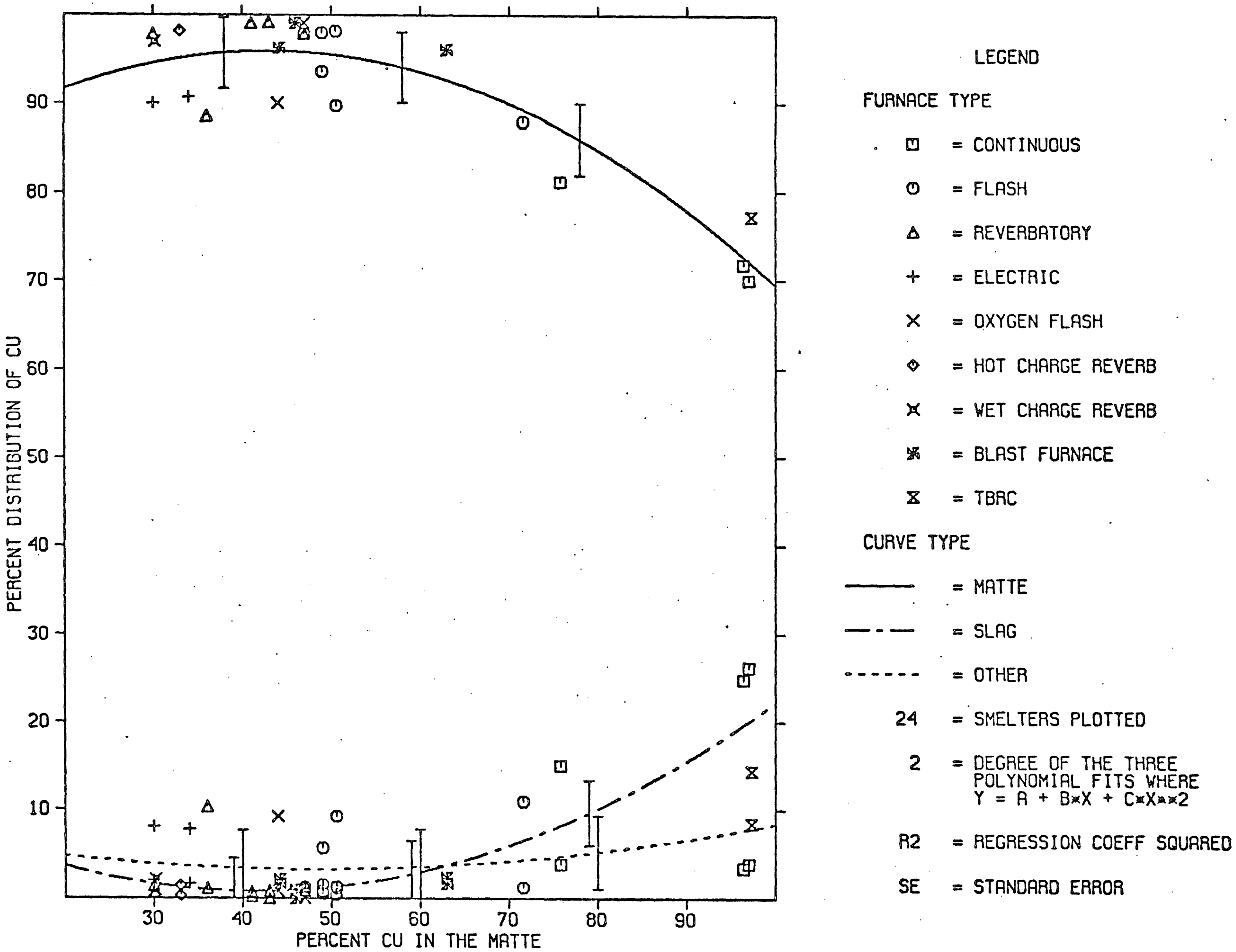
the distribution of iron;

- decreases in the matte phase
- increases in the slag phase

This information may be of use to the metallurgist in predicting trends to be anticipated with changes in operating procedures. The error bars,

Figure 52.

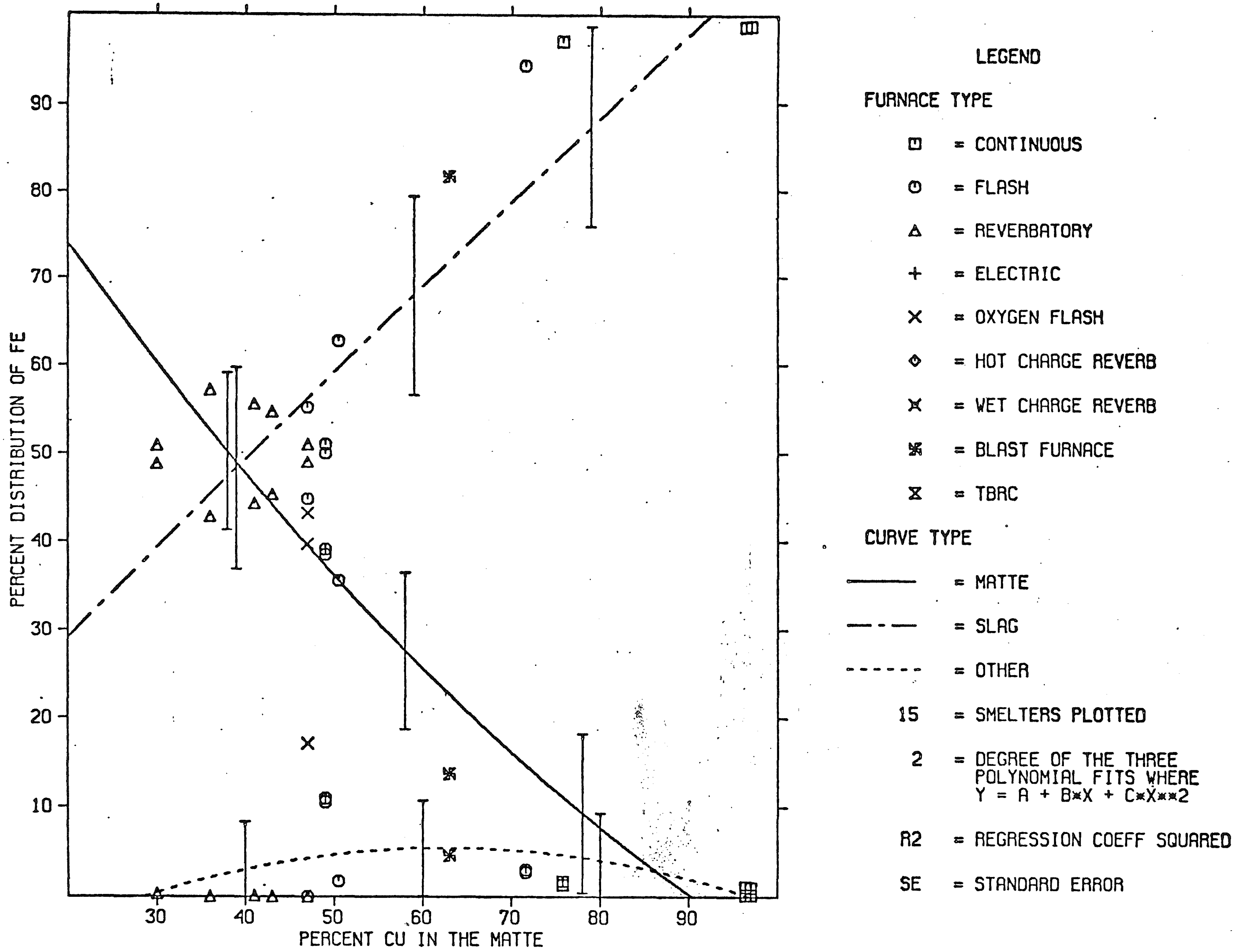
SMELTER DISTRIBUTION PLOT



MATTE CURVE. COEFFS	A = 9.1632	B = .3743	C = -.0814	R2 = .8085	SE = .4007
SLAG CURVE. COEFFS	A = .3615	B = -.2669	C = .0624	R2 = .7698	SE = .3668
OTHER CURVE. COEFFS	A = .4752	B = -.1075	C = .0190	R2 = .1189	SE = .4191

Figure 53.

SMELTER DISTRIBUTION PLOT



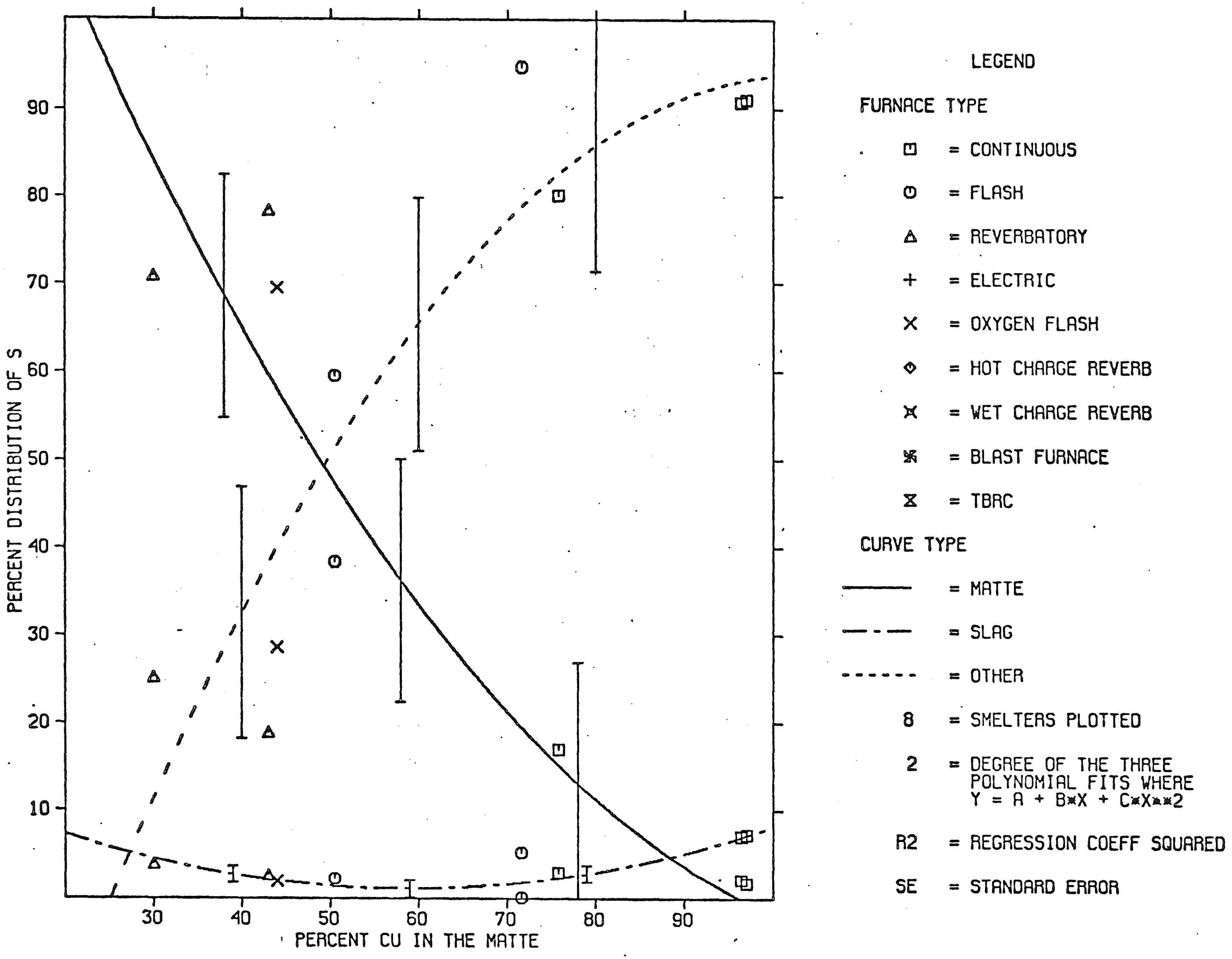
MATTE CURVE. COEFFS A = 7.3965 B = -1.4165 C = .0519 R2 = .8468 SE = .8899

SLAG CURVE. COEFFS A = 2.9135 B = 1.0175 C = -.0053 R2 = .7791 SE = 1.1382

OTHER CURVE. COEFFS A = -.3100 B = .3990 C [illegible] -.0466 R2 = .1262 SE = .5261

Figure 54.

SMELTER DISTRIBUTION PLOT



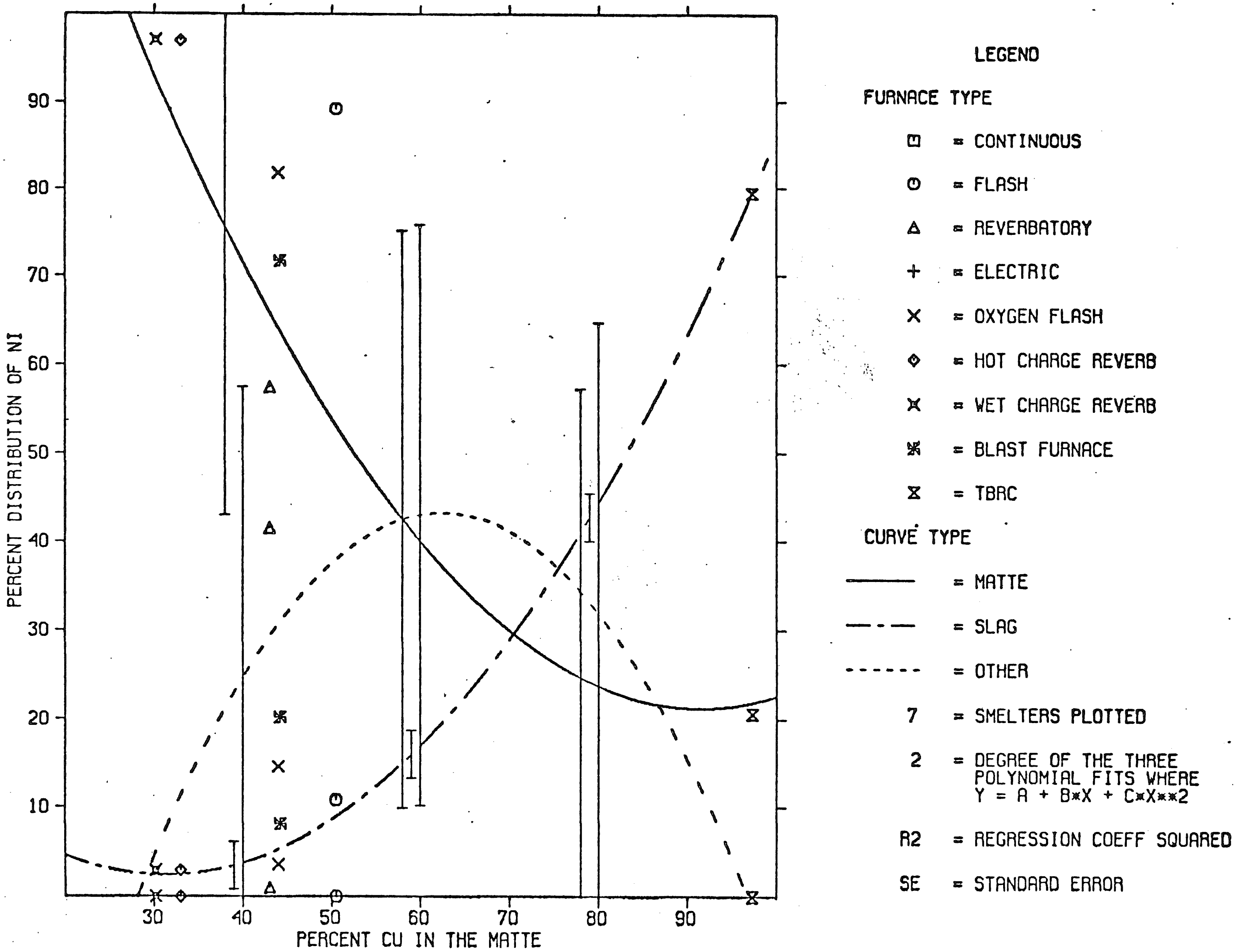
MATTE CURVE, COEFFS A = 10.5637 B = -2.2634 C = .1151 R2 = .8769 SE = 1.3842

SLAG CURVE, COEFFS A = .7242 B = -.3197 C = .0413 R2 = .8972 SE = .0951

OTHER CURVE, COEFFS A = -1.2880 B = 2.5832 C = -.1564 R2 = .8604 SE = 1.4374

Figure 55.

SMELTER DISTRIBUTION PLOT



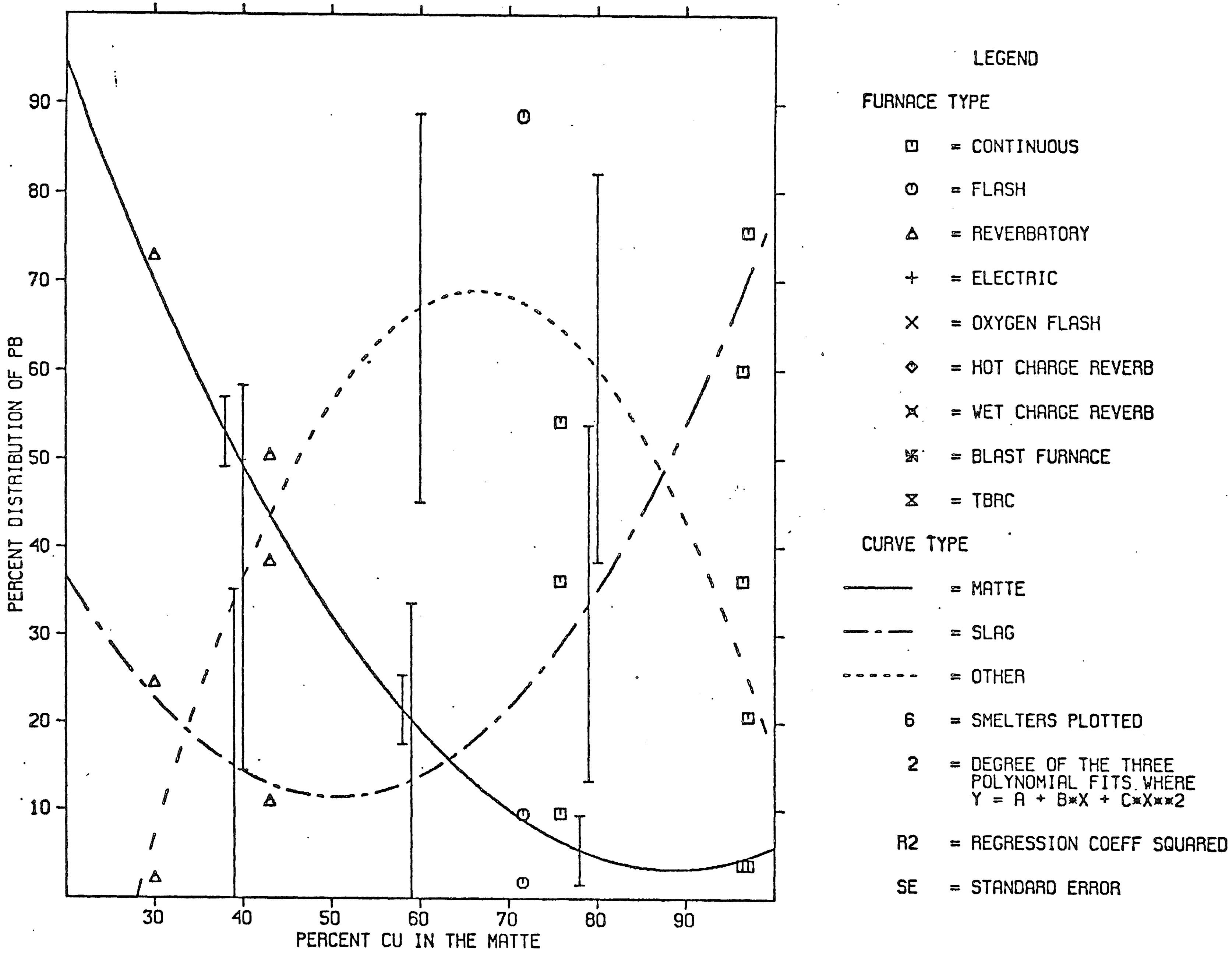
MATTE CURVE, COEFFS A = 11.8378 B = -2.7205 C = .1904 R2 = .4309 SE = 3.2558

SLAG CURVE, COEFFS A = .4611 B = -.3973 C = .1767 R2 = .9940 SE = .2684

OTHER CURVE, COEFFS A = -2.2988 B = 3.1178 C -.3672 R2 = .2449 SE = 3.2824

Figure 56.

SMELTER DISTRIBUTION PLOT



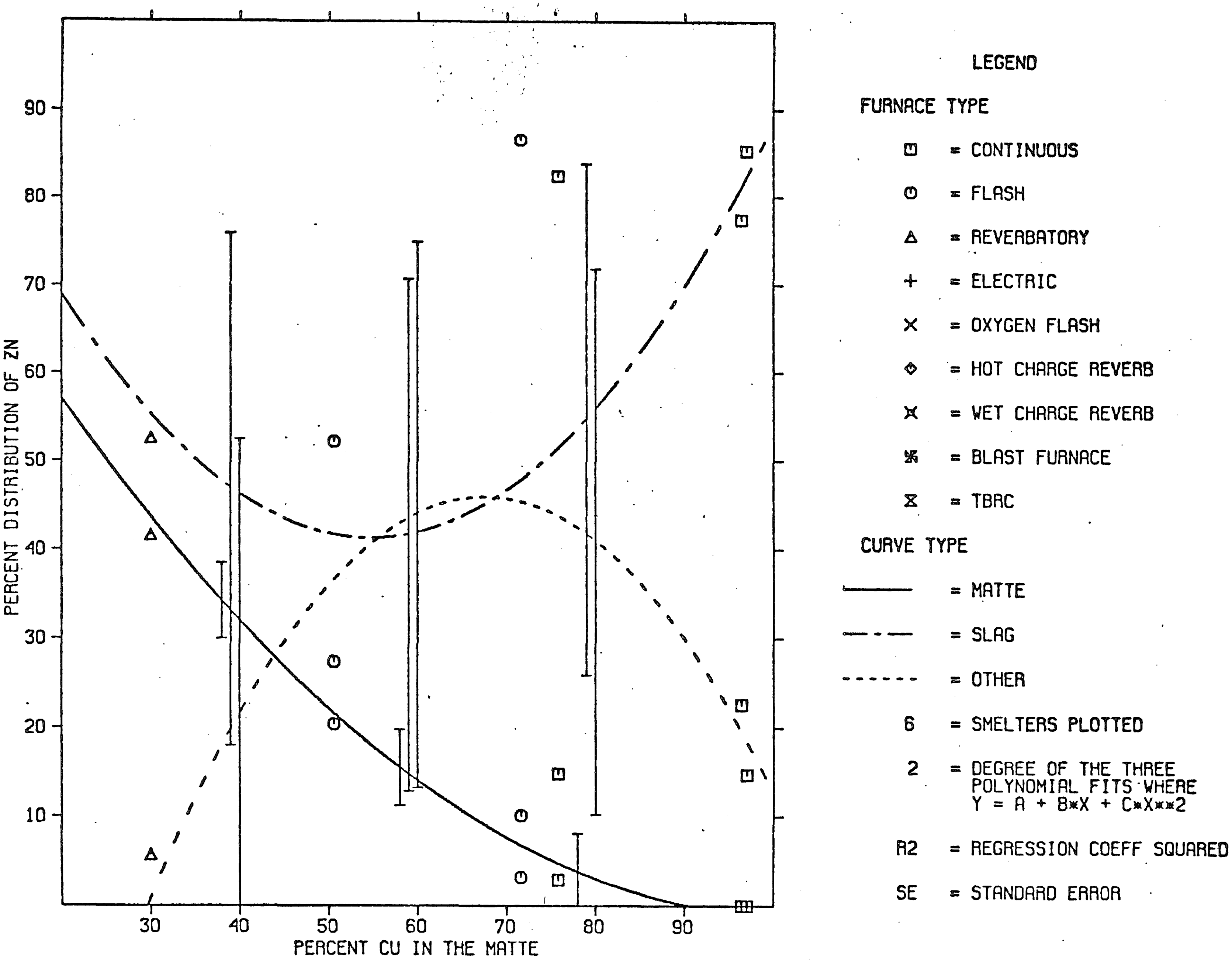
MATTE CURVE, COEFFS A = 9.4810 B = -2.6702 C = .1947 R2.= .9877 SE = .3962

SLAG CURVE, COEFFS A = 3.6697 B = -1.6605 C = .2723 R2 = .7191 SE = 2.0234

OTHER CURVE, COEFFS A = -3.1507 B = 4.3307 C = -.4671 R2 = .6649 SE = 2.1918

Figure 57.

SMELTER DISTRIBUTION PLOT



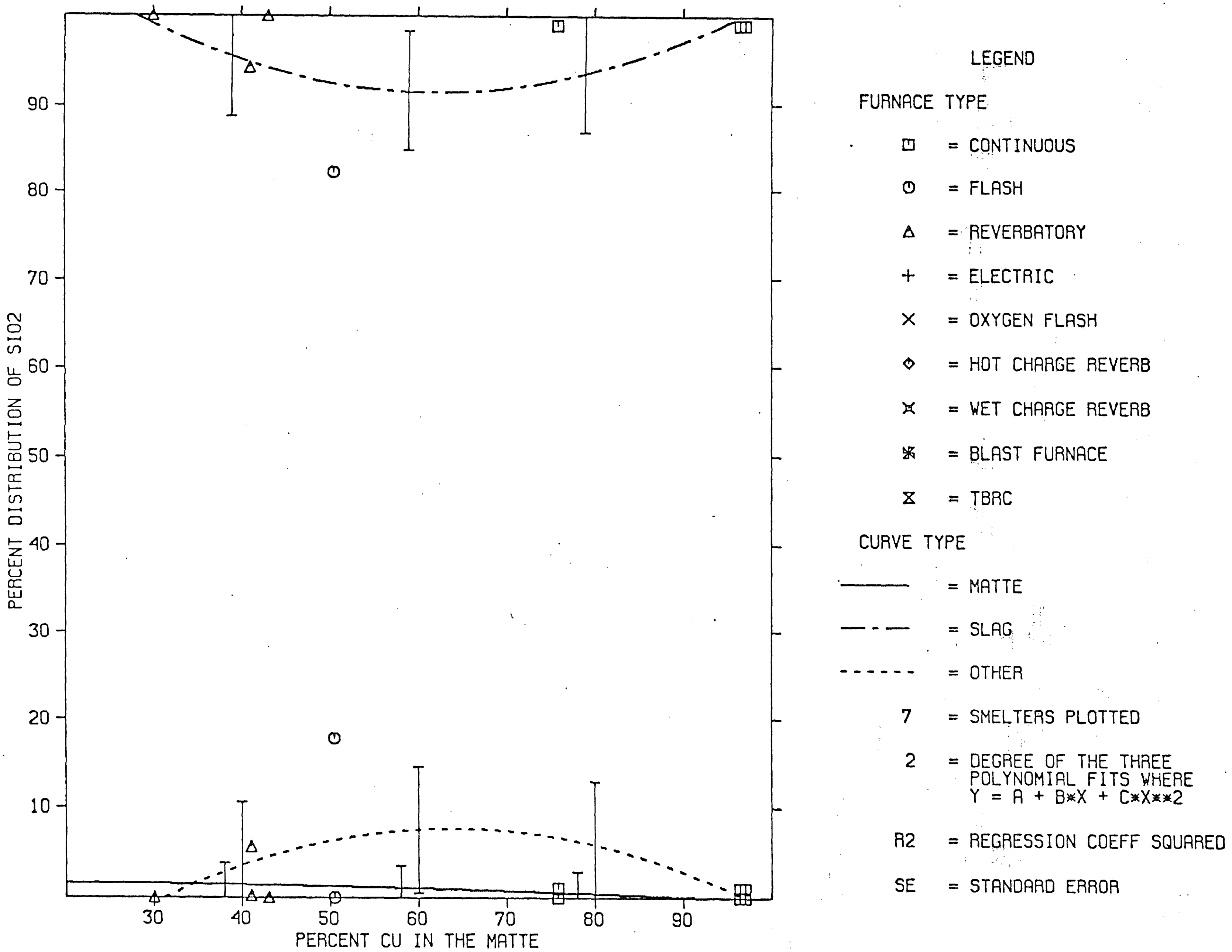
MATTE CURVE. COEFFS A = 5.7056 B = -1.4249 C = .0873 R2 = .9648 SE = .4281

SLAG CURVE. COEFFS A = 6.9018 B = -1.5967 C = .2299 R2 = .3780 SE = 2.8955

OTHER CURVE. COEFFS A = -2.6075 B = 3.0216 [illegible] -.3173 R2 = .3468 SE = 3.0840

Figure 58.

SMELTER DISTRIBUTION PLOT



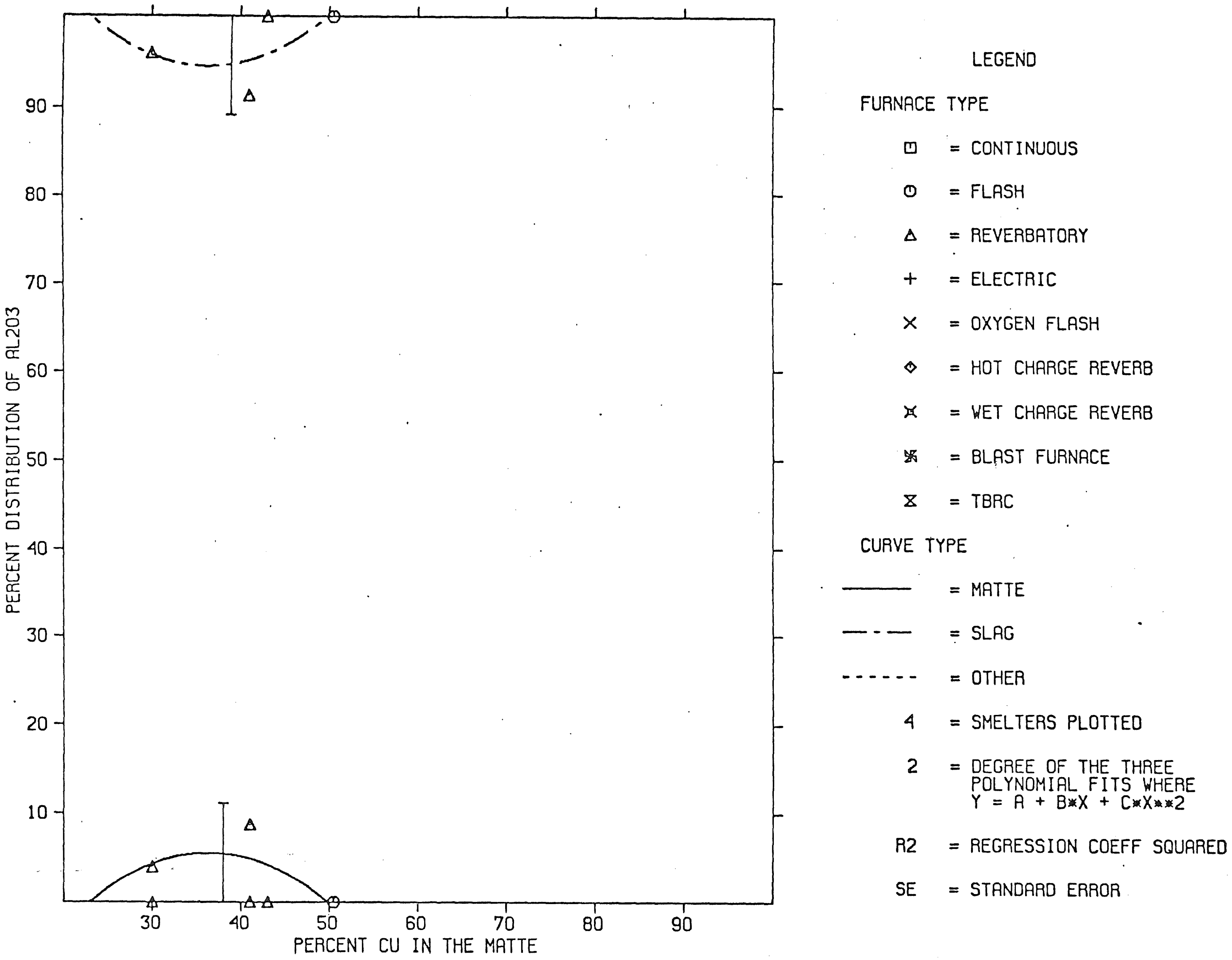
MATTE CURVE, COEFFS A = .1536 B = -.0045 C = -.0023 R2 = .1146 SE = .2422

SLAG CURVE, COEFFS A = 10.4531 B = -.6181 C = .0730 R2 = .2792 SE = .6733

OTHER CURVE, COEFFS A = -.6067 B = .6226 C = -.0707 R2 = .2240 SE = .7046

Figure 59.

SMELTER DISTRIBUTION PLOT



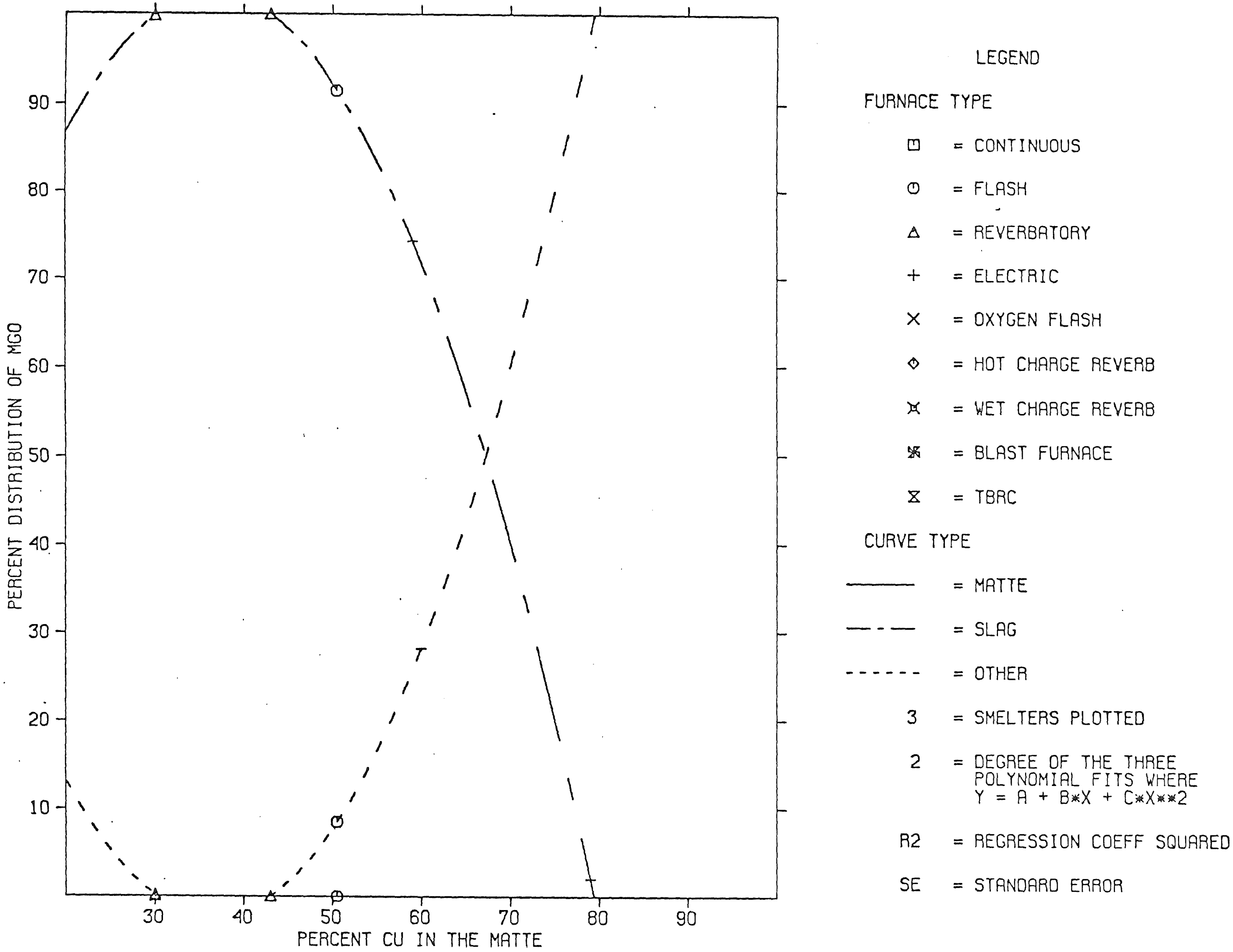
MATTE CURVE, COEFFS A = -.2624 B = .9858 C = -.3009 R2 = .3650 SE = .5682

SLAG CURVE, COEFFS A = 10.2639 B = -.9879 C = .3015 R2 = .3649 SE = .5689

OTHER CURVE, COEFFS A = -.0016 B = .0021 [illegible] = -.0005 R2 = .3868 SE = .0007

Figure 60.

SMELTER DISTRIBUTION PLOT



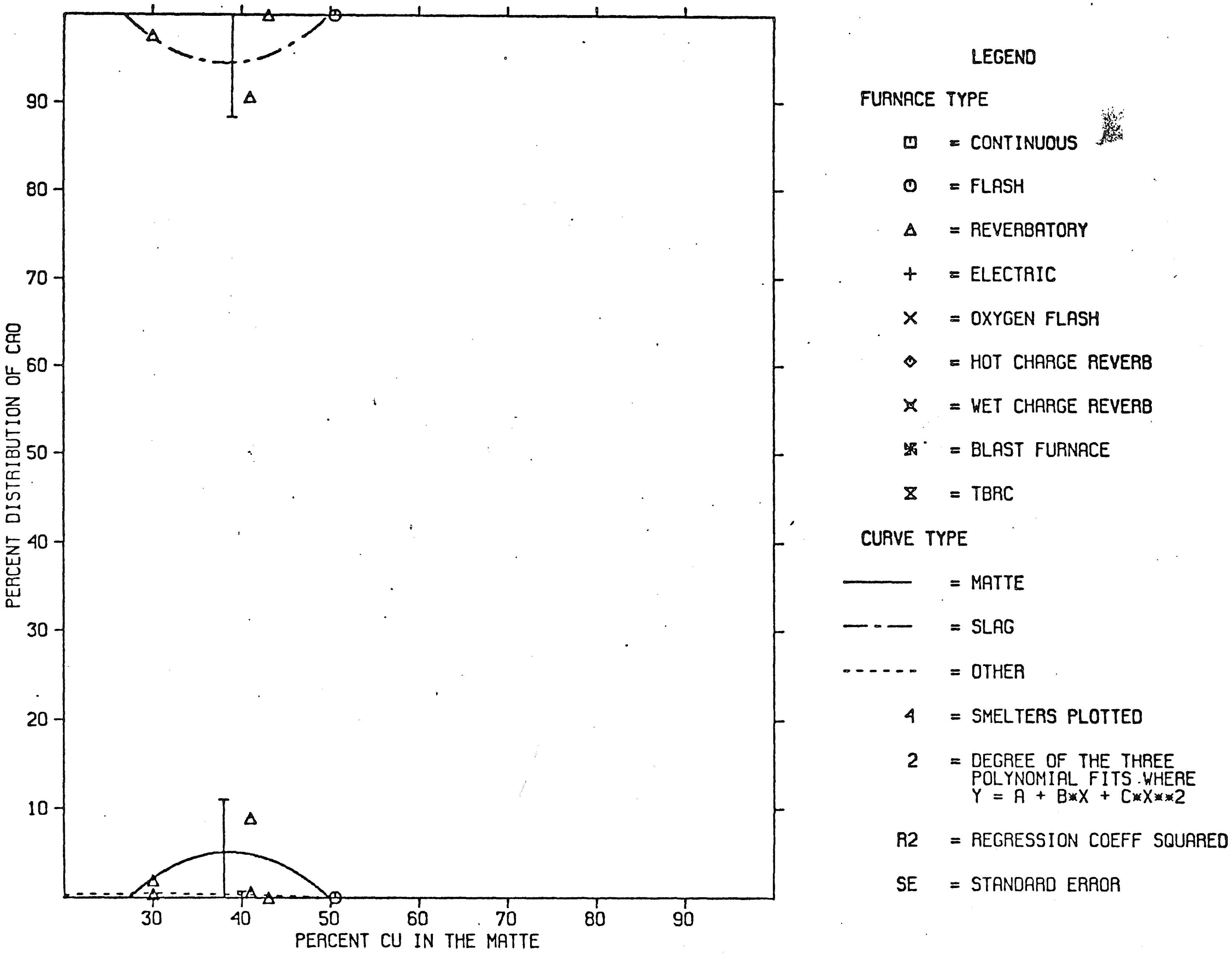
MATTE CURVE, COEFFS A = .0000 B = -.0000 C = .0000 R2 = R SE = 0

SLAG CURVE, COEFFS A = 8.6750 B = 1.8621 C = -.5591 R2 =1.0000 SE = 0

OTHER CURVE, COEFFS A = 1.3250 B = -1.8621 C = .5591 R2 =1.0000 SE = 0

Figure 61.

SMELTER DISTRIBUTION PLOT



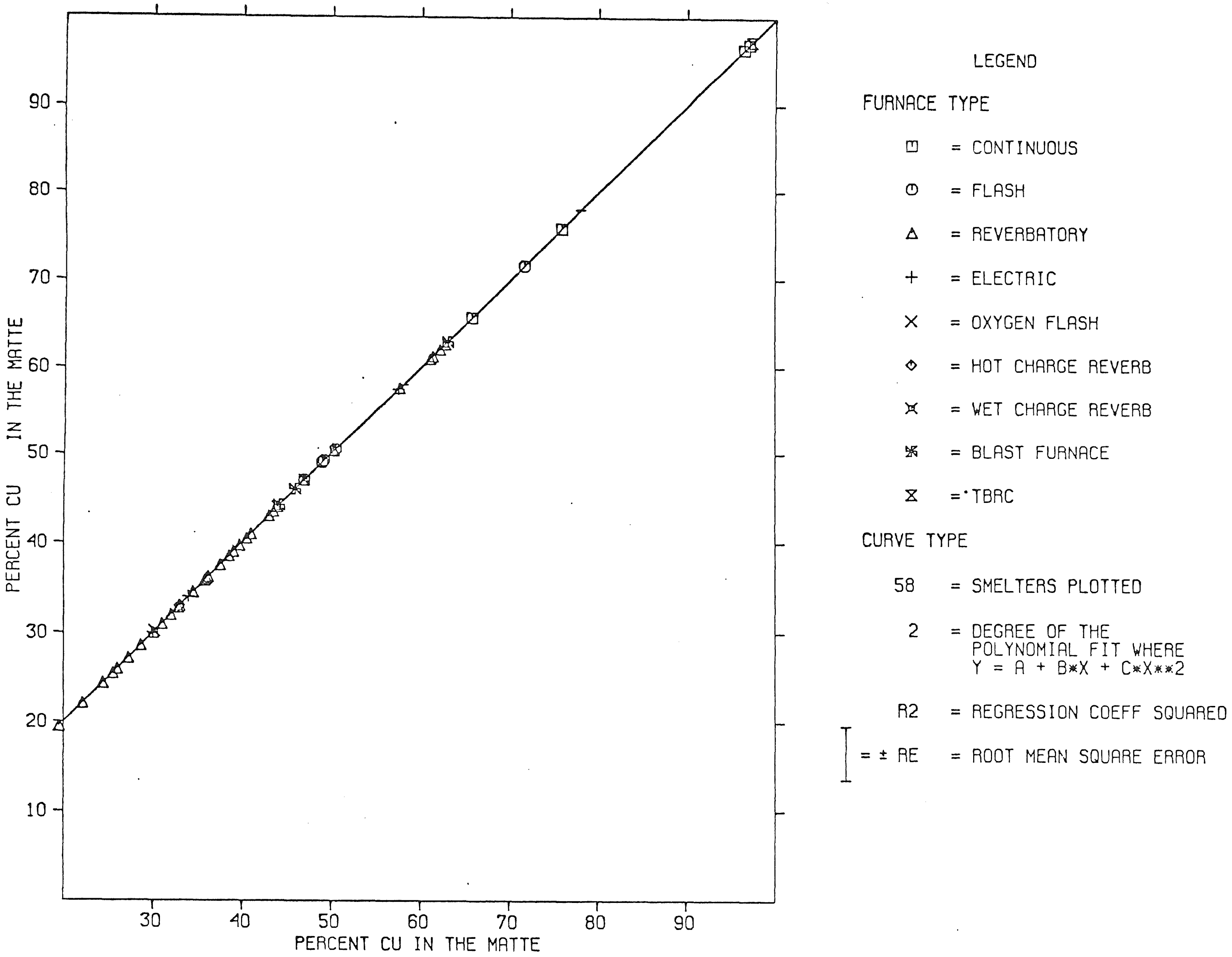
MATTE CURVE. COEFFS A = -.8668 B = 1.4794 C = -.3981 R2 = .3419 SE = .5919

SLAG CURVE. COEFFS A = 10.8331 B = -1.5033 C = .4099 R2 = .3398 SE = .6277

OTHER CURVE. COEFFS A = .0337 B = .0239 C -.0118 R2 = .5003 SE = .0359

Figure 62.

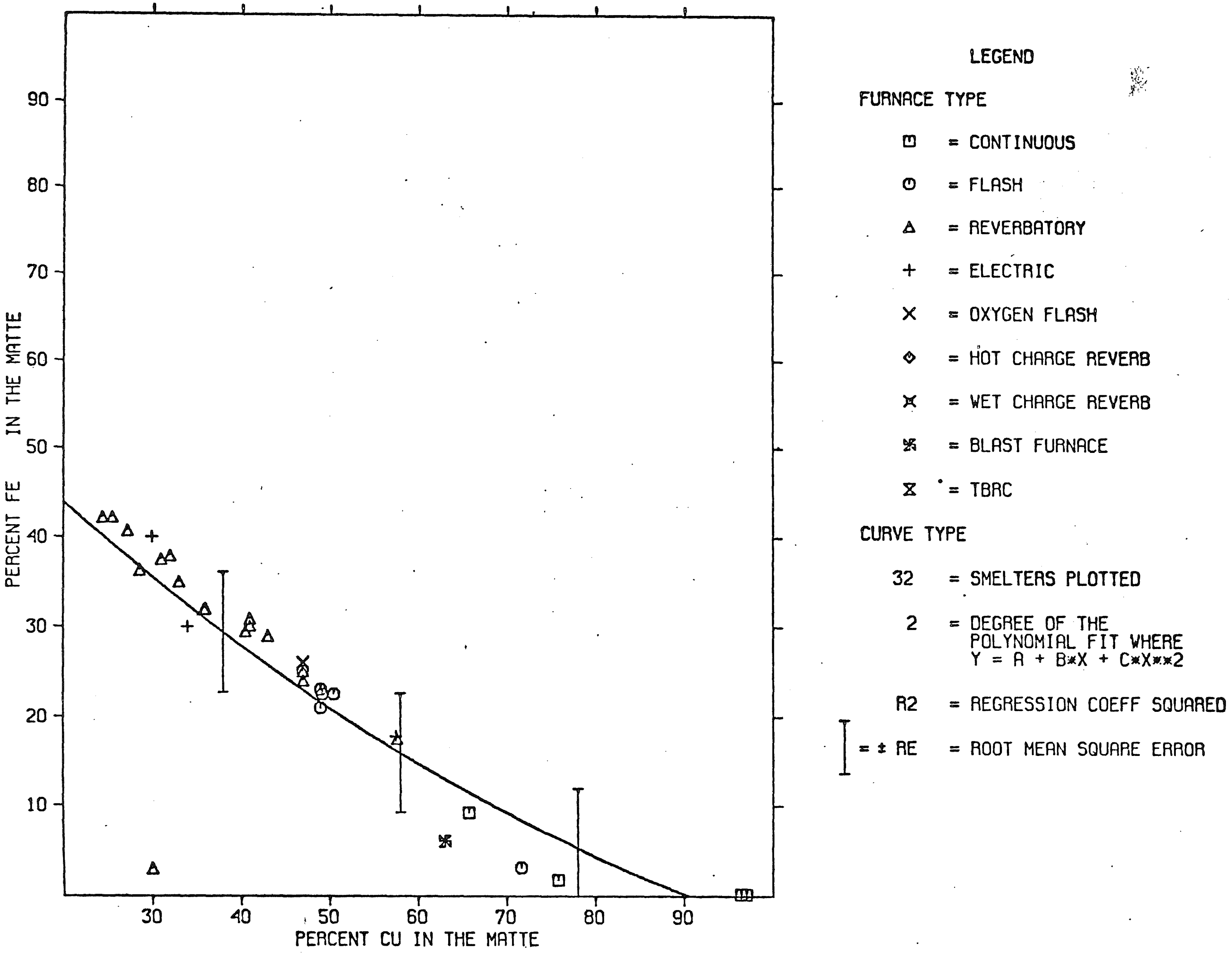
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = 2.0000 B = 1.0000 C = -.0000 R2 =1.0000 RE = .0000

Figure 63.

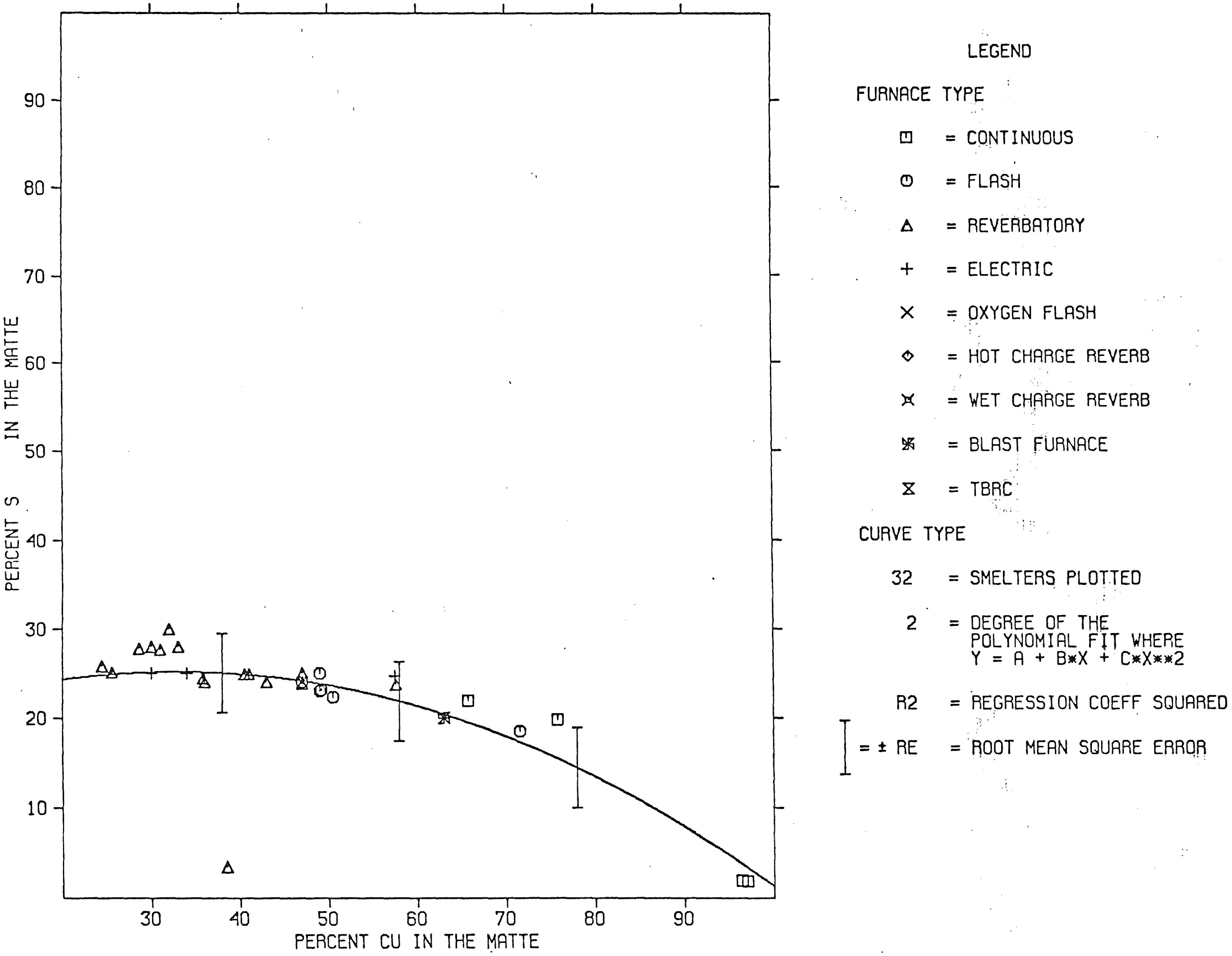
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = 4.3946 B = -.8732 C = .0355 R2 = .7531 RE = .6708

Figure 64.

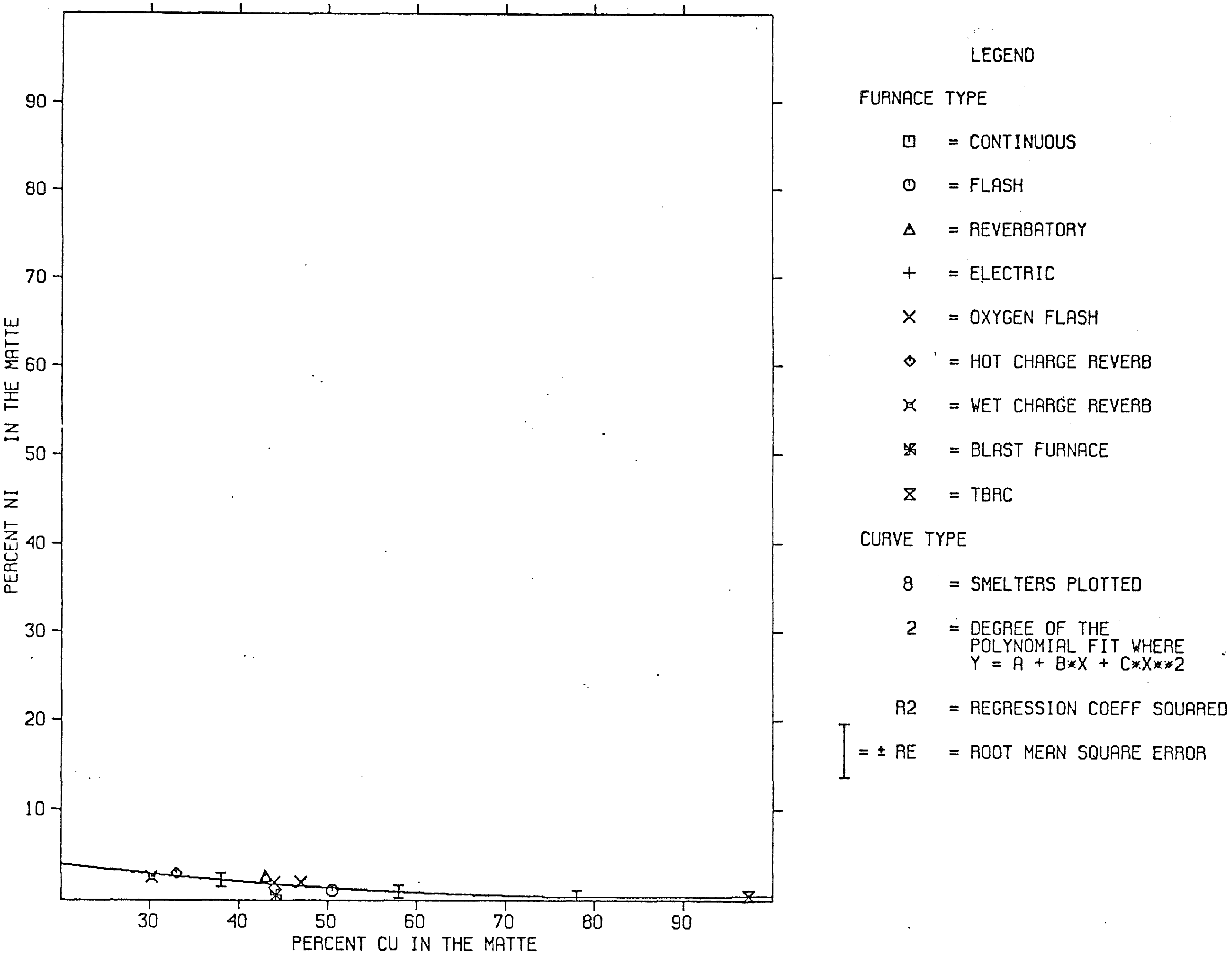
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = 2.4295 B = .1375 C = -.0530 R2 = .6188 RE = .4422

Figure 65.

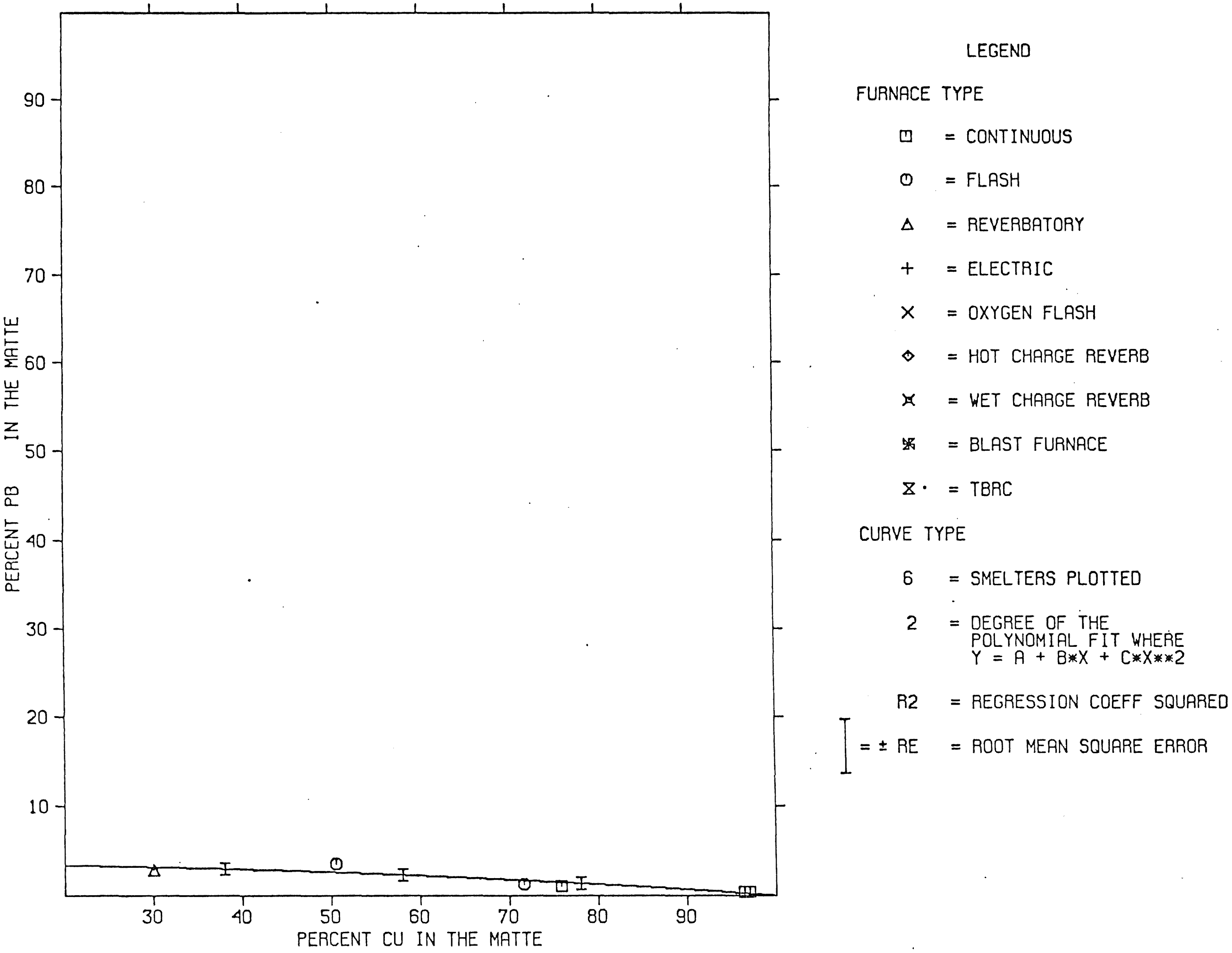
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = .3986 B = -.1121 C = .0086 R2 = .6070 RE = .0720

Figure 66.

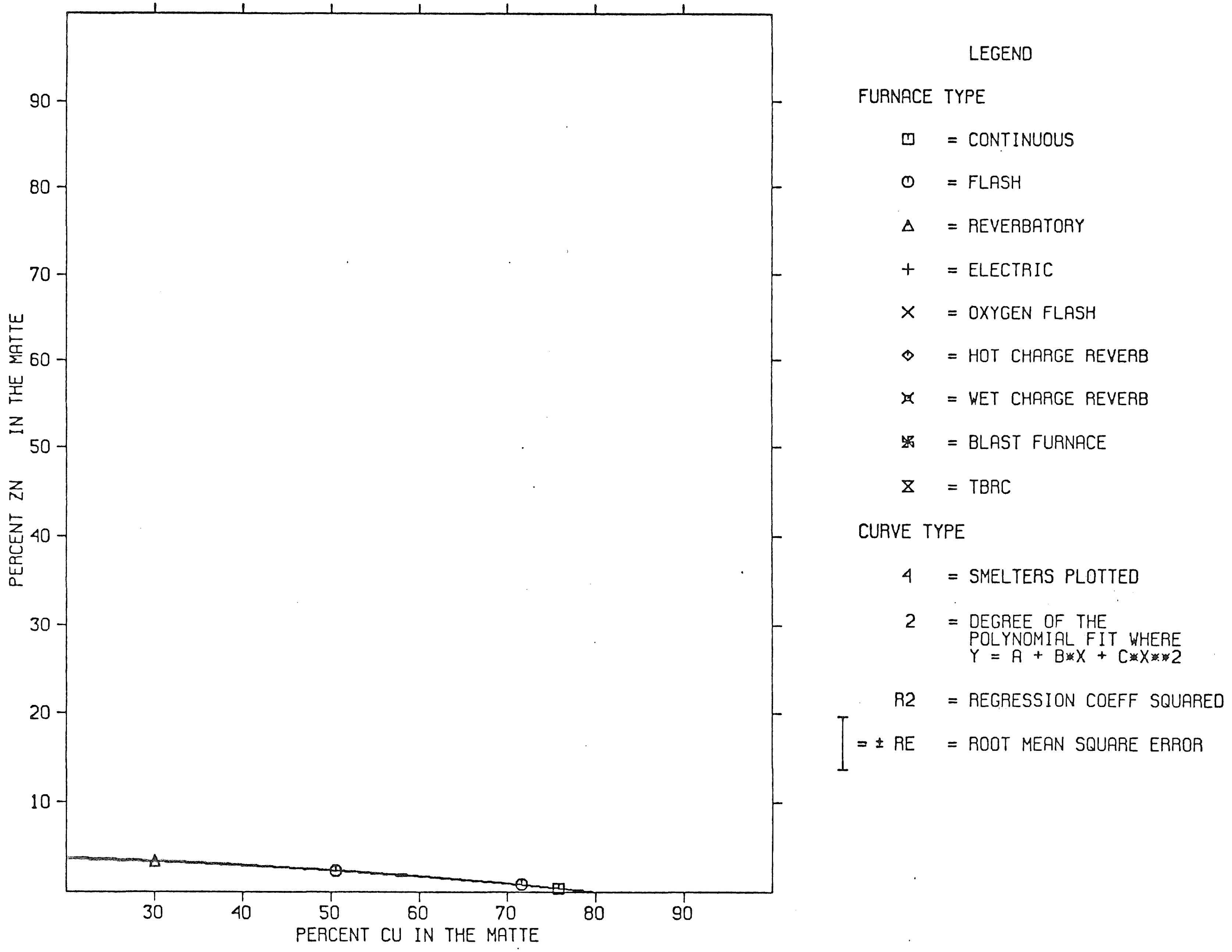
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = .3413 B = -.0145 C = -.0035 R2 = .8446 RE = .0673

Figure 67.

SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = .3919 B = -.0259 C = -.0065 R2 = .9992 RE = .0072

Figure 68.

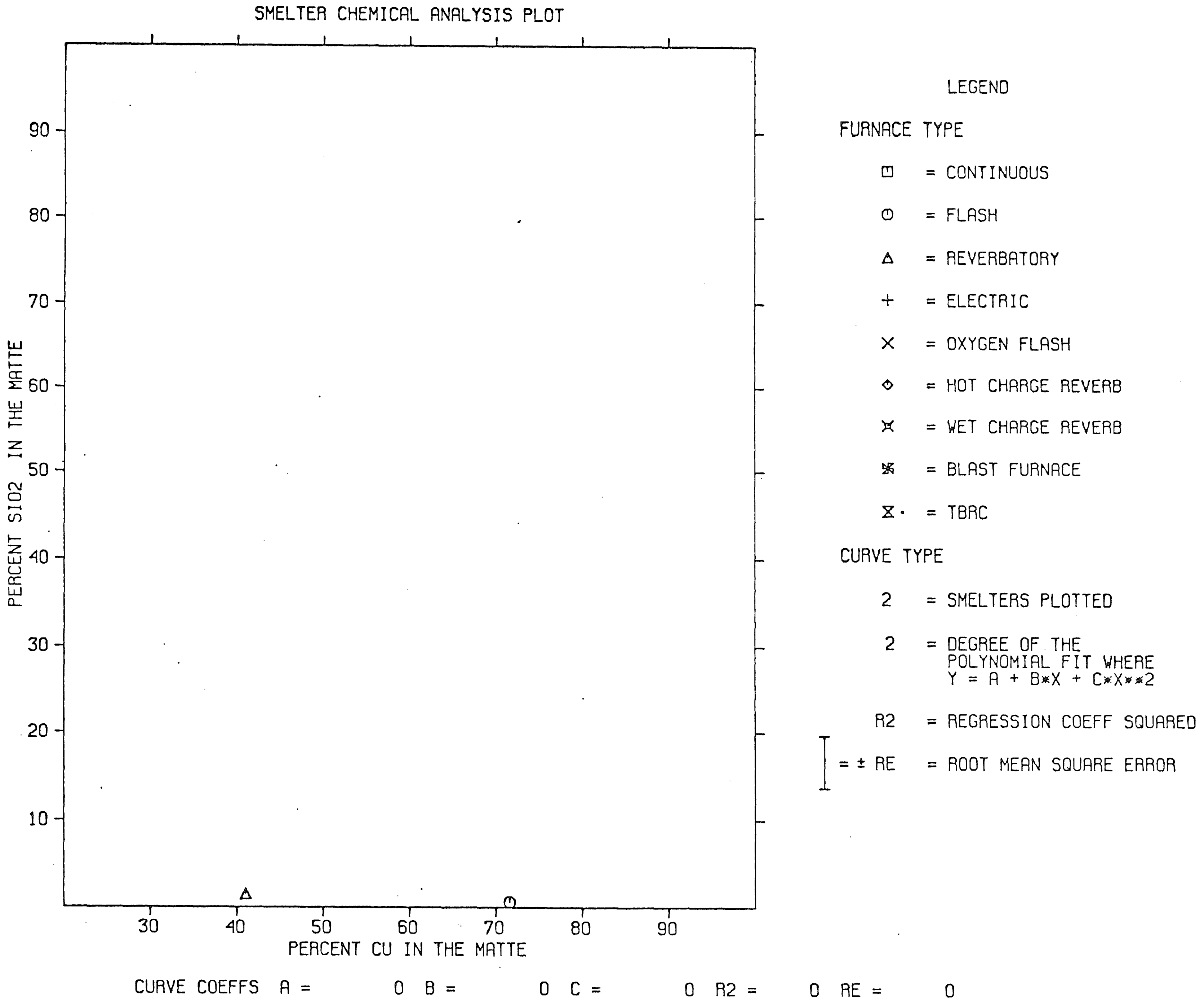
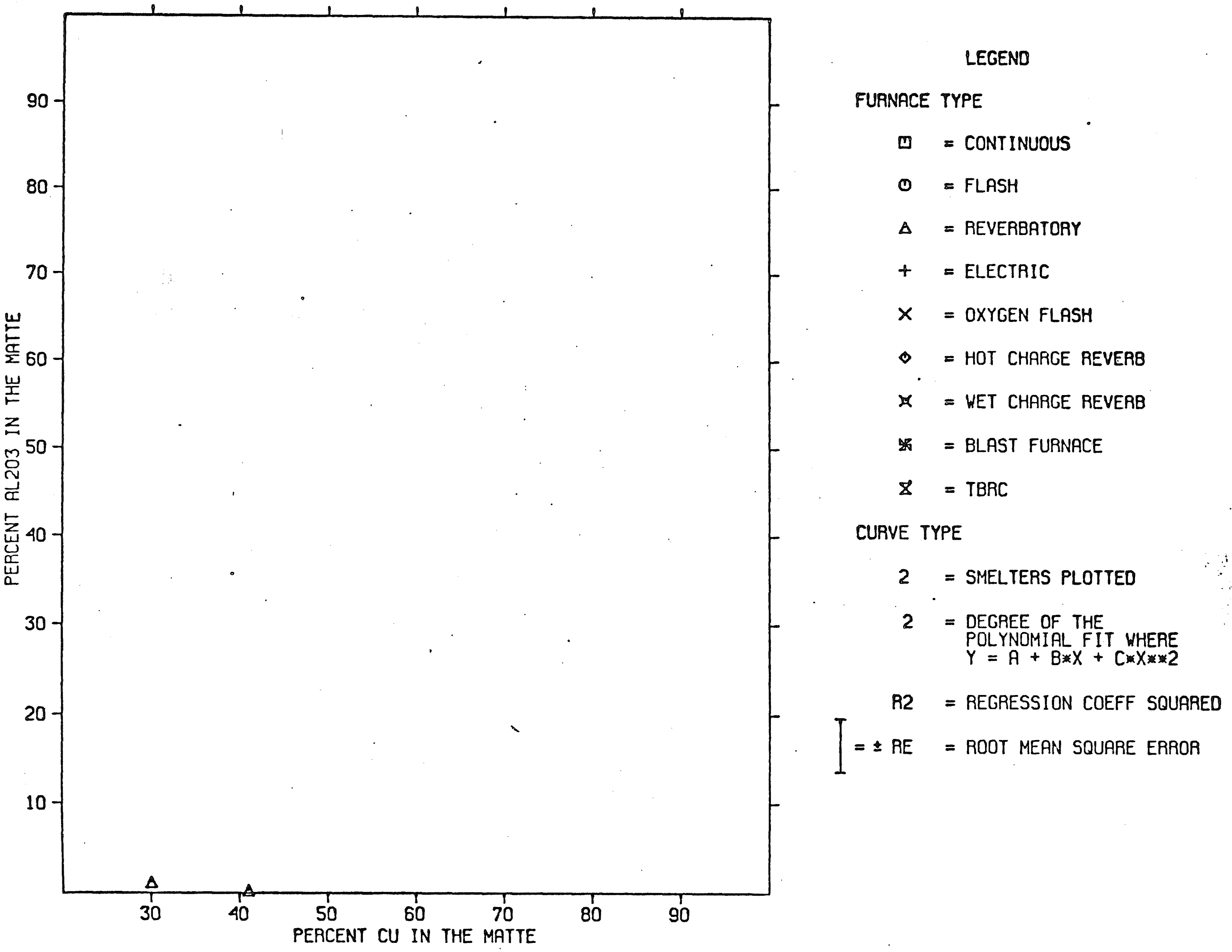

SMELTER CHEMICAL ANALYSIS PLOT
PERCENT SIO2 IN THE MATTE
10
20
30
40
50
60
70
80
90
PERCENT CU IN THE MATTE
30
40
50
60
70
80
90
LEGEND
FURNACE TYPE
⊡ = CONTINUOUS
⊙ = FLASH
△ = REVERBATORY
+ = ELECTRIC
× = OXYGEN FLASH
◇ = HOT CHARGE REVERB
⌘ = WET CHARGE REVERB
✻ = BLAST FURNACE
⧗· = TBRC
CURVE TYPE
2 = SMELTERS PLOTTED
2 = DEGREE OF THE POLYNOMIAL FIT WHERE Y = A + B*X + C*X**2
R2 = REGRESSION COEFF SQUARED
I = ± RE = ROOT MEAN SQUARE ERROR
CURVE COEFFS A = 0 B = 0 C = 0 R2 = 0 RE = 0

Figure 69.

SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = 0 B = 0 C = 0 R2 = 0 RE = 0

Figure 70.

SMELTER CHEMICAL ANALYSIS PLOT

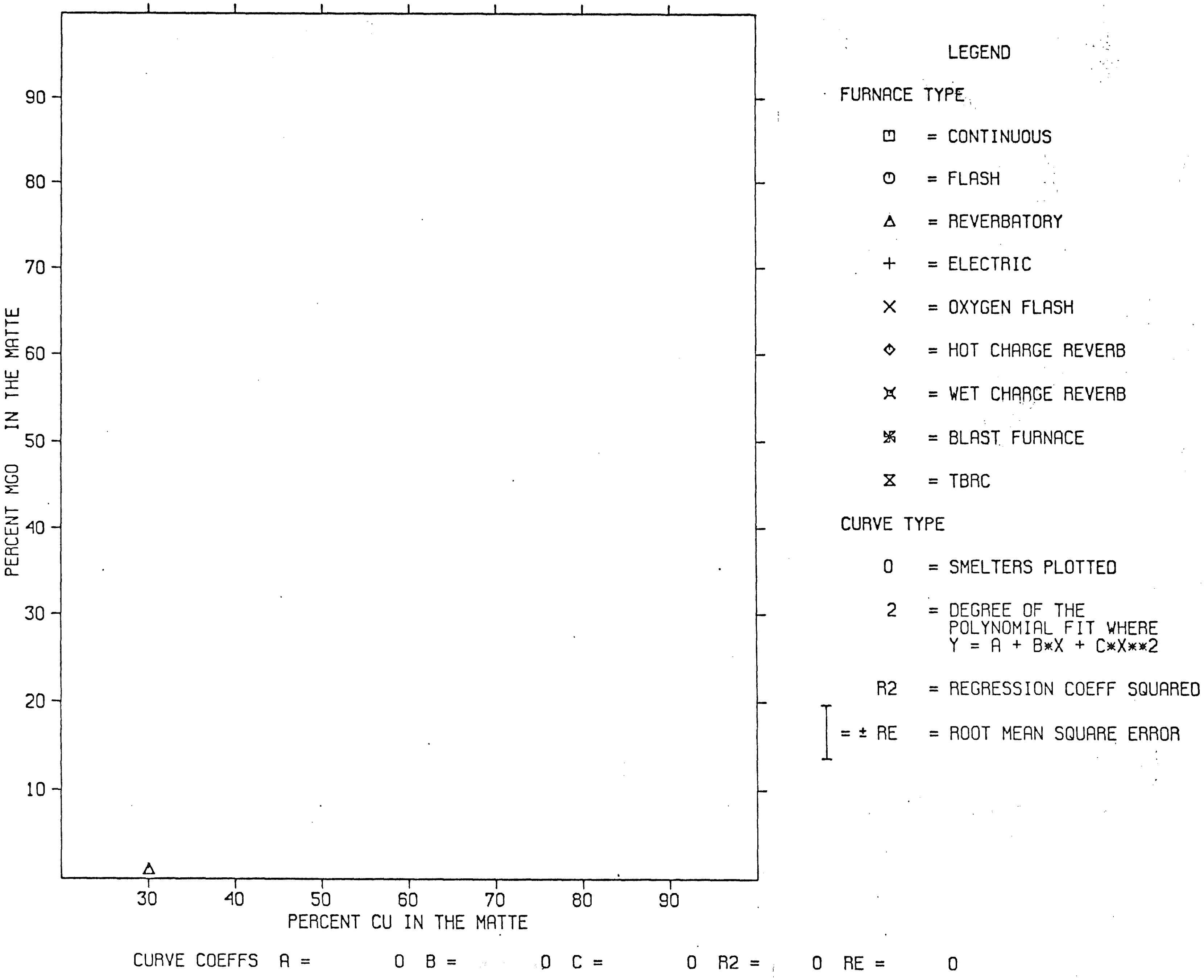
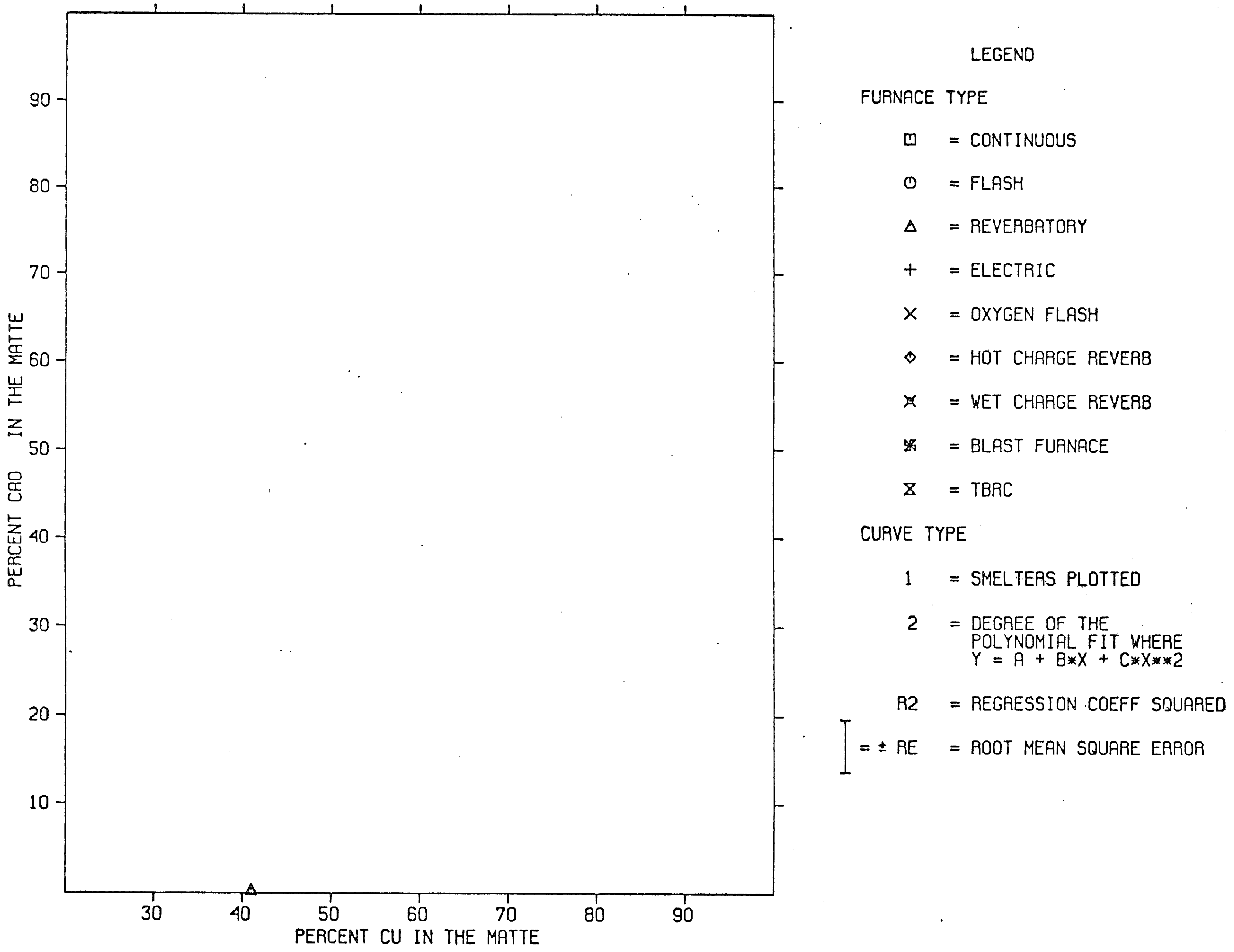


Figure 71.

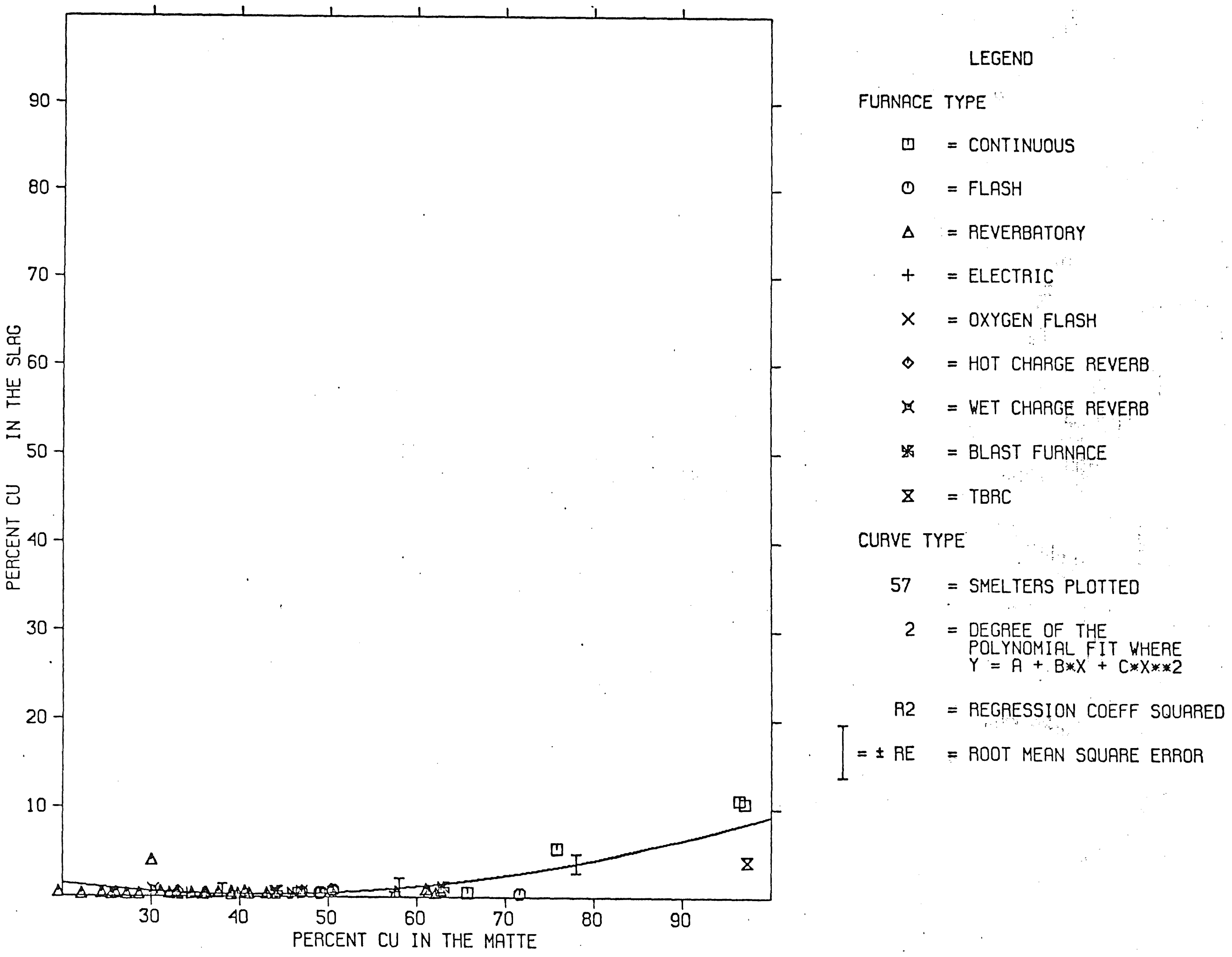
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = 0 B = 0 C = 0 R2 = 0 RE = 0

Figure 72.

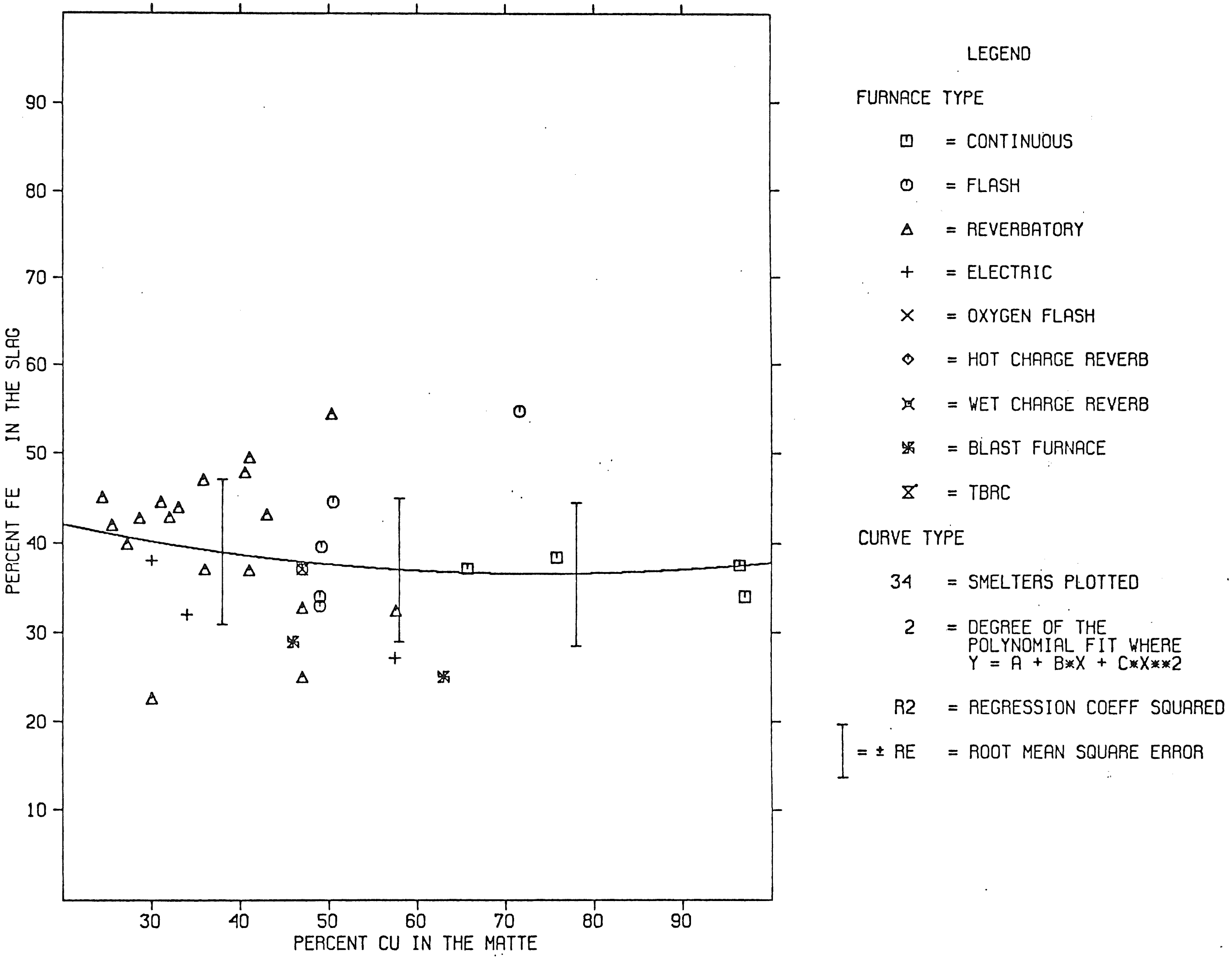
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = .1433 B = -.1098 C = .0258 R2 = .7431 RE = .1076

Figure 73.

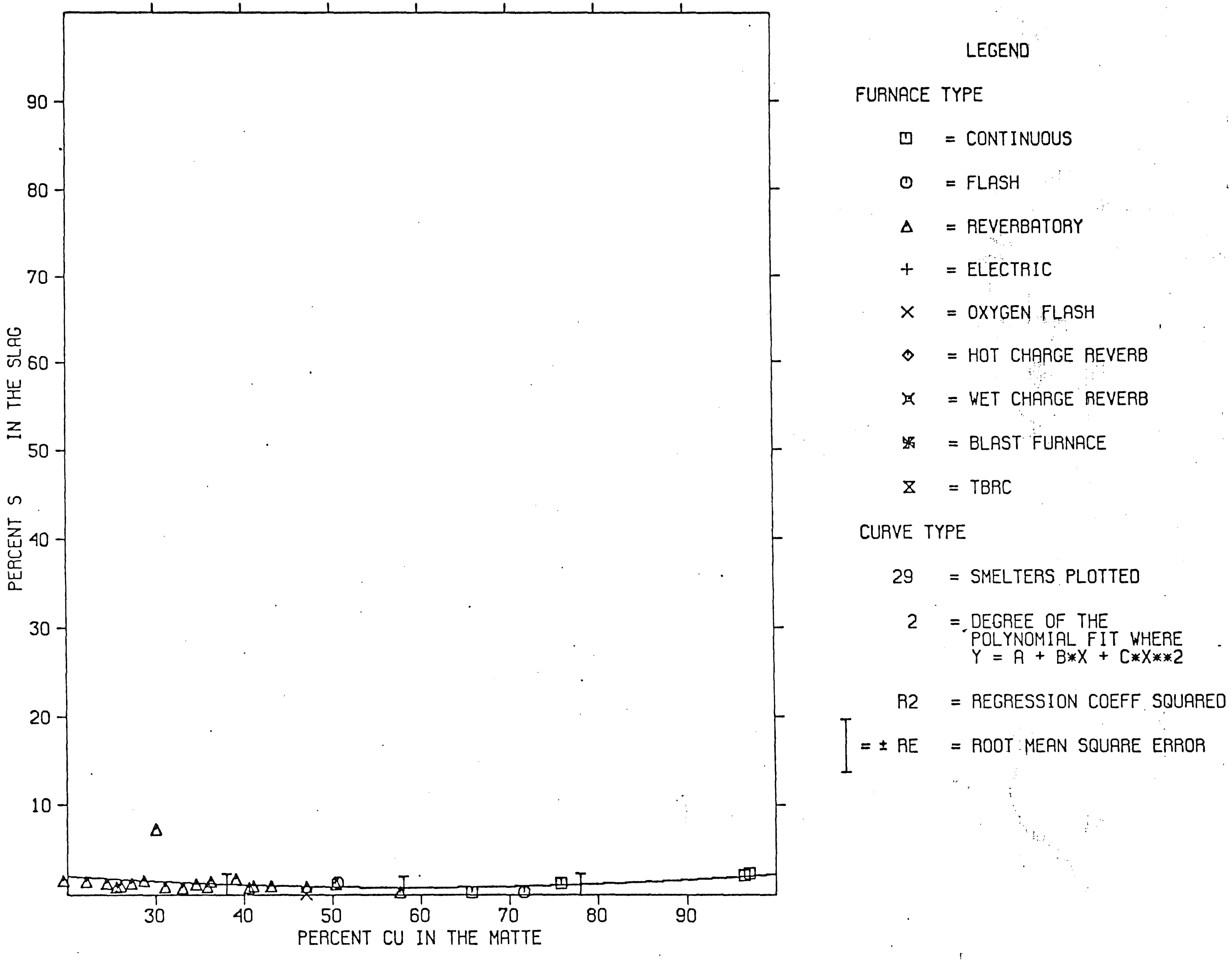
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = 4.2061 B = -.2065 C = .0191 R2 = .0304 RE = .8016

Figure 74.

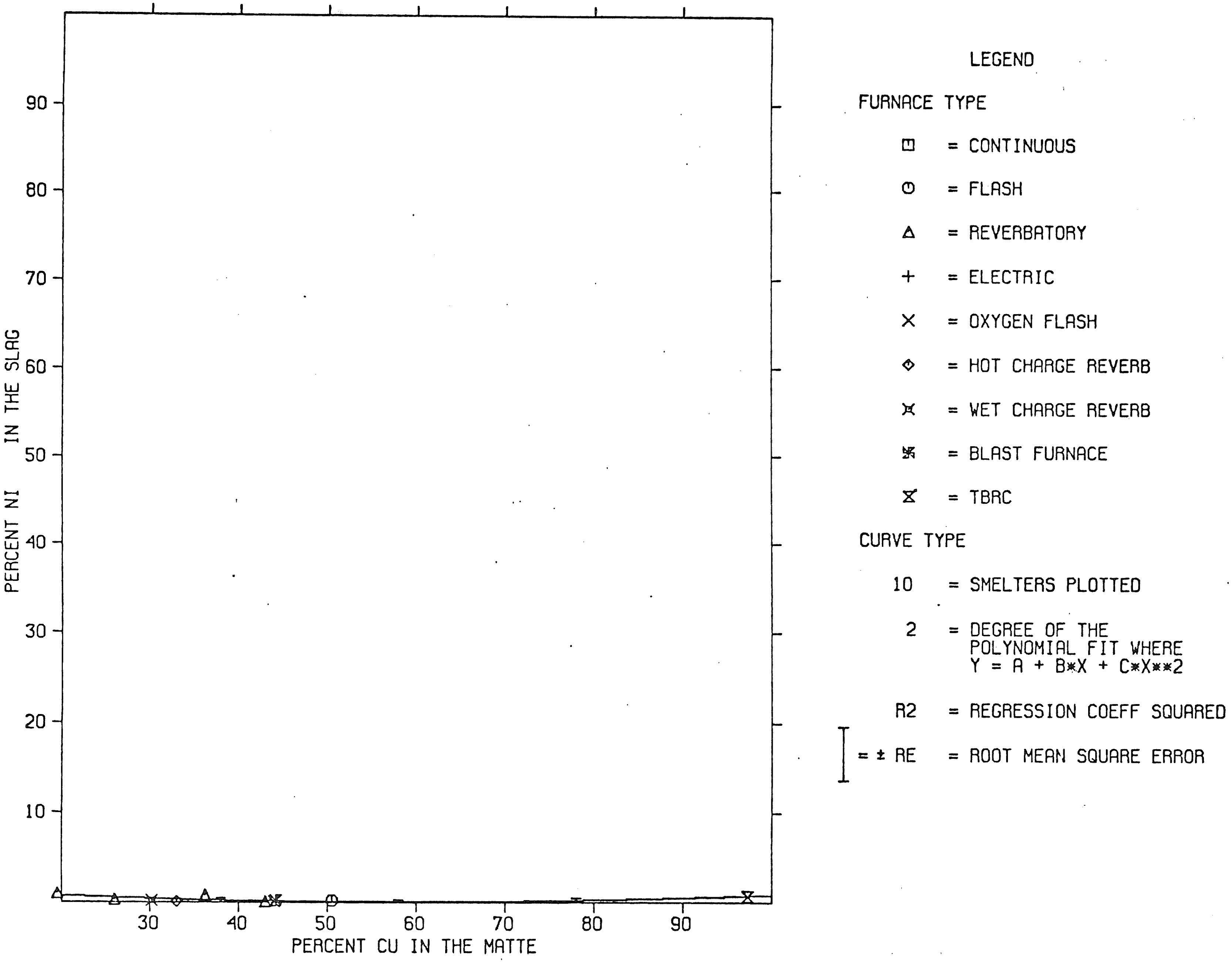
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = .2123 B = -.0675 C = .0088 R2 = .1141 RE = .1230

Figure 75.

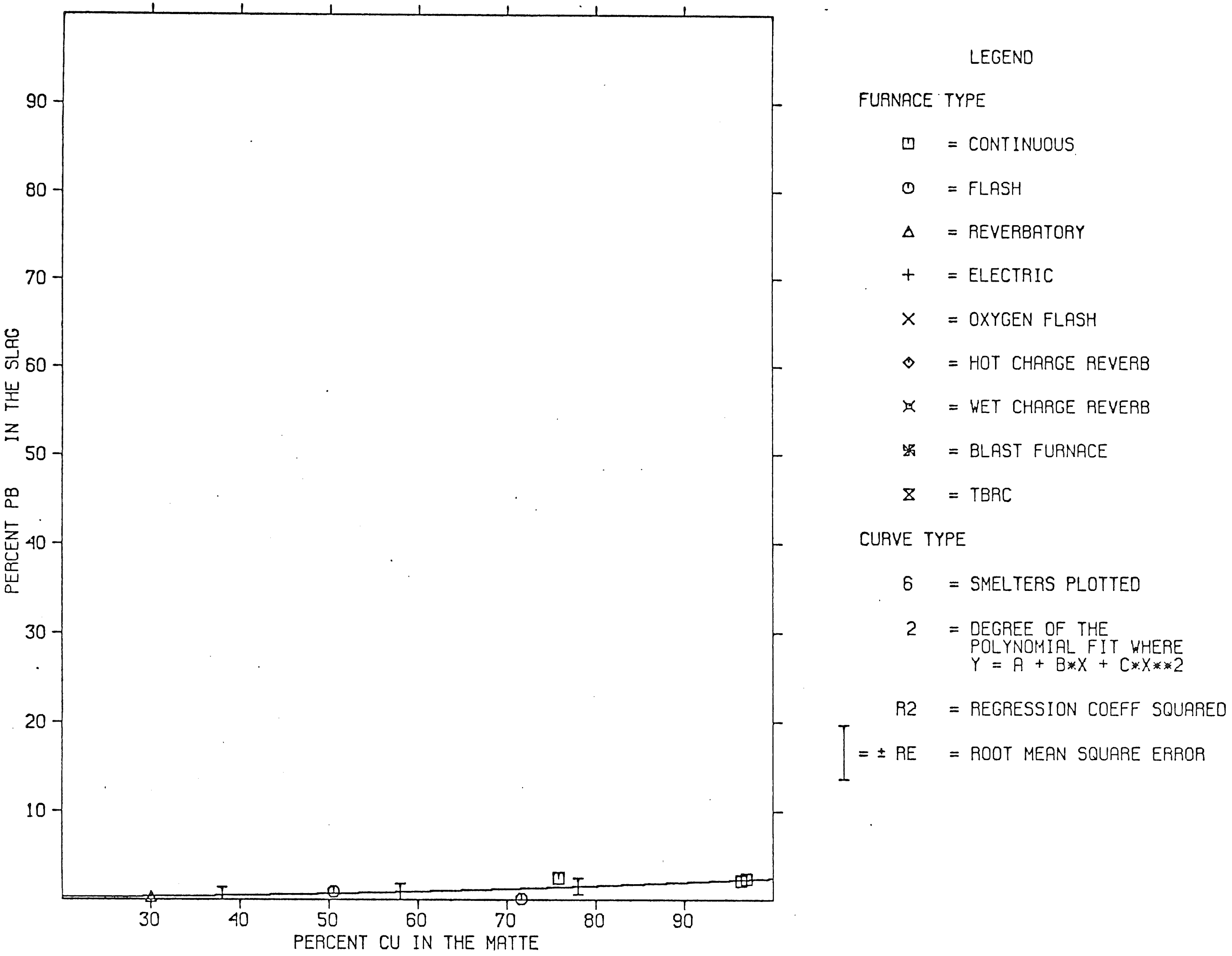
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = .0739 B = -.0348 C = .0045 R2 = .5148 RE = .0266

Figure 76.

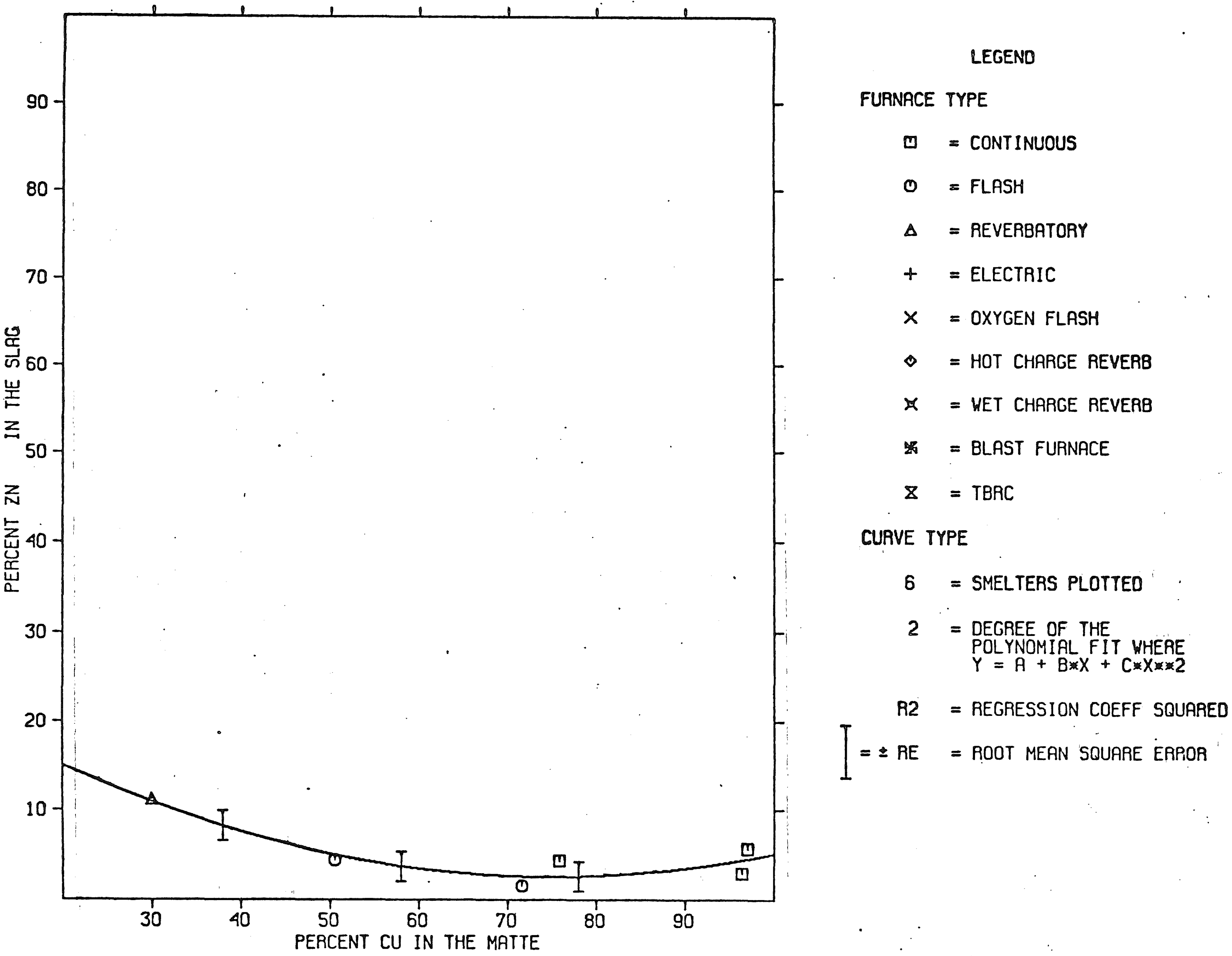
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = .0277 B = .0056 C = .0026 R2 = .5491 RE = .0900

Figure 77.

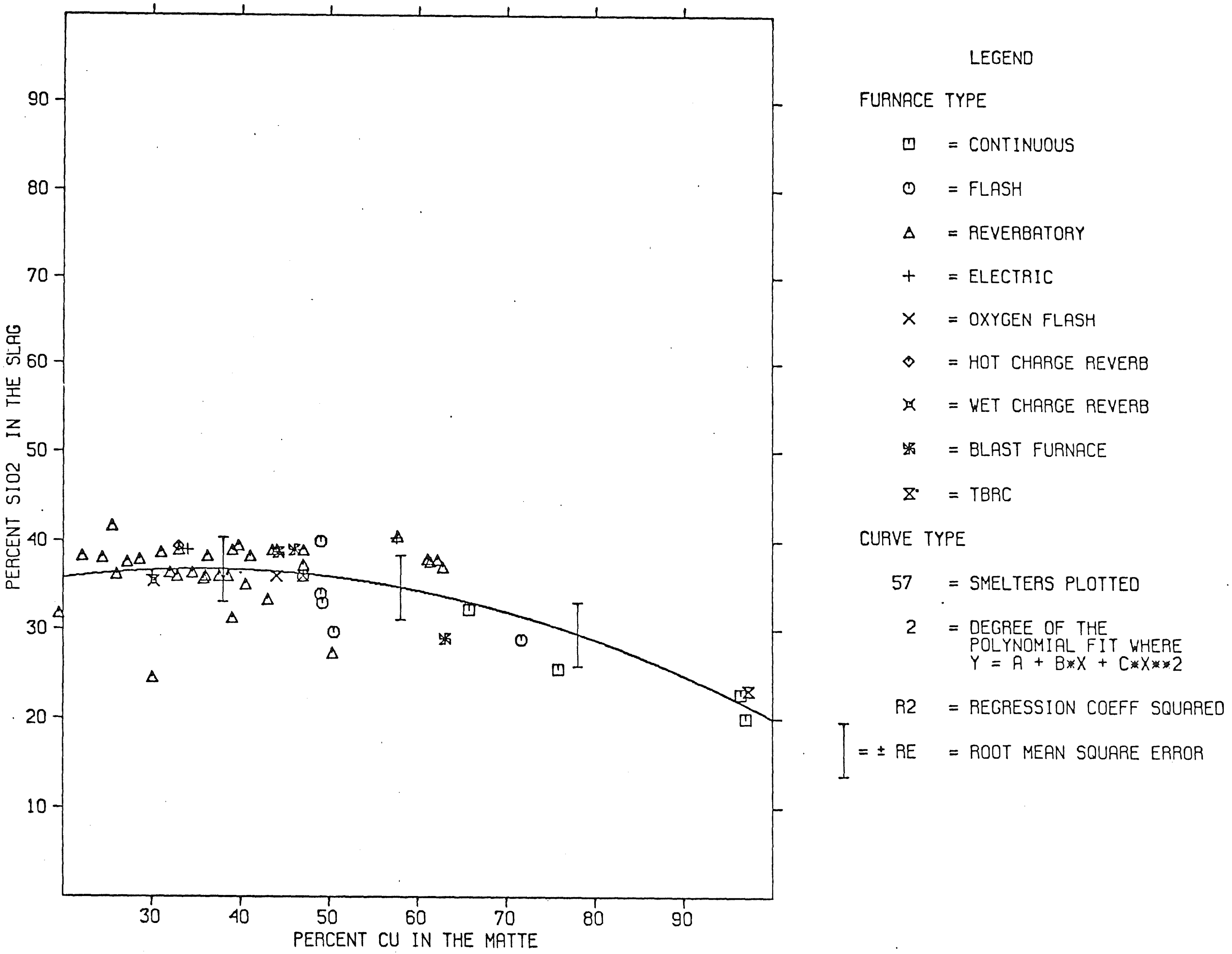
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = 1.5061 B = -.4531 C = .0412 R2 = .8516 RE = .1651

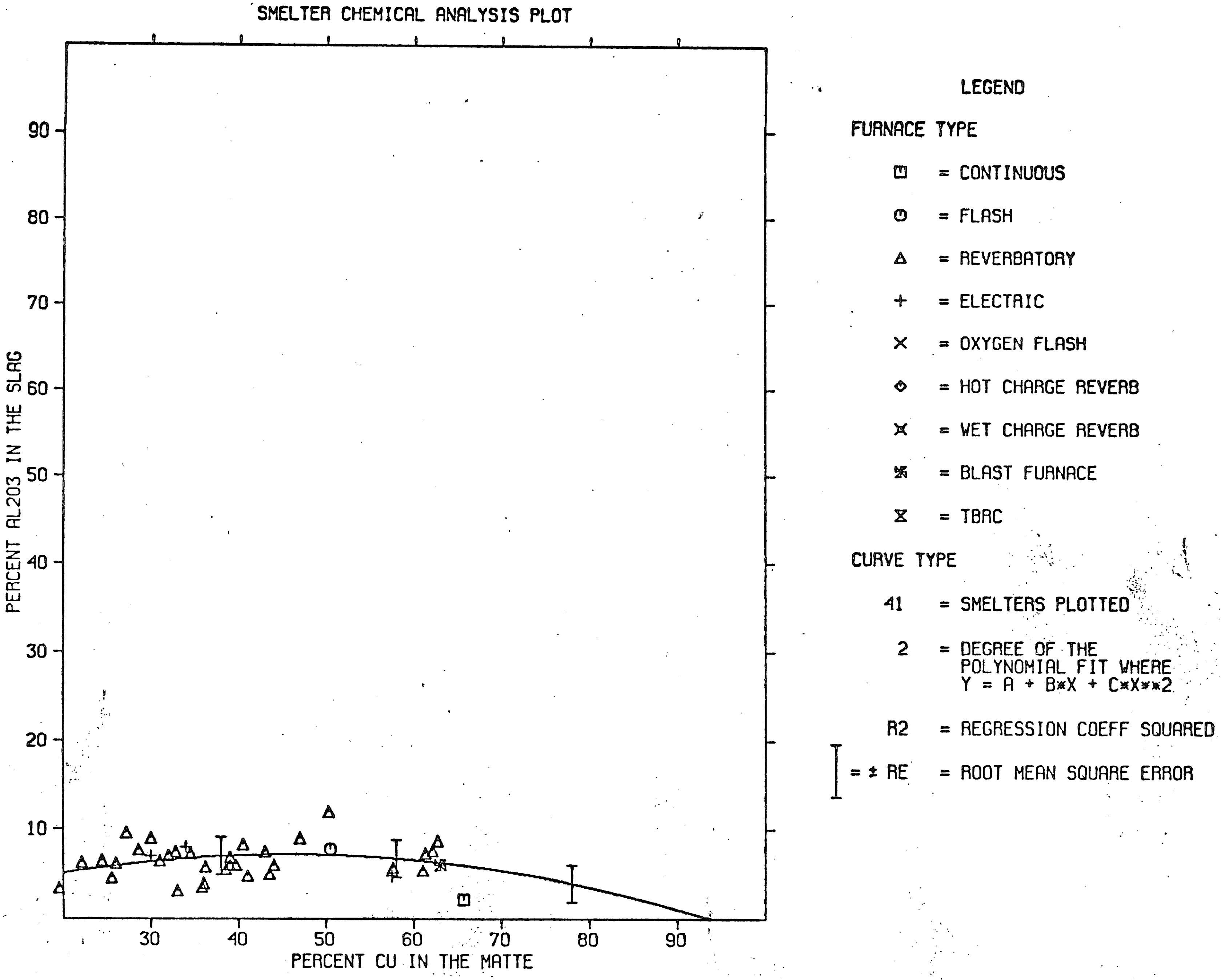
Figure 78.

SMELTER CHEMICAL ANALYSIS PLOT



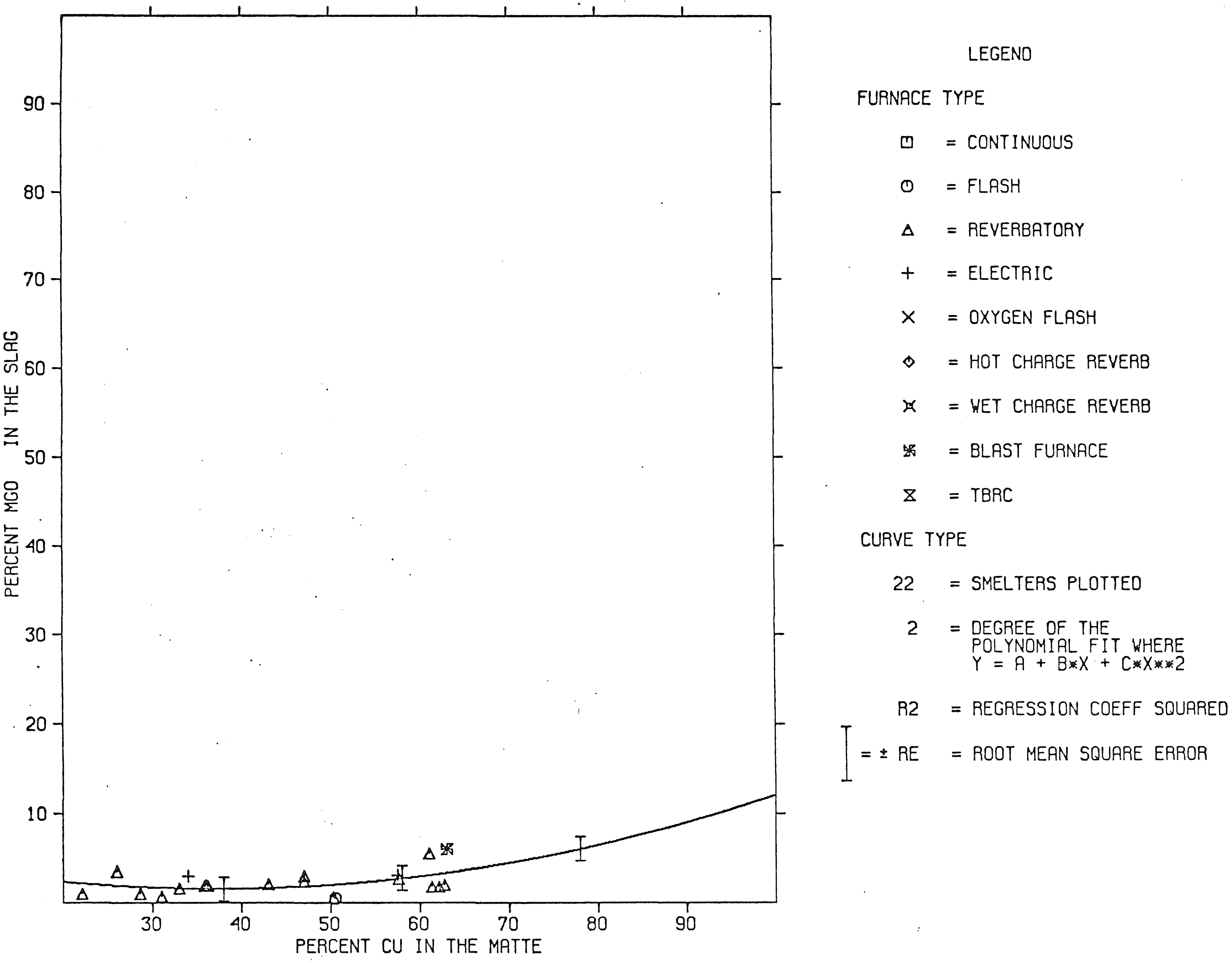
CURVE COEFFS A = 3.5772 B = .1282 C = -.0407 R2 = .4899 RE = .3630

Figure 79.



CURVE COEFFS A = .5087 B = .1618 C = -.0313 R2 = .0626 RE = .2089

Figure 80.

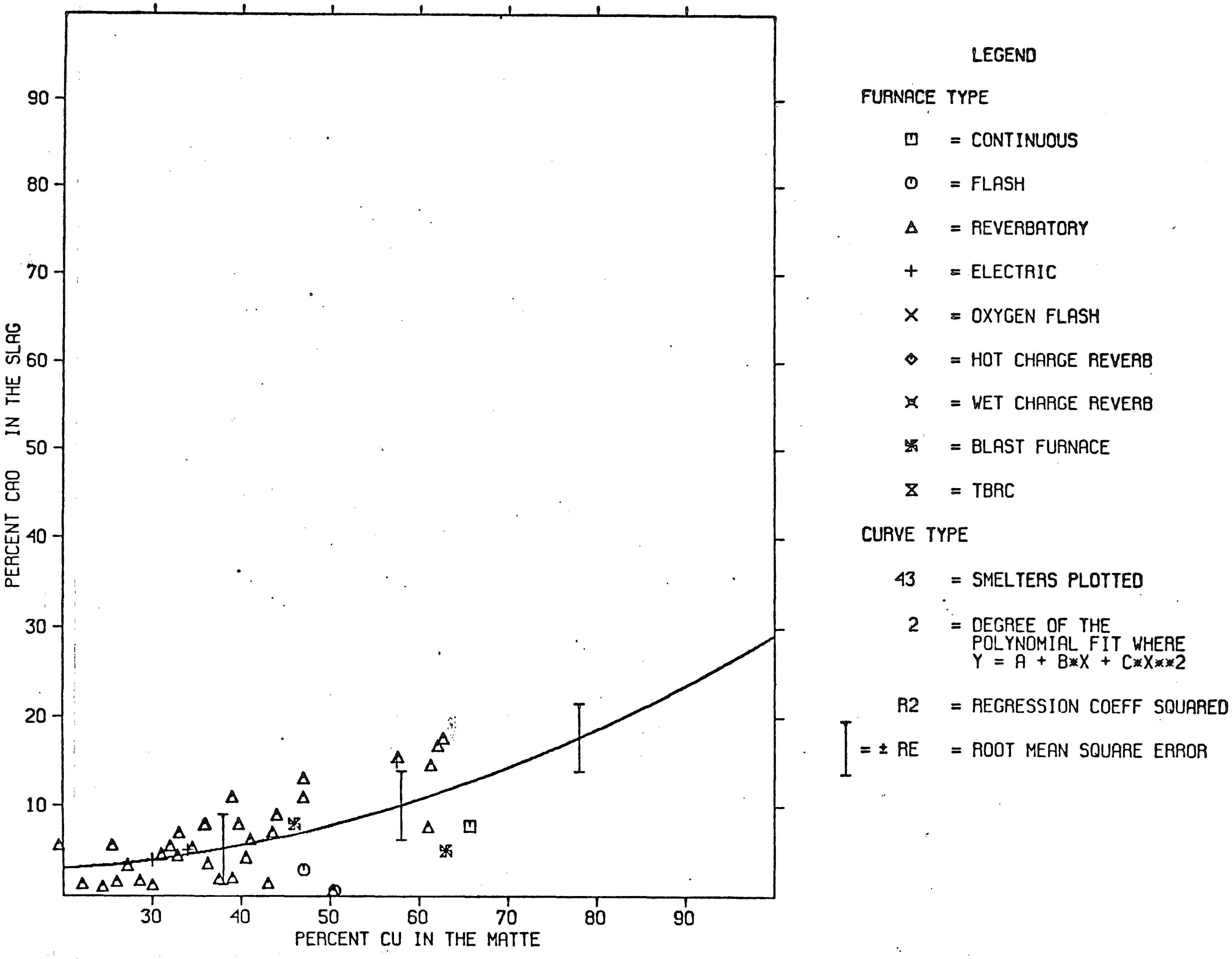


SMELTER CHEMICAL ANALYSIS PLOT

CURVE COEFFS A = .2378 B = -.0917 C = .0267 R2 = .2015 RE = .1351

Figure 81.

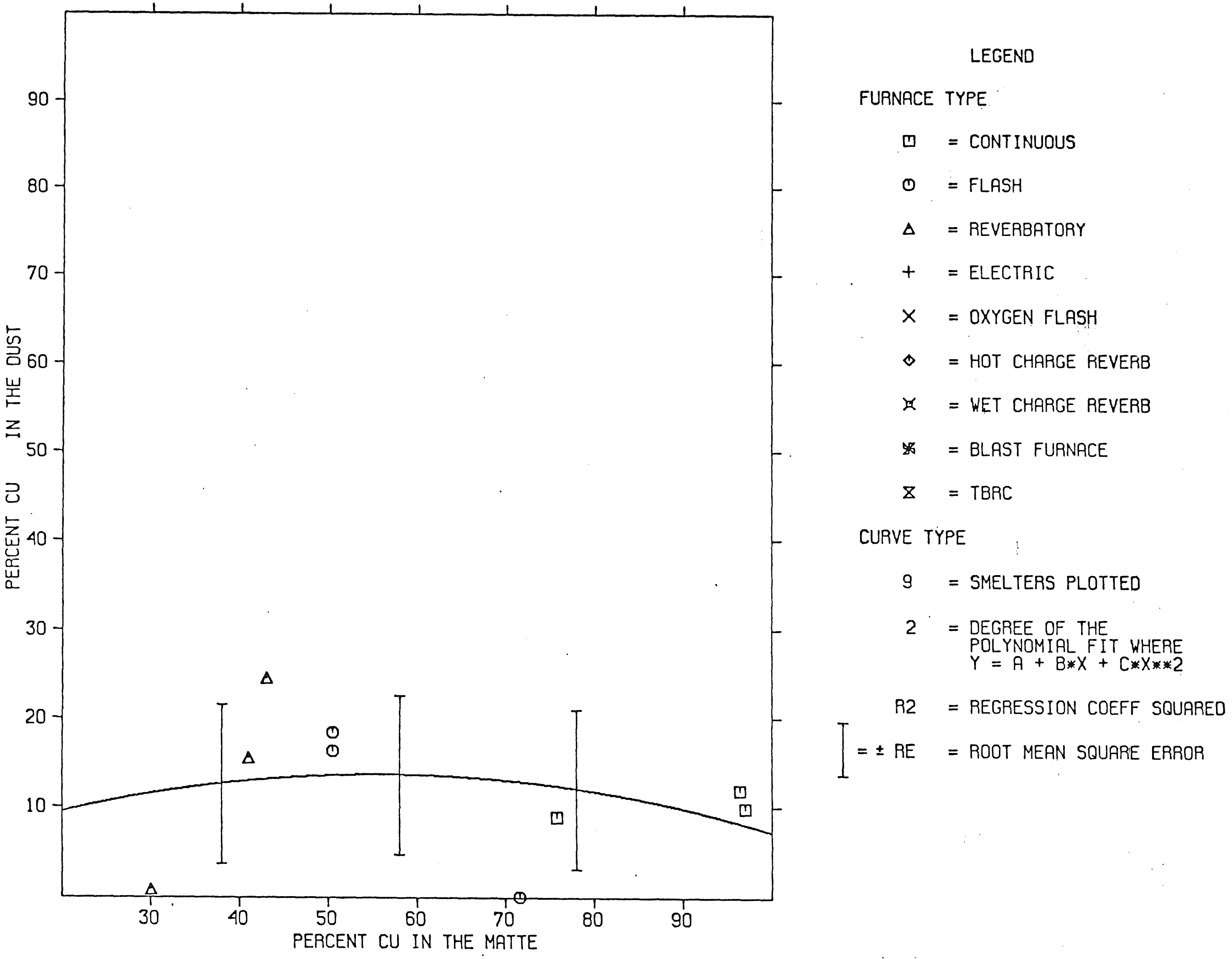
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = .3151 B = .0517 C = .0343 R2 = .3397 RE = .3851

Figure 82.

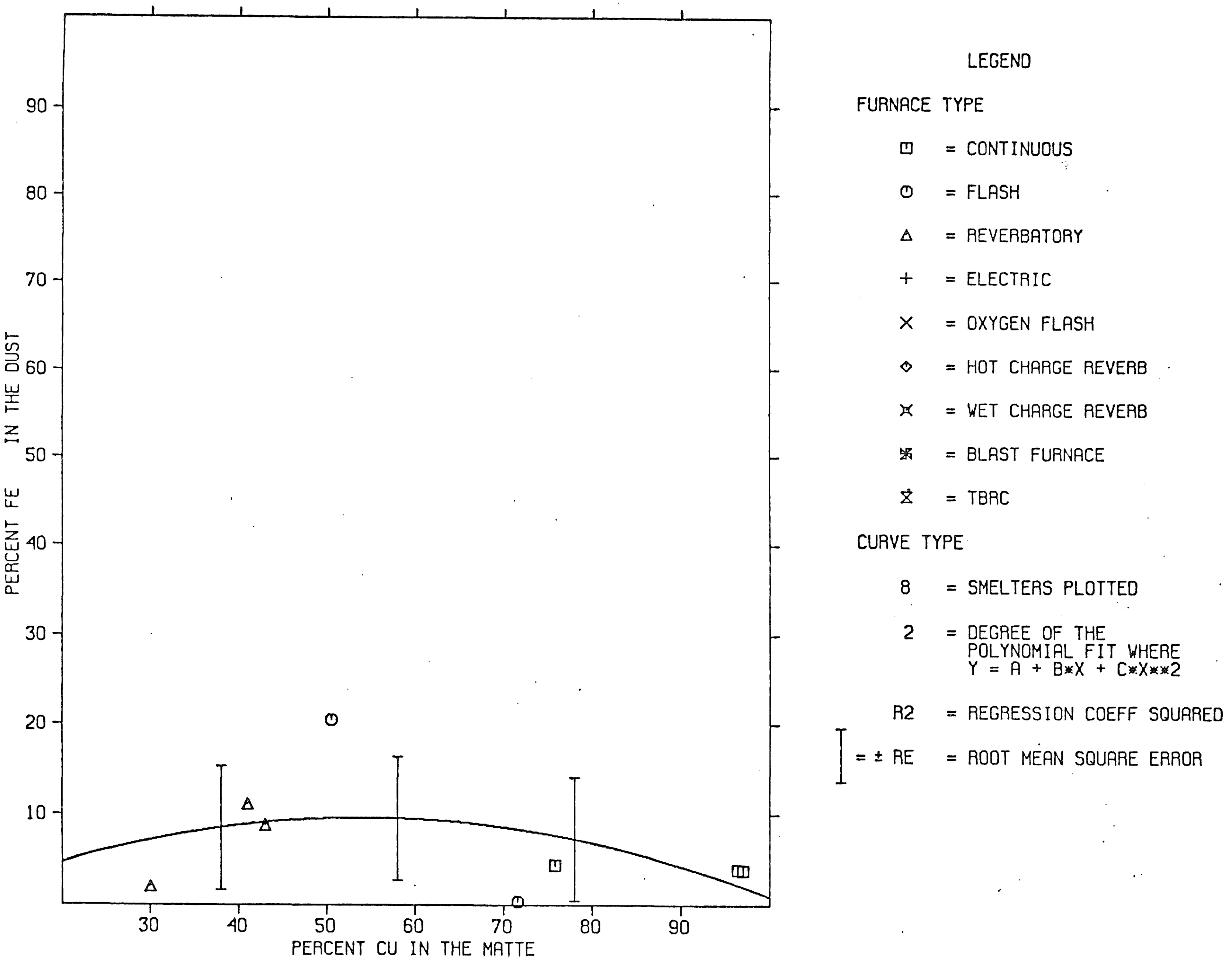
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = .9480 B = .2344 C = -.0327 R2 = .0730 RE = .8852

Figure 83.

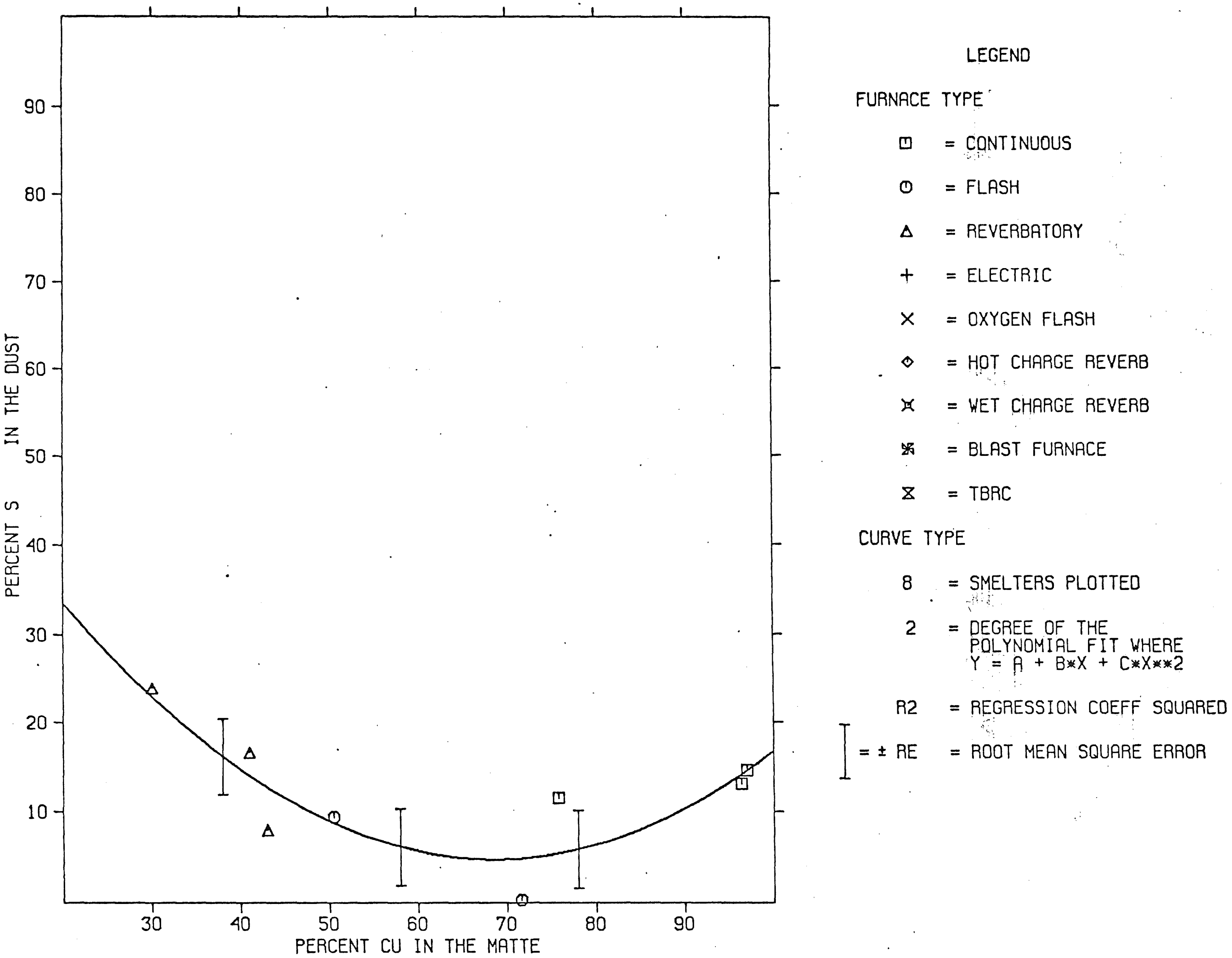
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = .4786 B = .2835 C = -.0414 R2 = .2164 RE = .6802

Figure 84.

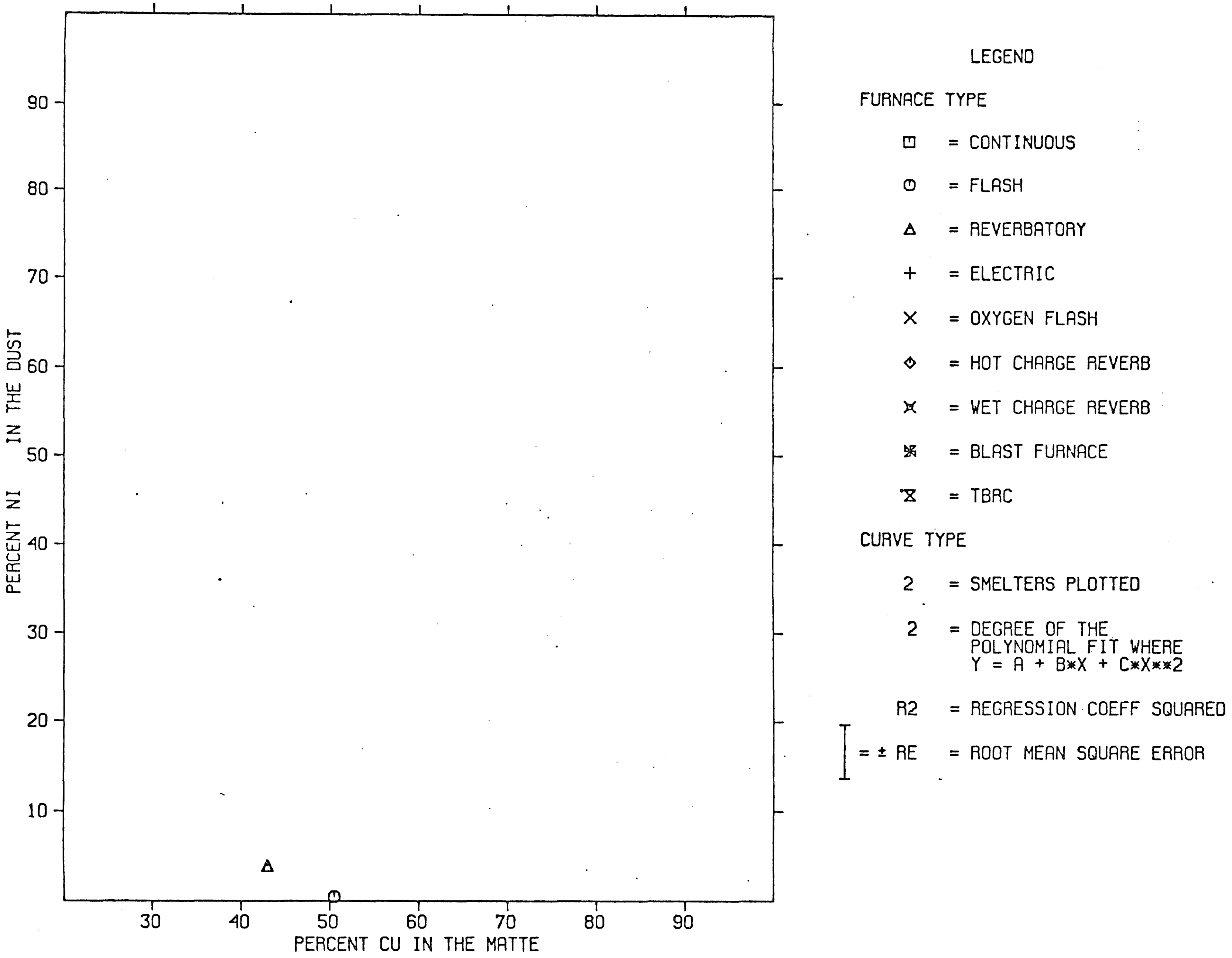
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = 3.3553 B = -1.1863 C = .1221 R2 = .7281 RE = .4240

Figure 85.

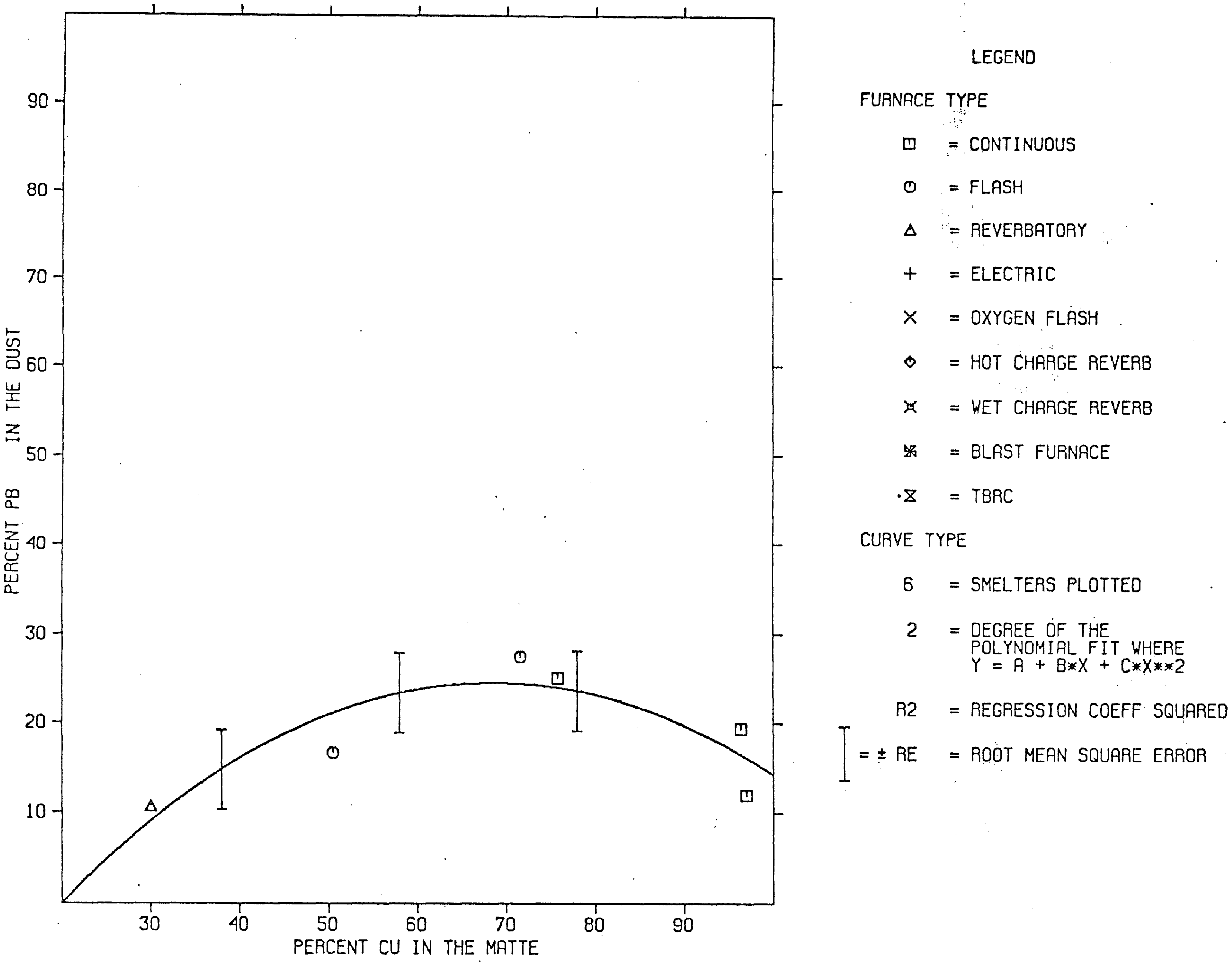
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = 0 B = 0 C = 0 R2 = 0 RE = 0

Figure 86.

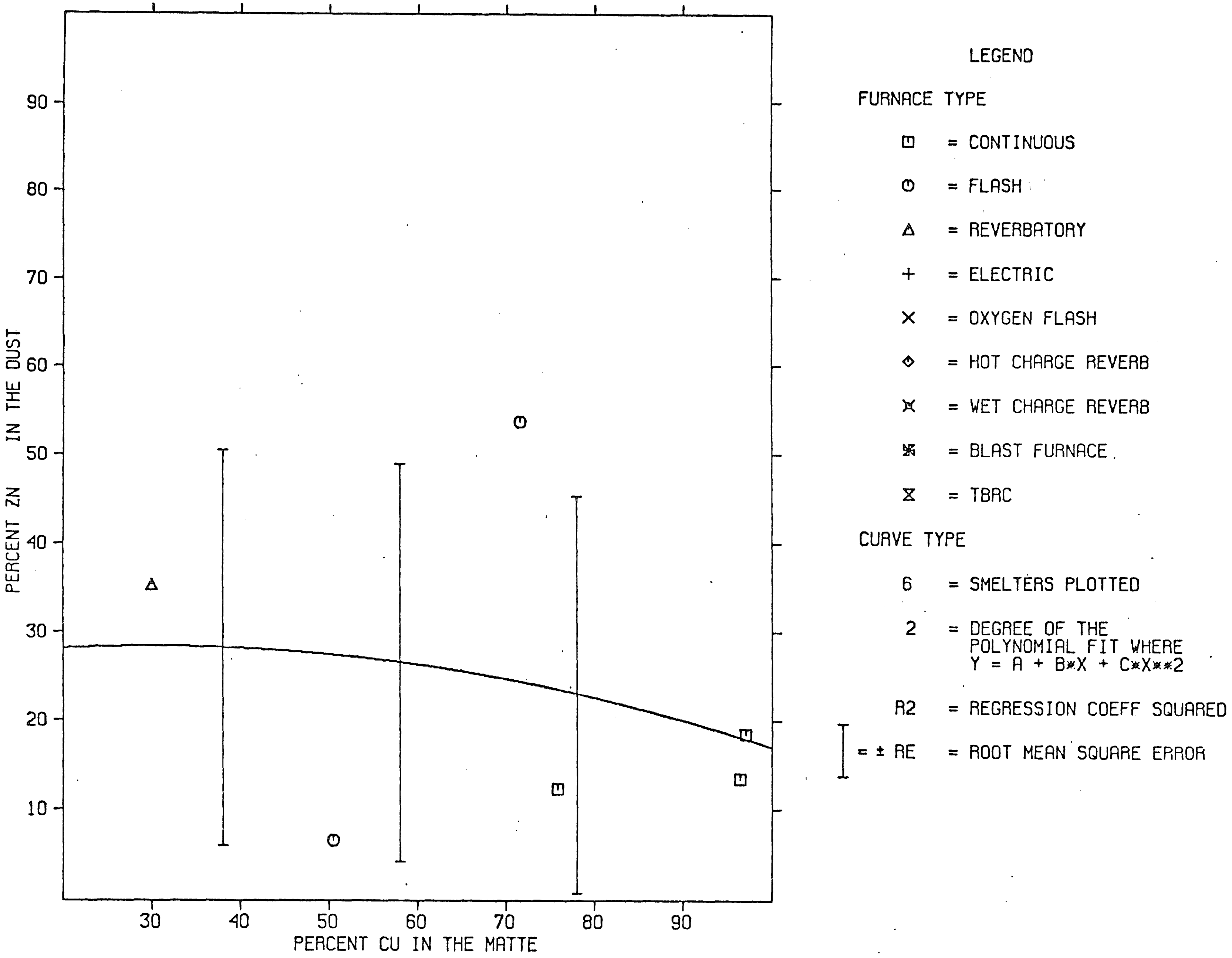
SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = .0028 B = 1.0063 C = -.1036 R2 = .7418 RE = .4441

Figure 87.

SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = 2.8125 B = .0496 C = -.0236 R2 = .0628 RE = 2.2274

Figure 88.

SMELTER CHEMICAL ANALYSIS PLOT

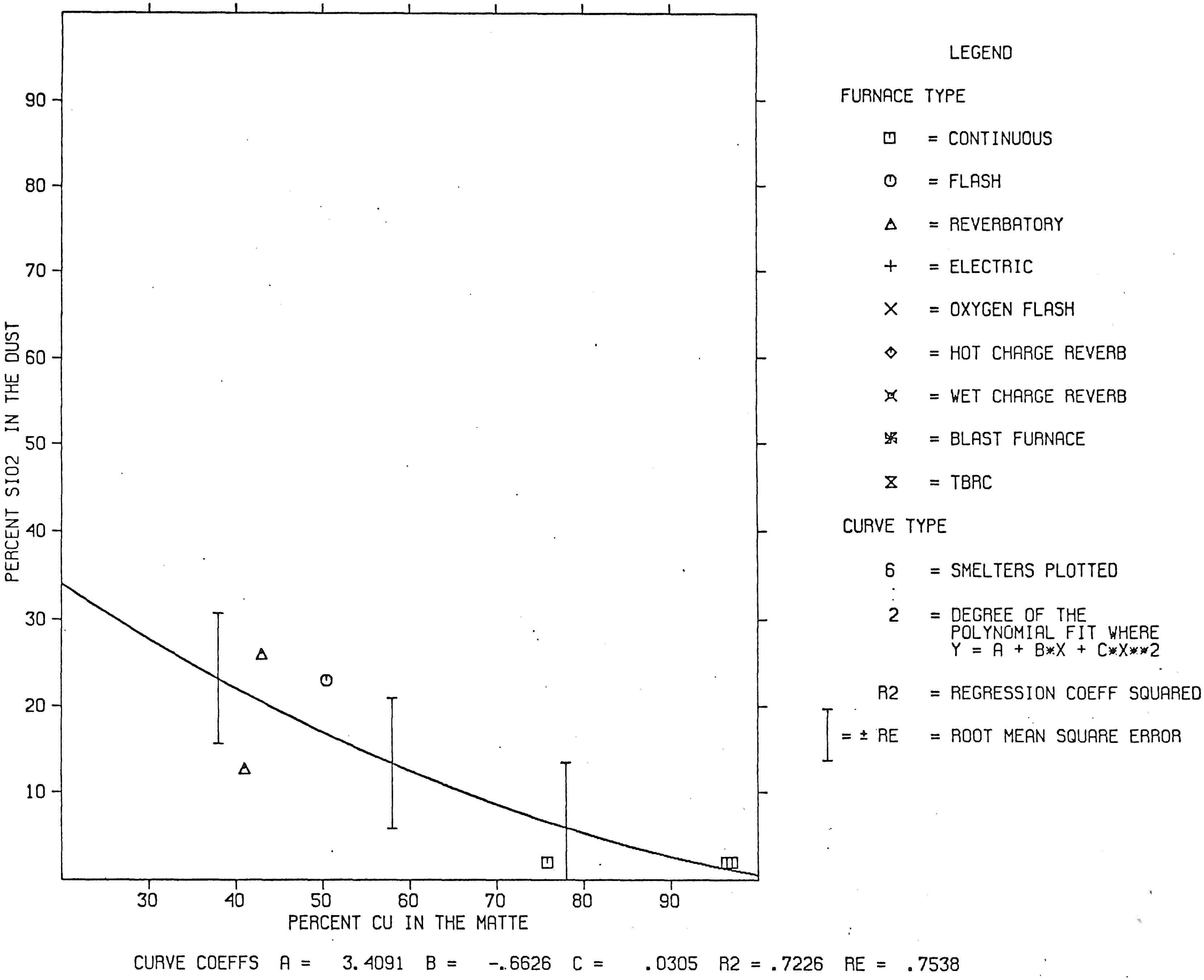
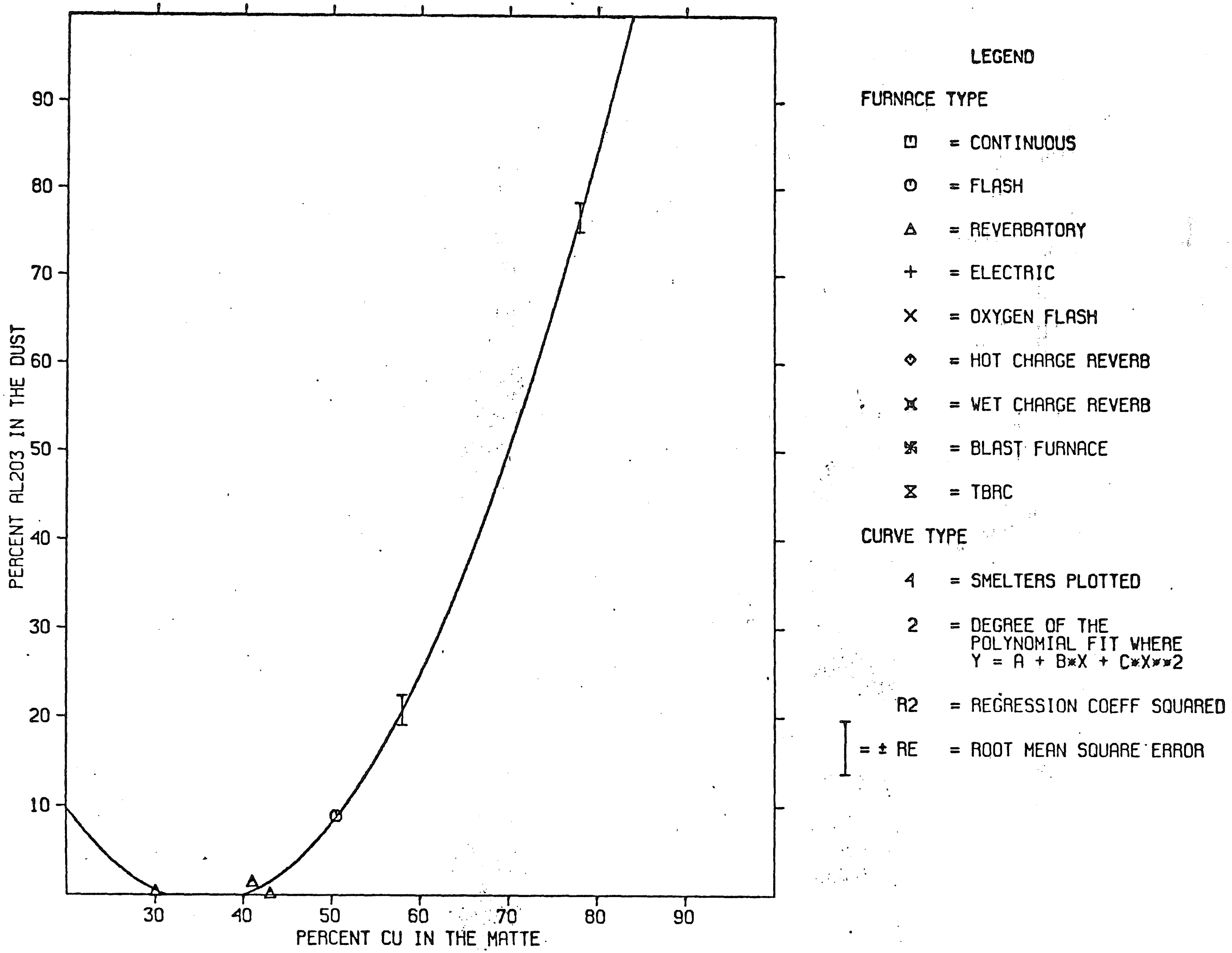


Figure 89.

SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = .9781 B = -1.3518 C = .4320 R2 = .9420 RE = .1706

Figure 90.

SMELTER CHEMICAL ANALYSIS PLOT

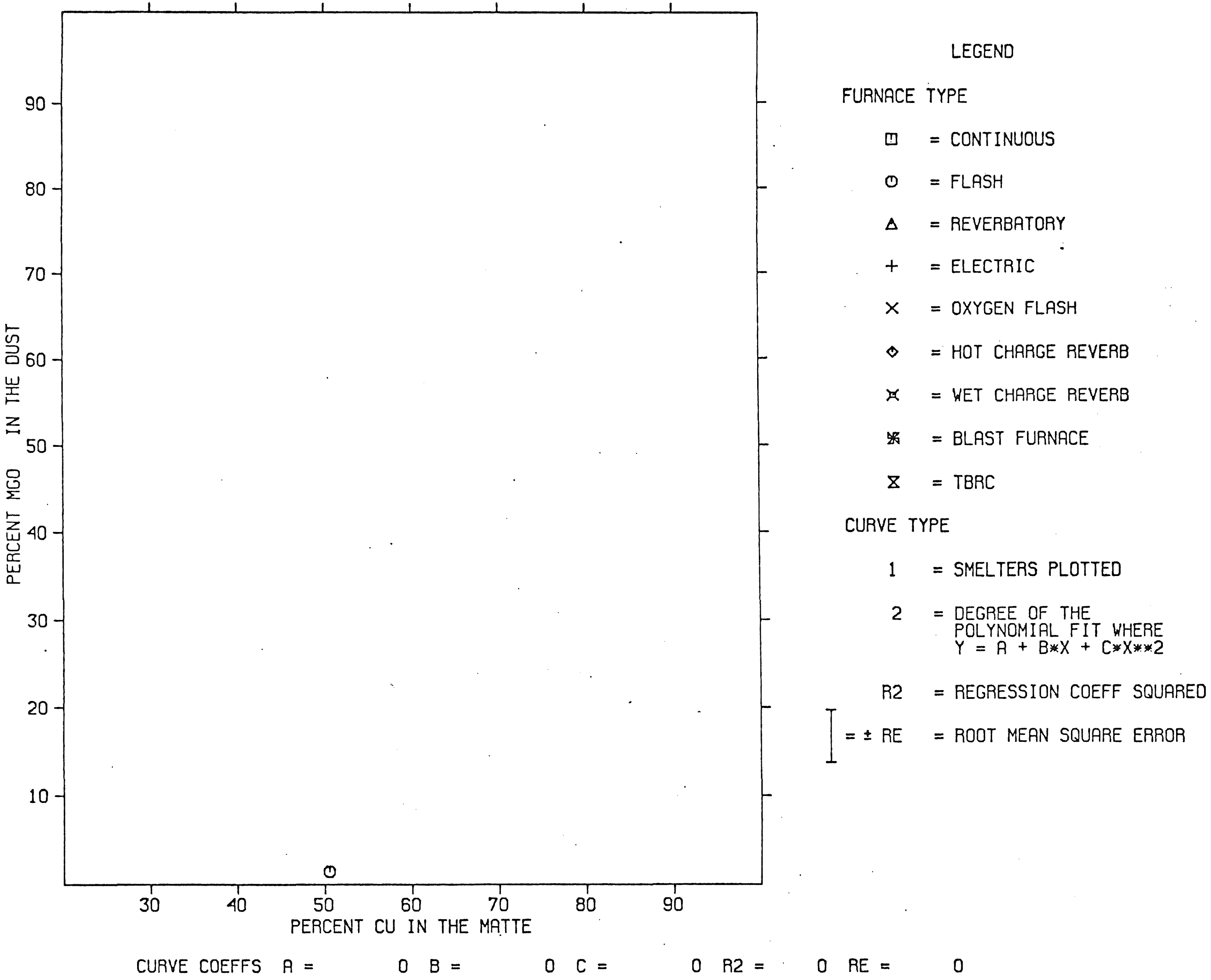
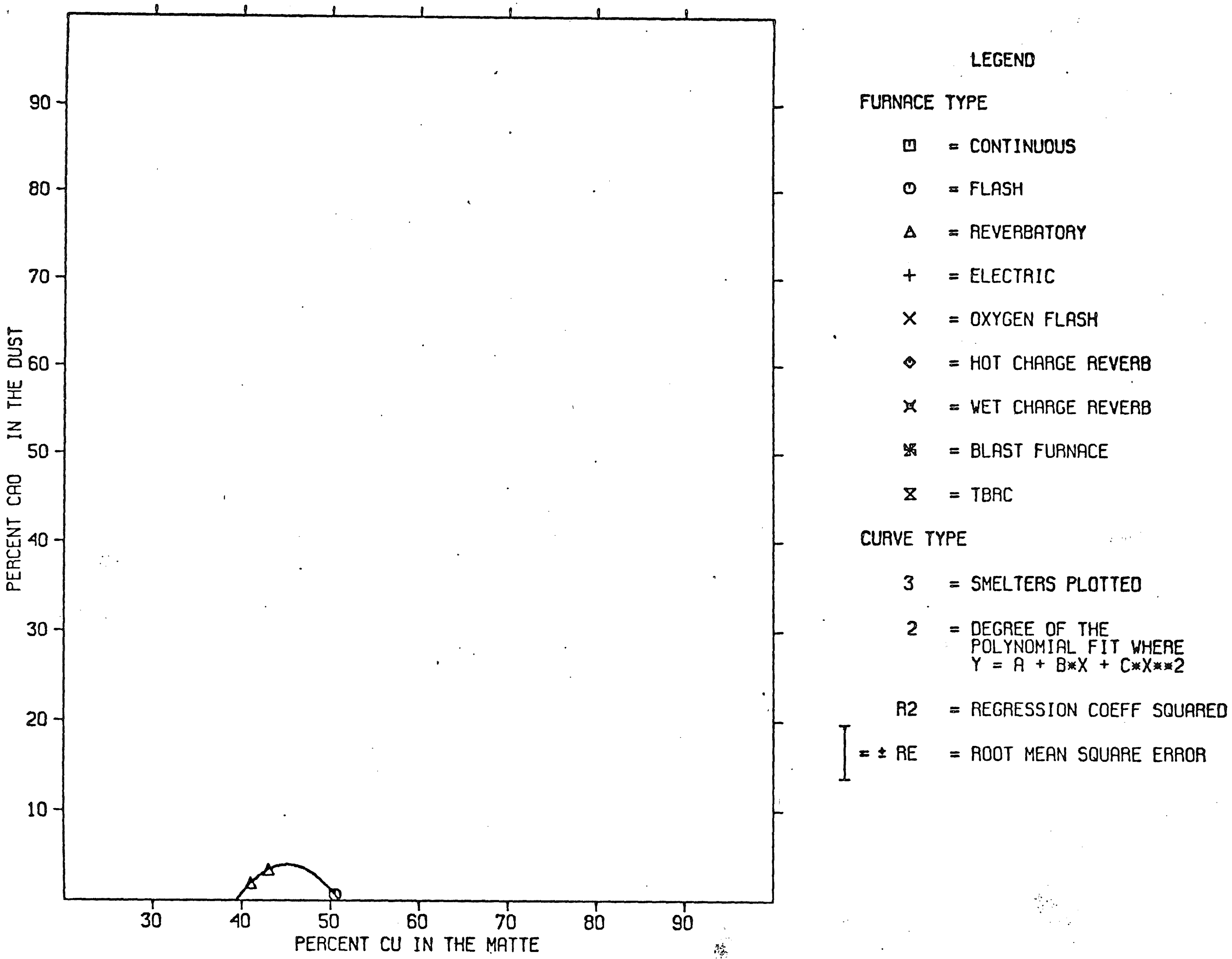


Figure 91.

SMELTER CHEMICAL ANALYSIS PLOT



CURVE COEFFS A = -7.1066 B = 5.9713 C = -1.1867 R2 =1.0000 RE = 0 however, indicate the questionable reliability of these curves, and suggest that only general trends be deduced, and that specific metallurgical balances not be hypothesized.

Figures 62 through 91 indicate the change in chemistry of the matte, slag, and dust fractions, also, as a function of matte grade. These curves, assuming that they are accurate, are indicative of elemental distribution. As the matte grade increases:

- the percent iron in the matte decreases
- the percent sulfur in the matte decreases
- the percent nickel in the matte decreases
- the percent copper in the slag increases
- the percent iron in the slag slowly decreases
- the percent sulfur in the slag is unchanged
- the percent nickel in the slag is unchanged
- the percent copper in the dust slowly increases to a maximum value at a 50 percent copper matte grade
- the percent iron in the dust slowly increases to a maximum value at a 50 percent copper matte grade
- the percent sulfur in the dust decreases to a minimum value at a 70 percent copper matte grade

The error bars again indicate the questionable nature of the curves, and suggest that only general inferences be made. As more detailed operational data is gathered, perhaps these curves will become more accurately defined.

Pollution Control

Figure 92 compares a variety of natural and man-made particulates by their size ranges. Figure 93 illustrates the general guidelines which are used in the preliminary evaluation of control equipment.

Table 20 lists the particular choices made by operators for specific applications in a variety of industries. For many applications, the choice of the type of collector is well established by operational precedent.

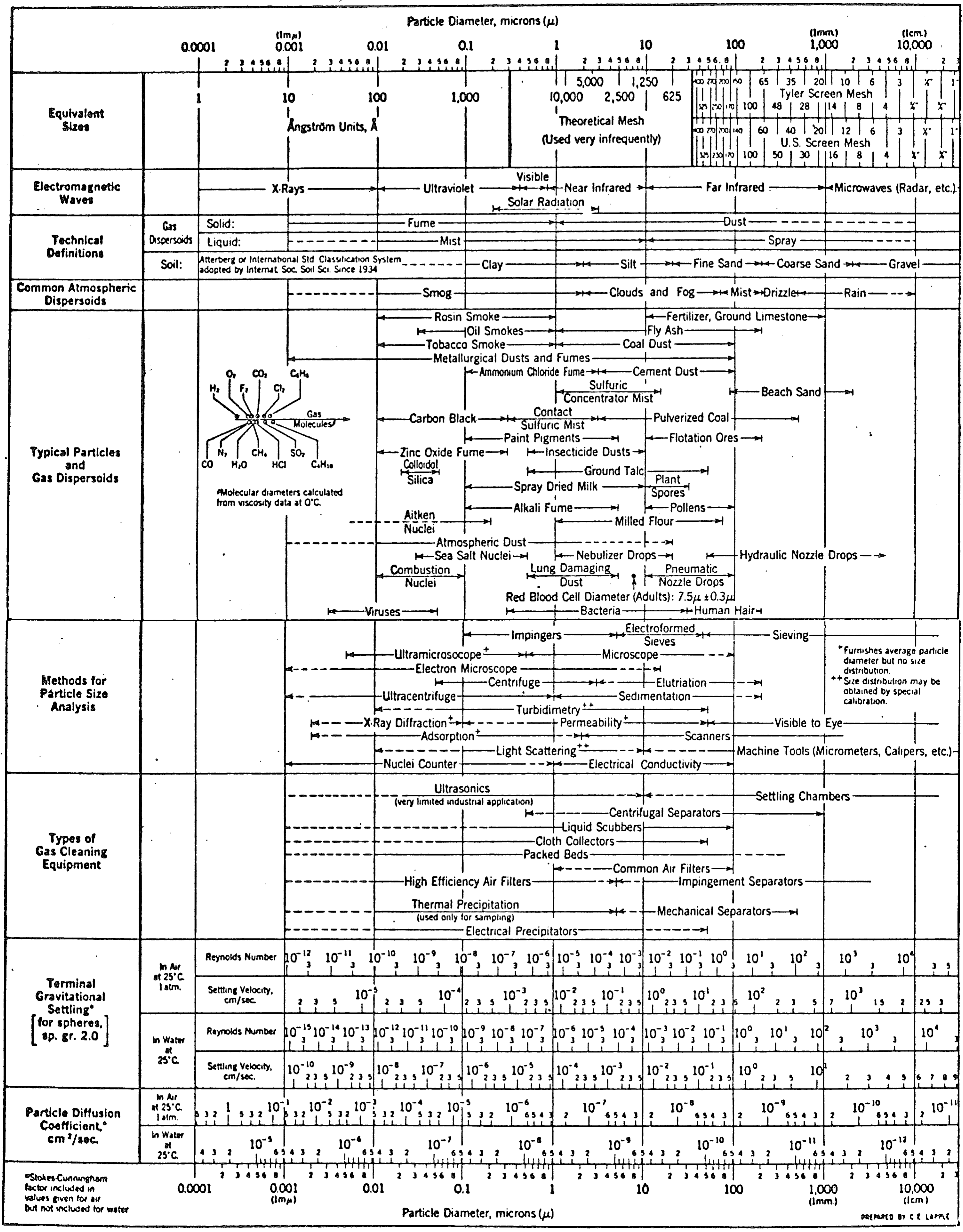
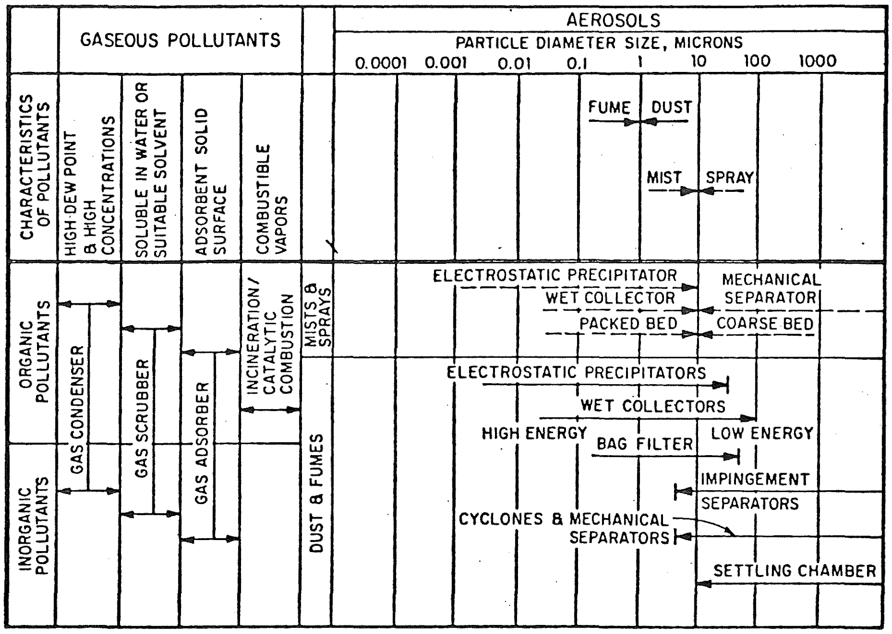


Figure 92. Characteristics of Particles and Particle Dispersoids. (Source: Ross, 1972)



General Notes on Table

(1) The selection of equipment for removing aerosols from air streams depends on the properties of the particles.

(a) Low concentration mists can be collected with an electrostatic precipitator, a wet collector, a high density fibrous filter, or a packed bed mist eliminator.

(b) Sprays can be removed with mechanical separators or coarse-packed beds.

(c) Fumes can be collected with high-energy wet collectors, bag filters, or electrostatic precipitators.

(d) Dusts can be removed from an airstream with wet collectors, inertia or impingement separators.

(e) Coarse particles can be removed from exhausts with settling chambers or cyclones.

(f) A primary cleaner, cyclone, or settling chamber should be considered for exhaust loadings greater than 10 grains/ft^3 to reduce the load on the principal abatement equipment.

(g) Abrasive particles can be collected with wet collectors or bag filters operated at reduced flow velocities. Electrostatic precipitators can be used, but the unit must be sized to reduce erosion wear.

(h) Adhesive or hygroscopic materials are best collected with wet collectors. Bag filters, electrostatic precipitators and cyclones are not effective control devices.

(i) Dusts with low resistivity characteristics are effectively collected with electrostatic precipitators.

(2) The selection of control equipment for gaseous pollutants is dependent on the chemical nature of the contaminant. Some guidelines for organic pollutants can be summarized as follows:

(a) Vapors present in high concentrations and with high dewpoint temperatures can be removed by condensation either by direct cooling or compression.

(b) Highly soluble organics can be removed from the exhaust by scrubbing with liquid or a suitable solvent.

(c) Pollutants which have molecular weights higher than the normal components of air can be removed by adsorption. Generally, adsorption is practical for removal of organic vapors which can easily be evaporated from the adsorbent at low pressure steam temperatures.

(d) Combustible vapors can be burned in direct or catalytic units.

(e) Vapors which are highly explosive or flammable, and soluble in water, are best collected in wet systems.

(3) Inorganic gaseous pollutants can be removed from airstreams by condensation, scrubbing, and with adsorbents such as silica gel, alumina, or activated carbon. The use of adsorbents for inorganics has limited application so that scrubbing and sometimes condensation remain the principal treatment methods.

Figure 93. Control Equipment Guidelines. (Source: Ross, 1972)

INDUSTRY AND SPECIFIC APPLICATION	TYPES OF COLLECTORS*		
	Most used	*Also used*	*Seldom used*
ROCK PRODUCTS			
Wet process cement kiln	MC, EP	CC, WS	FF
Dry process cement kiln	MC, EP, FF	CC	WS
Cement mills	FF	EP, MC	WS, CC
Cement silos	FF		
Raw material preparation	MC, FF	EP	WS, CC
Asphalt plant dryer	MC, WS	EP, FF	CC
Coal drying	MC, WS		FF, EP
Coal mills	FF	MC, WS	EP, CC
Alumina kiln	MC, FF, EP	WS	CC
Bauxite kiln	MC, EP	FF, WS	CC
Gypsum kiln	MC, FF, EP	WS	CC
Magnesium oxide	MC	FF, WS, EP	
Lime kiln	MC, EP	FF, WS	
Phosphate kiln	MC, WS, EP	FF	
Finishing	FF	MC, EP	WS, CC
IRON AND STEEL			
Blast furnance	WS, EP	MC	FF
Open hearth	WS, EP	FF	MC, CC
Basic oxygen furnance	WS, EP		
Electric furnance	FF, WS	EP	MC, CC
Sintering	MC, FF, EP	WS, CC	
Cupolas	WS	MC, FF, EP	CC
Ore beneficiation	MC	FF, EP, WS	
Ore pellitizing	MC	WS, EP	
Ore roasting	MC, WS, EP		FF, CC
Taconite	MC	FF, WS, EP	CC
Pyrites roasting	MC, WS, EP		FF, CC
Hot scarfing	WS, EP		MC, FF, CC
Coke ovens	EP	WS	MC, FF, CC
Ferro-manganese blast furnance	MC, EP	CC, WS	FF
Scrap preheating	WS		MC, FF, EP
MINING AND METALLURGY			
Ore preparation and beneficiation	MC, FF, WS	EP	CC
Aluminum pots	FF, WS, EP	MC	CC
Elemental phosphorous	EP	WS	FF, WS
Copper converter	EP	FF, WS, CC	MC
Copper reveratory furnace	EP	WS	MC, FF, CC
Copper roaster	MC, EP	WS, CC	FF
Ilmenite ore dryer	MC, EP	CC	FF, WS
Molybdenum	WS	EP	FF, MC, CC
Precious metals	FF, EP	MC, WS	CC
Lithium kiln	EP		
Lead furnace	FF, WS		MC, EP
Titanium dioxide	FF, WS	EP	MC, CC
Zinc cupola	EP	WS	
Zinc roaster	MC, EP	FF, WS	CC
Zinc smelter	FF, WS, EP		MC, CC
FOUNDRY AND INDUSTRIAL			
Ferrous cupola	WS	MC, EP	FF, CC
Non-ferrous cupola	FF, EP	MC, WS	CC
Foundry cleaning room	FF, WS	EP	MC, CC
Sand preparation and handling	FF	WS, EP	MC, CC
Ventilation	FF		MC, FF, EP
Machining	MC, FF	WS	EP
Feed and flour milling	MC, FF		WS, EP, CC
Lumber mills	MC	FF	WS, EP
Wood working	MC, FF	WS	EP
PULP AND PAPER			
Black liquor recovery furnace	EP	WS	MC, FF
Lime kiln	MC, WS	EP, CC	FF
Chemical dissolver tank	WS	MC, FF, EP	
ELECTRIC POWER—TYPE OF FUEL			
Stoker fired coal	MC	EP, CC, WS	FF
Spreader stoker fired coal	MC	EP, CC, WS	FF
Puverized coal fired	EP, CC	WS	FF, MC
Oil fired	MC	EP	FF, WS, CC
Natural gas fired	none	none	
Lignite fired	MC, EP, CC	WS	FF
Wood and bark fired	MC	WS, EP	FF, CC
Bagasse fired	MC	WS	FF, EP, CC
Fluid coke	MC, EP	WS	FF, CC
WASTE DISPOSAL INCINERATORS			
Apartment house	WS	EP	MC, FF, CC
Industrial (thermal oxidizer)	EP	WS	MC, FF, CC
Municipal	EP, WS	MC, CC	FF
CHEMICAL AND OIL REFINING			
Refinery cat cracker—reactor	MC	none	
—regenerator	MC	none	
—CO boiler	EP	MC, WS	FF, CC
Sulphuric acid mist	EP, WS	MC	FF, CC
Phosphoric acid mist	WS		
Nitric acid mist		WS	
Carbon black	FF	MC, EP, WS	CC
Oil shale distillation	MC, CC	EP	FF, WS
Raw material preparation	MC	WS, EP, FF	CC

*MC = mechanical collector, FF = fabric filter, WS = wet scrubber, EP = electric precipitator, CC = combined collector = MC + EP.

Table 20. Typical Use of Particulate Collectors. (Source: Ross, 1972).

Figure 94 is a generalized diagram showing some of the more popular alternatives in current and developing SO_2 control systems. While each system is unique in detail, many contain similar basic steps.

It is technically feasible to fix over 95 percent of the sulfur entering a pyrometallurgical smelter. This is particularly true if the smelting furnace is of the electric, flash, or in the future, the continuous copper-making type. The sealed Hoboken type of converter will also be advantageous for improving SO_2 collection. With extraordinary care it should be possible to prevent 99 percent of the sulfur entering the smelter from entering the atmosphere.

Sulfur dioxide is most easily fixed as H_2SO_4 and techniques are available to treat gases containing as low as 2 percent SO_2. The tail gases from even the best acid plants contain in the order of 0.1 percent SO_2 which can only be recovered by scrubbing with basic solutions. The products of these scrubber systems are small quantities of basic sulfates ($CaSO_4$, $MgSO_4$, Na_2SO_4) which are water soluble and hence not good for sulfur storage. They can, however, be recycled to the smelting furnace (after evaporation) to keep this part of the sulfur stream in a closed circuit.

Elemental sulfur is the best form for permanently storing sulfur. The production of elemental sulfur from gases containing 5 percent SO_2 is technically feasible but plants to date have not operated on gases containing less than 10 percent SO_2. The elemental sulfur plants have been able to fix only 90 percent of their input sulfur, but existing Claus Plant technology could raise this to 95+ percent.

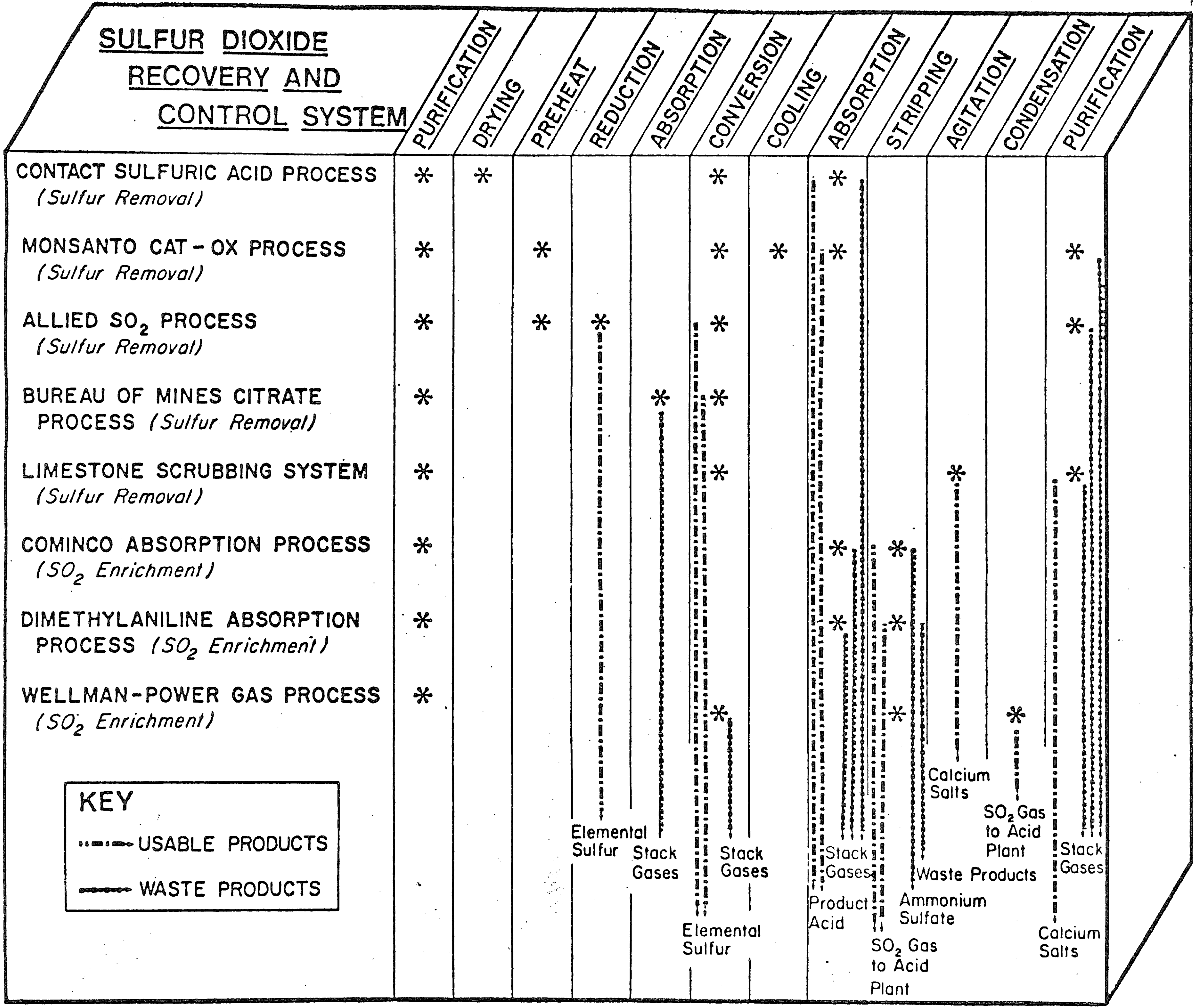


Figure 94. Major Sulfur Dioxide covery and Control Systems.

It is admitted that many of the sulfur treating processes are only in the testing stage and have not been proven in large scale situations. The most popular selection today is that of the contact sulfuric acid plant (a fully proven system) where sulfur dioxide gases are converted to sulfuric acid. This system might be considered in tandem with some sort of pyrometallurgical scheme should this type of metallurgy be utilized.

INTRODUCTION AND BACKGROUND

The copper and nickel ores of northeastern Minnesota generally occur as sulfides. Concentration of these elements as minerals in the orebody, however, is low, therefore, some type of treatment beyond physical concentration is necessary in order to recover the copper and nickel in a metallic form (a form which is usable in the manufacturing sector of the Minerals Industry).

Sulfides are not readily treated by hydrometallurgical methods (i.e. they are not easily leached),[4] consequently the vast majority of the extraction is by pyrometallurgical techniques starting with copper, copper-nickel, nickel-copper, or nickel concentrates. The extraction consists of the following steps:

1) concentration by froth flotation
2) roasting (optional), drying, smelting, and converting
3) refining

The most important aspects of steps 2 and 3 will be introduced here and each discussed in some detail later. The format for the detailed discussion will be that of a smelter-refinery model, in which a hypothetical copper-nickel concentrate (a concentrate blend which is typical of a Minnesota copper-nickel concentrate) is treated for recovery of its copper and nickel content.

SMELTING OF COPPER AND NICKEL

Drying

Prepared concentrates are generally too high in moisture content to be successfully smelted using present-day pyrometallurgical processes. The principal purpose of dryers, then, is to dry and heat the furnace charge prior to smelting. Water is removed to a level which will allow further treatment.

Smelting

Concentrate drying is followed by smelting in which dried concentrate, subjected to intense heat, melts, forming a liquid sulfide (matte) phase containing the copper, nickel, and precious metals, and a liquid slag phase, essentially free of copper and nickel. Smelting is generally accomplished by melting the entire furnace charge at approximately 1200°C, usually with a silica flux. The silica, alumina, iron oxides, lime, and other minor oxides form molten slag, the copper, nickel, sulfur, unoxidized iron, and the precious metals form the matte. The slag is lighter than, and almost immiscible with, the matte and is easily tapped off separately.[4]

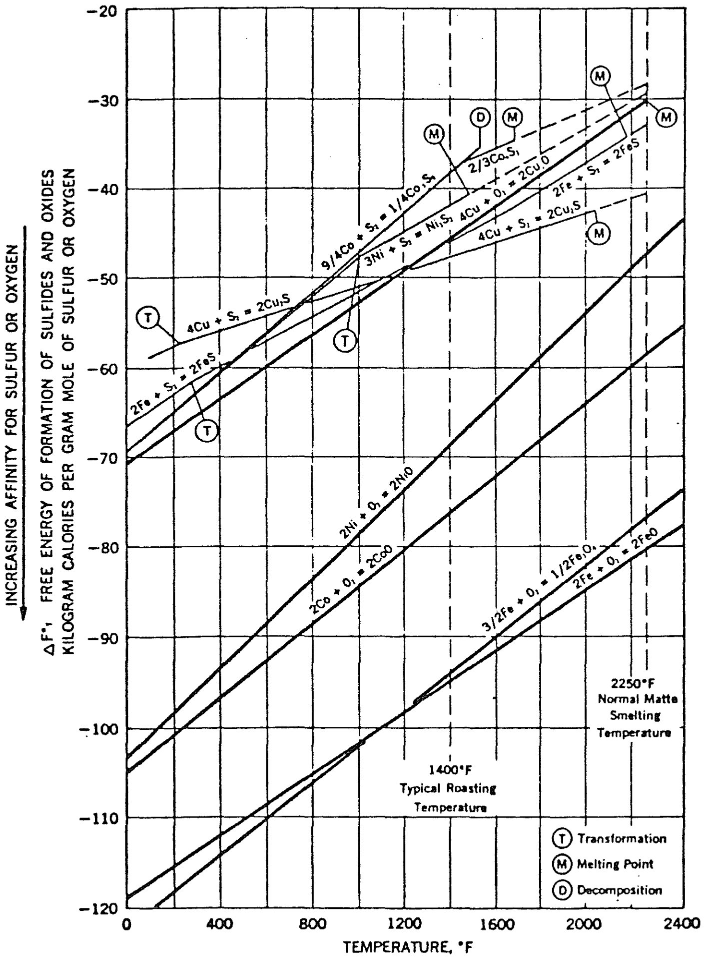
An important objective of matte smelting is to produce a discardable slag which is as low as possible in copper and nickel content. This is accomplished by keeping the slag near silica saturation (which promotes matte-slag immiscibility), by keeping the furnace sufficiently hot so that the slag is molten and fluid, and by avoiding excessively oxidizing conditions. This last condition is necessary to minimize the formation of solid magnetite which creates viscous conditions and consequently hinders the separation of matte from slag.[4]

The principal reactions which take place during smelting may be explained in terms of the differences in the thermodynamic properties of the metallic compounds. Thermodynamics provides the basis for predicting whether a reaction will take place or not. The speed with which a reaction takes place is a consequence of its chemical kinetics. At elevated temperatures most thermodynamically possible reactions take place (i.e. they will occur as rapidly as the reactants are brought into intimate contact with each other).[5]

Figure 95 illustrates in graphical form the relative affinities that the metals copper, nickel, iron, and cobalt have for sulfur and oxygen. At smelting temperatures iron sulfide is thermodynamically only slightly more stable (less likely to decompose into its constituent elements) than is nickel sulfide. Nickel sulfide and cobalt sulfide, on the other hand, have nearly equal stabilities, while the stability of copper sulfide is greater than that of the iron sulfide. Generally speaking, all the sulfides of the above mentioned metals are in the same stability range.[5]

In the presence of oxygen, each of the above sulfides is unstable with respect to its corresponding oxide. Iron has the greatest affinity for oxygen, followed by cobalt, nickel, and copper. This thermodynamic relationship between sulfides and oxides provides a key to separating the iron from the nickel, copper, and, to a lesser extent, cobalt values. If a mixture of the sulfides of the four metals is brought to equilibrium with a supply of oxygen insufficient to oxidize the mixture completely, the available oxygen will combine preferentially with the metals in the order of their oxygen affinities: iron, cobalt, nickel, and copper.[5]

Figure 95. Diagram Showing the Effect of Temperature on the Sulfur and Oxygen Affinities of the Principal Metals Encountered in Smelting. (Source: Boldt, 1967)



When a mixture of metal sulfides, iron oxide, gangue, and siliceous flux are melted together, the iron oxide, gangue, and silica form a layer (slag) which is lighter than the molten sulfides (matte). The slag is normally discarded leaving the matte which is a homogeneous solution of nickel, copper, cobalt, iron, and sulfur, with small amounts of other base elements, precious metals, and oxygen.[5]

The affinity that a particular metal or element has for another metal or element in a mixture is related to its concentration in the original mixture. Iron as a sulfide clearly demonstrates the greatest affinity for oxygen; however, as the oxygen combines with the iron to form an oxide compound, thereby decreasing the iron sulfide concentration, the affinities of the other sulfides for available oxygen (i.e. cobalt, nickel, and copper sulfide in the original mixture) now become prevalent, that is, cobalt, nickel, and copper sulfide react to form their respective oxides. A separation, consequently, is never complete and precise, simply because the various affinities and reactivities of one element for another element are constantly changing as reactions proceed.

Physical Chemistry of Smelting--The major constituents of a copper-nickel smelting charge are the sulfides and oxides of copper, nickel, and iron. The charge also contains the oxides of Al_2O_3, CaO, MgO, and SiO_2, which are either concentrated in the charge feed originally or are added as flux. The factors which control the process reactions are:

1) chemistry of the original feed (the amount of iron, copper, nickel, sulfur, and oxygen present in the feed which largely controls the chemistry and physical constitution of the matte-slag system)

2) the oxidation/reduction potential of the gases which are used to heat and melt the charge[4]

The first purpose of smelting is to ensure the sulfidization of all the copper and nickel present in the charge so that it enters the matte phase. The presence of FeS in the matte ensures the above conversions by sulfidizing the nonsulfidic copper and nickel of the charge by the following principal reactions:

$$\mathrm{FeS}(\ell) + \mathrm{Cu_2O}(\ell,\ \mathrm{slag}) \rightleftarrows \mathrm{FeO}(\ell,\ \mathrm{slag}) + \mathrm{Cu_2S}(\ell)$$

and

$$7\mathrm{FeS}(\ell) + 9\mathrm{NiO}(\ell,\ \mathrm{slag}) \rightleftarrows 7\mathrm{FeO}(\ell,\ \mathrm{slag}) + 3\mathrm{Ni_3S_2}(\ell) + \mathrm{SO}\ (g)$$

where ℓ denotes the liquid matte phase, slag denotes the slag phase
( ) denotes location

In the liquid state the matte is a homogeneous solution of copper, nickel, iron, and sulfur, with small amounts of other base elements, precious metals, and oxygen. The quantity of iron in the matte can be controlled by adjusting the degree of oxidation of the charge to the smelting furnace. Since the amount of sulfur removed as sulfur dioxide is proportional to the amount of iron oxidized, the amount of sulfur in the feed becomes a convenient measure of the extent to which the charge is oxidized. The more sulfur present in excess of the amount required to combine with the copper and nickel in the matte, the greater the iron content of the matte will be, and the lower will be its grade (percent Cu). Each smelting operation has its optimum grade matte. As a rule, smelting is carried to the point at which substantially all the iron sulfide has been eliminated from the original feed to the slag phase, and the metal values are concentrated in a high grade matte.[5]

Isolating Copper in Matte--The efficiency with which the copper of the charge is isolated in the matte depends upon some general operating conditions which promote a matte-slag separation with a minimum loss of copper to the slag phase:

- Silica in slag. Silica promotes the formation of separate matte and slag phases. The most complete isolation of copper in matte occurs at near saturation conditions 35 to 40 percent SiO_2. Silica is added directly and in recycled converter slag to achieve these conditions.
- Lime and alumina. Addition of these oxides tends to stabilize the slag structure and they are beneficial up to about ten percent in slag.
- Matte grade (percent Cu in matte). Experimental and industrial studies with mattes of industrial grade, 30 to 50 percent Cu, show that the concentration of copper in the slag phase is proportional to matte grade. Thus, with constant slag weight, a high matte grade causes a high loss of copper in the slag. This situation is reflected in the ratio

  $$\frac{\text{Wt \% Cu in slag}}{\text{Wt \% Cu in matte}}$$

  which is typically 0.01 to 0.02 in industrial operations.
- Slag weight. The weight of copper lost in slag is proportional to the weight of slag, other conditions being constant. Slag weight is minimized by charging concentrates of high copper grade and by avoiding the recycle of converter slag to the smelting furnace.
- Temperature and oxygen potential. A high smelting temperature (1200°C) leads to a more fluid slag, and a matte-slag separation with low copper losses in the slag. Highly oxidizing conditions lead to high copper losses in the slag.

Copper Converting

Smelting is generally followed by converting, where the matte from smelting is oxidized (with air), removing the iron and sulfur from the matte. The process is almost universally carried out in the horizontal basic refractory lined cylindrical Pierce-Smith Converter. Matte is added to the converters at 1100°C and the heat generated in the converter by the oxidation of iron and sulfur is sufficient to make the process autogenous.[4]

Copper converting is carried out in two sequential stages, both of which involve blowing air into the molten sulfide phase. These are:

1) The FeS elimination slag forming stage

$$2FeS + 3O_2(air) + SiO_2(flux) \rightarrow 2FeO \cdot SiO_2(slag) + 2SO_2$$

2) The blister copper (98.5% - 99.5% copper) forming stage

$$Cu_2S + O_2(air) \rightarrow 2Cu(blister\ copper) + SO_2$$ [4]

Copper making (stage 2) does not occur until the matte contains less than one percent iron. In normal industrial practices, the converter is charged with matte in several steps, each being followed by partial oxidation and slag removal. This results in a gradual accumulation of Cu_2S in the converter and it is followed by a final "copper making" blow.[4]

The product of the converting process is blister copper, which contains in the order of 0.02 to 0.1 percent sulfur. Significant formation of copper oxide does not occur until the sulfur content is below 0.02 percent so that oxidation of the copper is not a problem.[4]

Nickel Converting

In nickel converting, iron and sulfur are oxidized, eliminating substantially all of the remaining iron sulfide. Nickel, copper, precious metals, and most or all of the sulfur that was combined with them remain in the matte.

A new pyrometallurgical process is top blowing with oxygen in a rotary furnace. Cylindrical in shape, the furnace is rotated continuously about its longitudinal axis. While this device can perform normal converting, the excellent mixing and thermal efficiency of the arrangement permit operations not possible with standard converters. For example, molten nickel sulfide can be oxidized directly to liquid nickel metal, according to the overall reaction:

$$Ni_3S_2 + 2O_2 = 3Ni + 2SO_2$$

The success of this operation requires high temperatures and intimate contact between the gas, the molten nickel-sulfur, and solid nickel oxide. With appropriate control of temperature, agitation, and gas addition, this turbulent bath concept can also be successfully applied to copper-nickel mattes and copper mattes to produce directly cupro-nickel and copper with low impurity levels.[5]

The principal factors which determine the production rate of converters are:

- matte grade
- air blowing rate

The quantity of air required for conversion of sulfide to oxide or metal decreases dramatically with an increasing matte grade. This is, of course, due to the fact that there is much less iron sulfide to be oxidized. Thus, the rate of matte conversion can be appreciably increased by charging a high-grade matte. Increasing the air blowing rate will also increase productivity. An upper limit, however, exists above which the liquids are excessively ejected from the converter.

REFINING OF COPPER AND NICKEL

Crude anode copper often has the following approximate composition:[4]

Element	Anodes (Range of %)
Cu	97.5-99.5
Fe	0.01-0.1
Ni	0.1-0.5
S	0.03-0.3
Zn	0-0.03
Pb	0.05-0.26
Sb	0-0.3
As	0-0.12
Sn	0-0.1
Bi	0-0.05
Se	0-0.1
Te	0-0.1
O_2	0-0.1
Ag	0-0.1
Au	0-0.005

Crude anode nickel has the following approximate composition:[5]

Element	Anodes (Range of %)
Cu	4.0-17.0
Fe	0.8-2.0
Ni	76.5-93.5
S	0.6-1.4
Pb	0.01-0.05
As	0.05-0.5

Crude copper and crude nickel must be subjected to a refining type process in order to remove the various impurities present. Generally speaking, there are two methods of refining (i.e. pyrometallurgical and electrolytic). With the pyrometallurgical process it is only possible to obtain low grades of copper, since the impurities are not completely removed in the process. In order to remove the impurities most completely and to recover the valuable elements, the crude metals must be subjected to both forms of refining separately.

Fire Refining and Electrorefining of Copper

Electrorefining requires strong, flat, thin anodes to interleave with the cathodes in the refining cell; however, these cannot be obtained by directly casting blister copper. Residual sulfur and oxygen contained in the impure copper form large blisters of SO_2 during solidification which: 1) unacceptably weaken the anodes (in the region of the SO_2 blisters); and 2) cause rough surfaces of uneven thickness. Thus, the sulfur and oxygen are removed by fire-refining techniques prior to anode casting; the sulfur by injecting air into the molten blister copper; and then the oxygen by injecting hydrocarbon gas.[4]

Fire refining is carried out in rotary-type refining furnaces (anode converters) resembling Pierce-Smith converters or in small hearth furnaces. The temperatures of operation is approximately 1130 to 1150°C which provides sufficient superheat for the subsequent casting of anodes. There is very little heat produced by the refining reactions, therefore, some combustion fuel is necessary to maintain the temperature of the furnace.

The essential reaction for the removal of sulfur by air is:

$$[S]_{Cu} + O_2(g) \rightarrow SO_2(g)$$

while simultaneously oxygen is dissolving in the copper by the reaction:

$$O_2(g) \rightarrow 2[O]_{Cu}$$

The oxygen concentration in the desulfurized copper is reduced and/or removed from the molten copper with hydrocarbons such as natural gas (CH_4), reformed natural gas ($CO + H_2$), propane (C_3H_8), or wood, producing carbon monoxide, carbon dioxide, and water as residual products from the chemical reactions.

In the past, a final copper product was produced by fire refining in this manner; however, many impurities cannot be removed by this technique and the product must undergo electrorefining.

The electrorefining of copper consists of the electrochemical dissolution of copper from the impure anodes and the plating of pure copper (without anode impurities) onto copper cathodes. The electrolyte is generally an aqueous solution of H_2SO_4 and $CuSO_4$ usually with a trace amount of chlorine. Many of the anode impurities are insoluble in this electrolyte (Ag, Au, Bi, Pb, Pt, Sn) and they do not interfere with the electrolysis. Other impurities such as As, Fe, and Ni are partially or fully soluble and these must be kept to a low concentration in the electrolyte to prevent them from being occluded with the copper on the cathode. This is accomplished by continuously bleeding part of the electrolyte through a purification circuit.[4]

Typical anode to cathode voltages are 0.20 to 0.25 volts at current densities of 200 amperes per square meter of cathode.

The application of an electrical potential between a copper anode (positive electrode) and a copper cathode (negative electrode), both immersed in a cell containing an acidified copper sulfate solution, causes the following reactions and processes to take place:

1) Copper is electrochemically dissolved from the anode into solution,

$$Cu^{0}_{Anode} \rightarrow Cu^{+2} + 2e^{-} \quad E^{0} = +0.34V \qquad (a)$$

2) The electrons produced by reaction (a) are conducted towards the cathode through the external circuit and power supply.

3) The Cu^{+2} cations in the solution are conducted to the negative electrode (cathode).

4) The electrons and the Cu^{+2} ions recombine at the cathode surface to produce copper metal which plates on the cathode,

$$Cu^{+2} + 2e^{-} \rightarrow Cu^{0} \qquad E^{0} = +0.34V \qquad (b)$$

The net effects are the electrochemical dissolution of copper from the anode; the migration of electrons and copper ions towards the cathode; and the plating of copper on the cathode surface. The overall electrochemical reaction is the sum of reactions (a) and (b),

$$Cu^{o} \rightarrow Cu^{o}$$

for which the theoretical (reversible) potential, (i.e. the difference between the electrode potentials) is 0 volts.[4]

Electrowinning of Copper

Copper can be recovered in marketable form by electrowinning from strong leach solutions or from the electrolytes produced by solvent-extraction techniques. Electrowinning is similar to electrorefining except that the anode is composed of an inert material, usually antimonial lead. The overall reaction for copper electrowinning may be written:

$$CuSO_4 + H_2O \overset{2V}{\twoheadrightarrow} Cu^{o}{}_{cathode} + \uparrow \tfrac{1}{2}\, O_{2anode} + H_2SO_4$$

Copper is produced at the cathode which is started on a copper base, evolves at the anode, and sulfuric acid is regenerated for reuse as a leachant.[4]

Electrowinning requires approximately ten times the voltage used in electrorefining and thus it uses considerably more electrical energy. In addition, the cathode product is less pure than electrorefined copper, mainly because of contamination from the lead anode which is not completely inert.[4]

Electrorefining of Nickel

In nickel refining, nickel anodes and thin "starting sheets" of pure nickel (starter cathodes) are immersed in an electrically conducting aqueous solution called an electrolyte, and connected to each other through a source of direct current electricity. The current source creates a potential difference, or voltage, and as the current flows through the cell, the nickel and some of its associated impurities in the anode dissolve in the adjacent electrolyte (the anolyte). This solution is pumped from the cell, treated for removal of impurities, then returned to the cell as catholyte (the electrolyte adjacent to the cathode). Nickel from the catholyte is then deposited on the cathode.[5]

In the refining of crude nickel anodes, the principal anodic reaction is the dissolution of nickel metal as nickel ions, leaving two electrons in the anode to be transmitted to the cathode:

$$\mathrm{Ni}_{\text{anode}} = \mathrm{Ni}^{+2}_{\text{anolyte}} + 2e^{-}_{\text{anode}}$$

The principal cathodic reaction taking place in the cell is the reduction of nickel cations from the catholyte by their picking up two electrons at the cathode and depositing as nickel atoms:

$$\mathrm{Ni}^{+2}_{\text{catholyte}} + 2e^{-}_{\text{cathode}} = \mathrm{Ni}_{\text{cathode}}$$

The net cell reaction (the sum of the principal anodic and cathodic reactions) is zero, a result characteristic of any electrorefining process.[5]

Electrowinning of Nickel

Pure nickel cathodes may also be produced by electrowinning. Two different methods of electrowinning nickel are currently in use:

1) electrolysis of soluble anodes of nickel sulfide
2) utilization of insoluble anodes to extract nickel from a leach liquor

Electrowinning with Sulfide Anodes--Nickel sulfide anodes (essentially Ni_3S_2 in composition) involve anodic reactions where oxidation of sulfide sulfur to the elemental state takes place with the release of metal ions into solution and electrons to the anode. The reaction is summarized below.[5]

$$Ni_3S_2 = 3Ni^{+2} + 2S + 6e^-$$

The cathodic reaction is represented by:

$$3(Ni^{+2} + 2e^-) = 3\ Ni$$

and the net cell reaction is represented below as:

$$Ni_3S_2 = 3Ni + 2S$$

Electrowinning With Insoluble Anodes--Electrowinning with insoluble anodes provides a means of recovering nickel from leach liquors. Since the nickel being extracted at the cathode enters the electrolyte by a leaching reaction, the only function of the anodes is to transfer electrons to the external circuit for delivery to the cathode. For this reason the anodes are made of an insoluble material, such as antimonial lead, that has a long life in sulfate electrolytes. Thus, there is no need to cast and handle anodes or to collect tank slimes.[5]

Electrolyte (the purified leach liquor) is an aqueous solution of nickel sulfate, sodium sulfate, and boric acid. In order for current to pass through the cell, a chemical reaction must occur at the anode. Since the anodes are insoluble, this reaction must be either decomposition of water or the oxidation of anions. The former reaction is in fact the principal anodic reaction:

$$2H_2O = O_2 + 4H^+ + 4e^-$$

Hydroxyl ions are also oxidized at the anode potential employed, but their concentration in acid solution is so low that this reaction accounts for very little anodic current. Since the electrode potentials required for the oxidation of sulfate anions are much greater than that for the decomposition of water, this reaction does not proceed. Adding the principal anodic reaction to the principal cathodic reaction

$$2(Ni^{+2} + 2e^- = Ni)$$

the net cell reaction is found to be:

$$2Ni^{+2} + 2H_2O = 2Ni + 4H^+ + O_2$$

or

$$2NiSO_4 + 2H_2O = 2Ni + 2H_2SO_4 + O_2$$

The sulfuric acid generated at the anode is utilized by recirculating the anolyte to the leach circuit.[5]

SMELTER-REFINERY MODEL

Assuming that the preliminary and/or feasibility engineering studies indicate continuance of the development of the mining operation, the next logical step is to predict and/or develop concepts or models of each and all of the vertical stages of mineral development (i.e. exploration, mining, milling, processing, smelting, refining). The various physical models are necessary as they establish the specific and detailed criteria which form the basis for actual operation.

GENERAL ASSUMPTIONS

The models which follow represent "generic regional" smelting and refining operations. The following general assumptions have been made relative to each of the models.

- A "material balance" is assumed to represent the quantitative flow of constituents through a process flowsheet. A "complete material balance" represents the quantitative flow of all the constituents of an input material through a process flowsheet. (This assumes that a complete and total chemical analysis is known with regards to the input material.) A "smelter/refinery material balance" represents the quantitative flow of constituents through a process flowsheet which involves the smelting and refining of said constituents. A "complete smelter/refinery material balance" represents the quantitative flow of all the constituents of an input material through a process flowsheet which involves the smelting and refining of said input material. (This assumes that a complete and total chemical analysis is known with regards to the input material.) A "fully" integrated smelter/refinery material balance" represents the quantitative flow of all input constituents through a process flowsheet which involves the smelting and refining of said input constituents to a final product (this assumes that no intermediate products exist in the balance which could be further treated, stored, or discarded).

- All models developed from the various material balances contained in this report were without the aid of specific and detailed data relating to the treatment of Minnesota copper-nickel concentrates. Hence, all quantitative data have minimum acceptable accuracies of ±50 percent. In many cases the true accuracy of individual numbers will exceed ±100 percent.

- The flowsheet as outlined in Figure 1, which is the basis for the material balances found in this report, is itself, an assumption. Flowpaths have been hypothetically selected and are used as a tool in order to predict end-use quantities of metals, acid, concentrates, etc. The particular flowsheet chosen here may or may not be real, and hence, may or may not be indicative of treatment of Minnesota copper-nickel ores.

- The material balances are based on the production of ≈ 100,000 MTY of high grade copper-nickel matte. The smelter/refinery model is based on the production of ≈ 100,000 MTY of metal (copper + nickel). This, of course, means that the quantities of any one particular constituent appearing in the material balances will have to be multiplied by a "multiplying factor" to yield the quantities as would be indicated by the smelter/refinery model.

- The output products as indicated in the smelter/refinery material balance are:

  metallics
  Fe-Cu-Ni concentrate
  cathode copper
  cathode nickel

- To convert the output products to a final metal product basis (copper + nickel), it will be assumed that the metallic fraction and the Fe-Cu-Ni concentrate fraction (both of these fractions contain copper and nickel) will be treated for copper and nickel recovery with no losses of either element. An overall copper recovery of 96.31 percent and an overall nickel recovery of 91.68 percent. These recoveries yield a Cu/Ni ratio of 7.00, which implies a "multiplying factor of" 1.37394 to be used in scaling up the numbers found in Table 1 to yield a smelter/refinery model production of 100,000 MTY of metal (copper + nickel). Refer to Figure 2 to determine the "multiplication factor" for other size operations. Note how the "mulciplication factor" will change as the copper and nickel recoveries change.

- The multiplication factors to be used in the scaling of the quantities found in Table 1 are determined by the equation below:

$$MF = \frac{A}{(x \cdot B + y \cdot C)}$$

  where
  MF = the multiplication factor
  A = the desired size of operation (metric tons of metal (copper + nickel) per year)
  X = the copper in the concentrate feed to the smelter (metric tons of copper per year)
  B = the overall percent copper recovered
  Y = the nickel in the concentrate feed to the smelter (metric tons of nickel per year)
  C = the overall percent nickel recovered

- Material balances found in this report do not represent "complete material balances."

- Smelter/refinery material balances found in this report do not represent "complete smelter/refinery material balances."

- Smelter/refinery material balances found in this report do not represent "fully integrated smelter/refinery material balances."

- Oxygen is not considered in the material balances. It is not possible to predict oxygen or air pathways, efficiencies, or air dilution values. Therefore, it is assumed that sufficient oxygen as air and/or pure oxygen will be available to carry out the required reactions in the smelting and converting of copper-nickel ores. Total gas flow volumes are not known; however, estimates of gas flow, based on historical information, will be attempted.

- All models are considered to be "off site" models with respect to all parameters and variables (i.e. capital and operating costs, manpower requirements, surface land use, water needs, etc.). The only exception to the above assumption will be with respect to transportation costs. Detailed transportation cost data was not generated at the time of the writing of this report. Smelters and/or refineries located "on site" will necessarily incur lower capital and operating costs, reduced manpower requirements, smaller surface land use, etc.

- Values predicting sulfur removal, fugitive emissions, stack emissions, metal recoveries, and so on, are assumed values.

- Specification sheets relating to quantity, size, and type of equipment used in the smelting and refining operations have not been included in the modeling. Flowsheets have been generalized to include only major types and quantities of equipment needs.

- Capital costs, operating costs, energy requirements, water requirements, manpower needs, and so forth, are broken down by generalized categories. (These values will exclude any contributions resulting from any intermediate mineral processing which might occur. Mineral processing values will be interpolated from the main mineral processing model.)

SMELTER MODEL

The smelting model which follows is broken down into two parts, namely:

- concentrate drying
- concentrate smelting

Concentrate Drying

Figure 96 illustrates, in generalized form, the flowsheet for the drying of copper. Table 21 summarizes the material balance for the drying flowsheet, and includes the major metals, trace elements, and principal impurities contained in the wet concentrate. Please refer to Table 22 for a list of the specific assumptions made here. Figures 97 through 122 indicate quantitatively the principal contaminants (principally fugitive and stack particulate emissions) to the atmosphere resulting from the drying of copper-nickel concentrates. (The starred values are those values assumed in the material balance.) The solid black lines indicate how each starred value quantitatively changes as the assumptions relating to particulate matter removal or fugitive particulate matter generation (the x-axes) changes. The dashed lines and the consequent shaded areas indicate the extent of the error (±50%) associated with each value represented, again,

Table 21. Metallurgical Material Balance of the Principal Constituents of a Typical Cu-Ni Concentrate (Concentrate Drying) (Quantities are in Metric Tons Per Year).

	Elements																
	Cu	Ni	Fe	S	SiO_2	CaO	MgO	Al_2O_3	As	Pb	Zn	Cd	Ag	Au	Rh	Pt	Pd
<u>INPUT</u>																	
Concentrate (Dry Basis)	66,118	9,929	153,911	119,613	66,586	6,546	23,436	12,879	13.52	41.62	1,965	17.72	23.26	0.72	.0088	0.89	2.11
<u>OUTPUTS</u>																	
Concentrate	65,536	9,842	152,557	118,560	66,000	6,488	23,221	127,66	13.40	41.25	1,947.3	17.56	23.06	0.71	0.00870	0.88	2.09
Fugitive Particulates	331	50	769	598	333	33	117	64	0.07	0.21	9.82	0.09	0.12	.004	.00006	.005	0.01
Stack Particulates	251	37	585	455	253	25	98	49	0.05	0.16	7.88	0.07	0.08	.006	.00004	.005	0.01

BASIS: 100,000 Metric Tons Per Year Production of White Metal Matte.

by the solid black line. (Similar figures—125 through 144—found later in this report should be interpreted in like fashion.)

Figure 96. Generalized Flowsheet for Process Concentrate Drying.

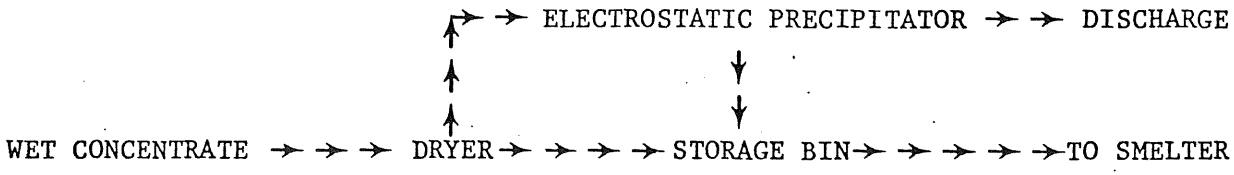


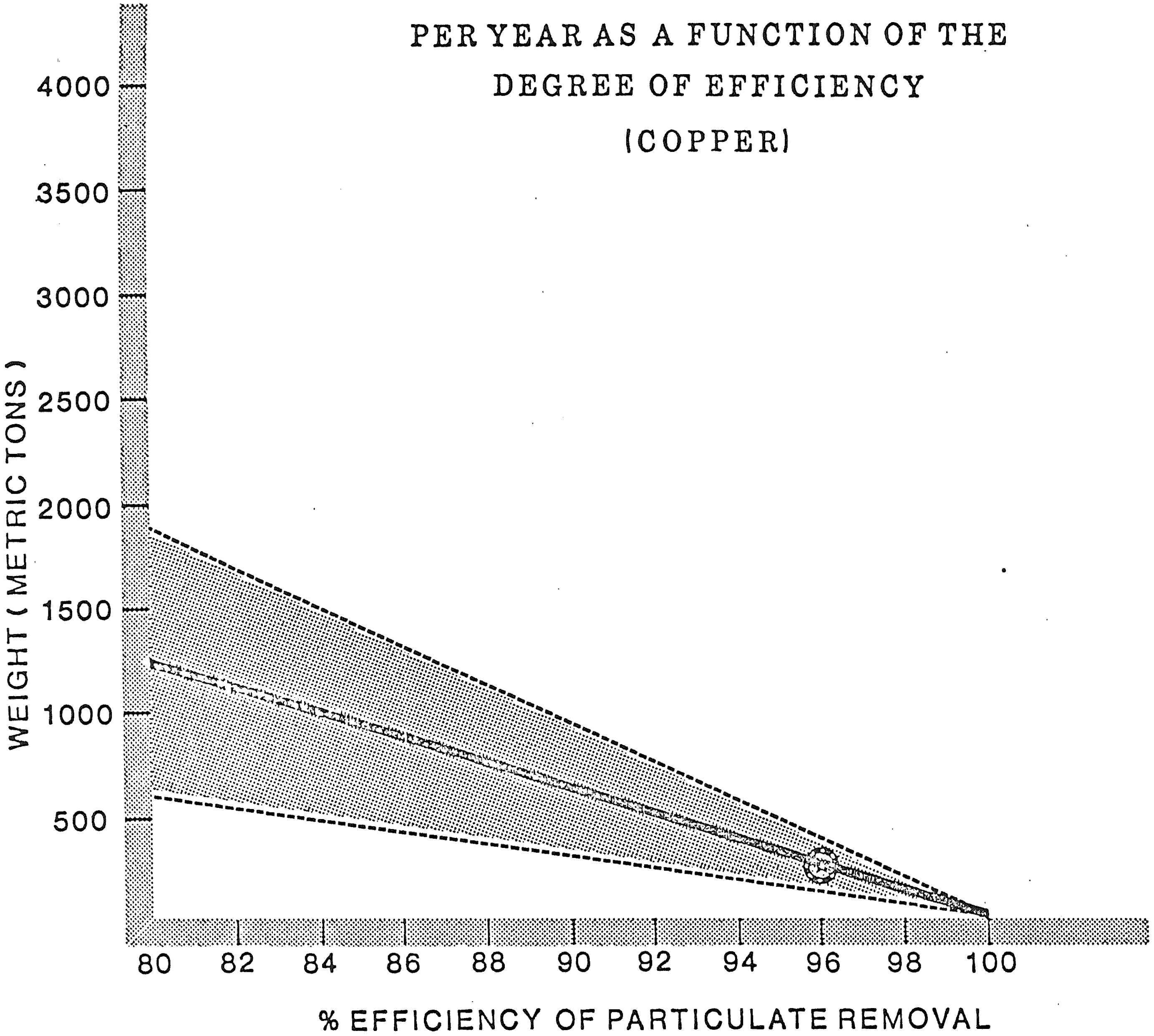
Table 22. Table of "Drying" Assumptions.

- The moisture content of the concentrate to the smelter is 15 percent.
- Ten percent by weight of the concentrate will become entrained in the dryer gas stream.
- Five percent of the total dust generated will be considered to be fugitive.
- Ninety-six percent of the dust generated (excluding fugitive dust) is treated and removed and returned to the system.

Figure 97.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY

(COPPER)

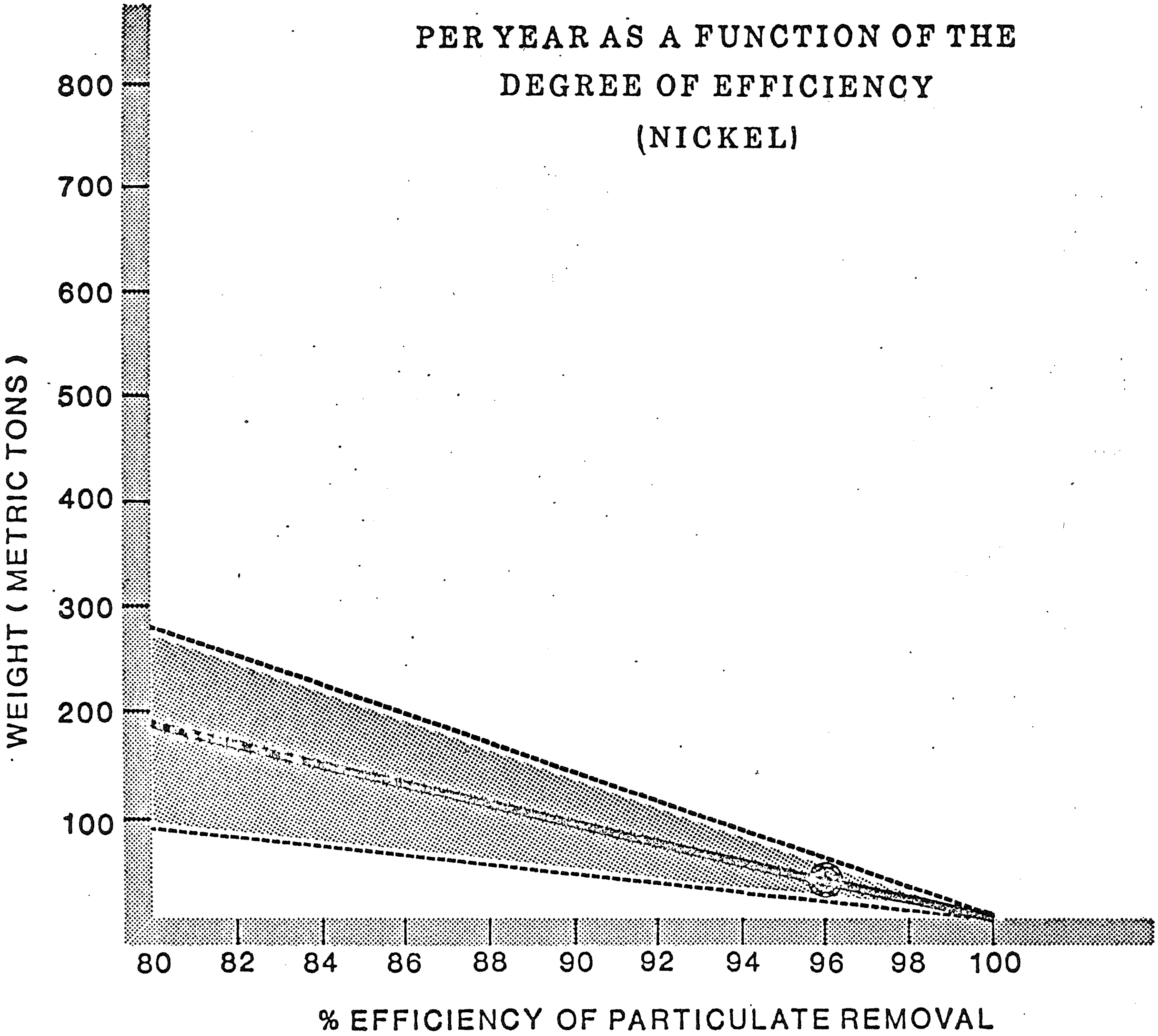


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 98.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY (NICKEL)

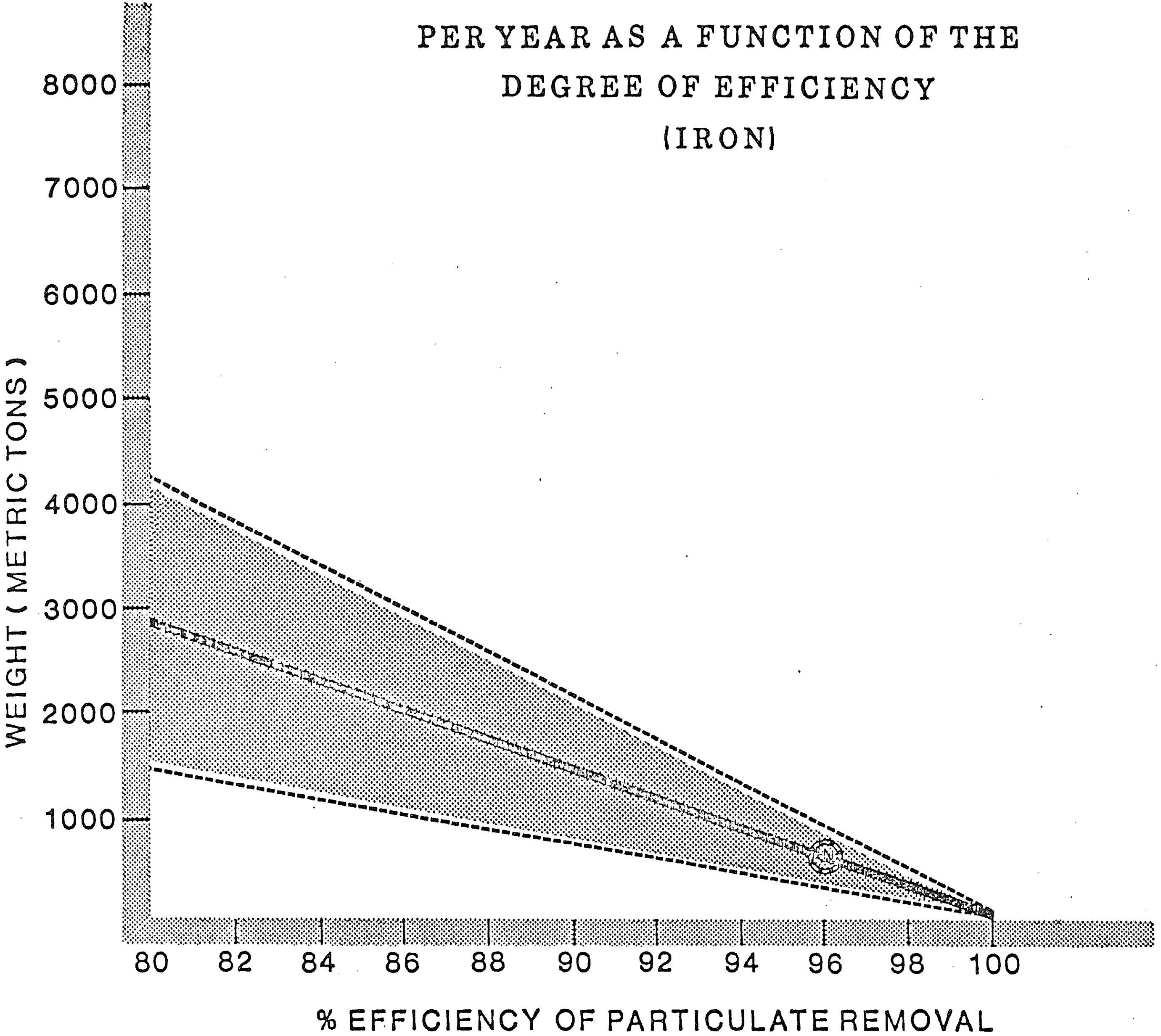


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 99.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY (IRON)

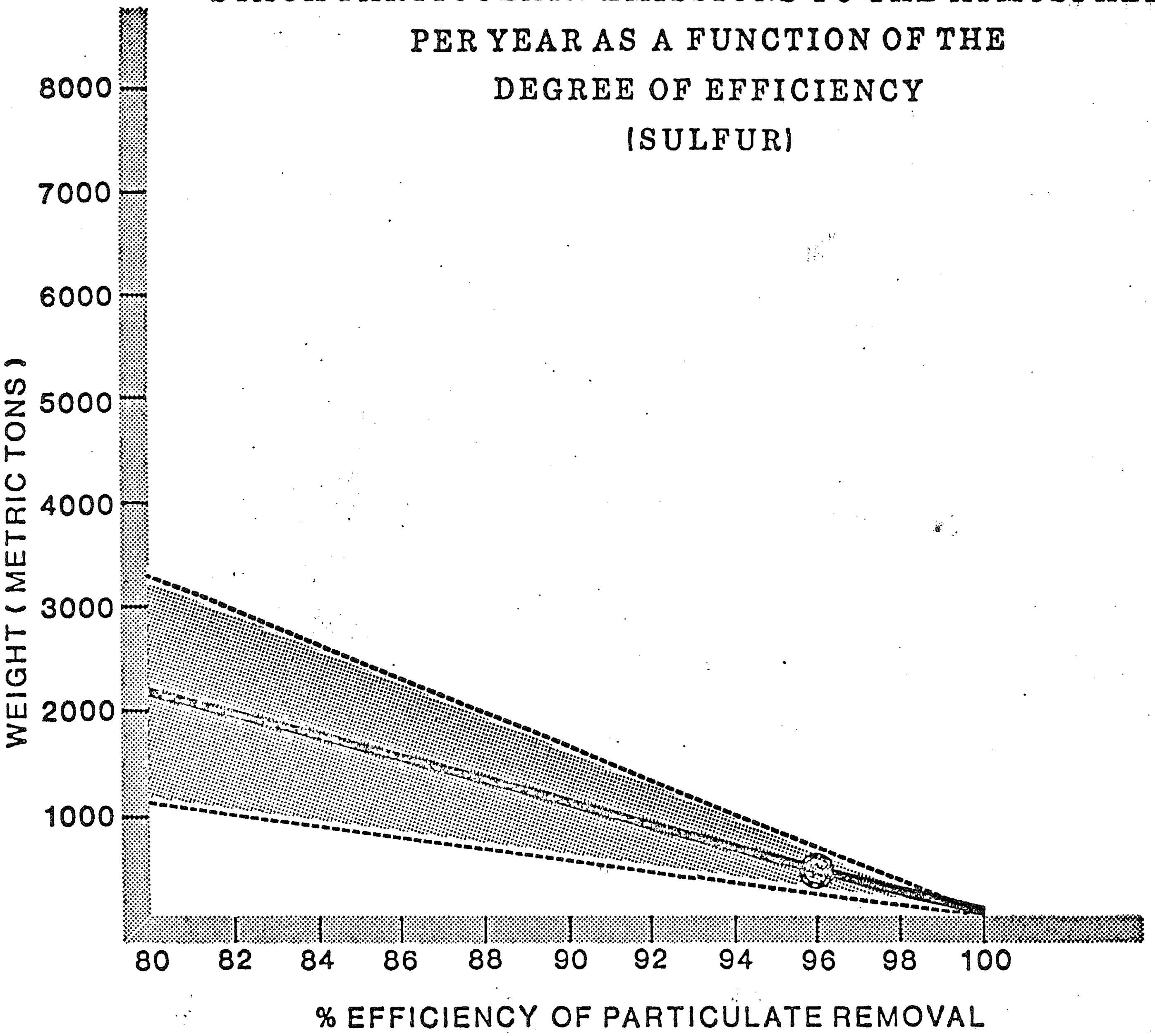


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 100.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY (SULFUR)



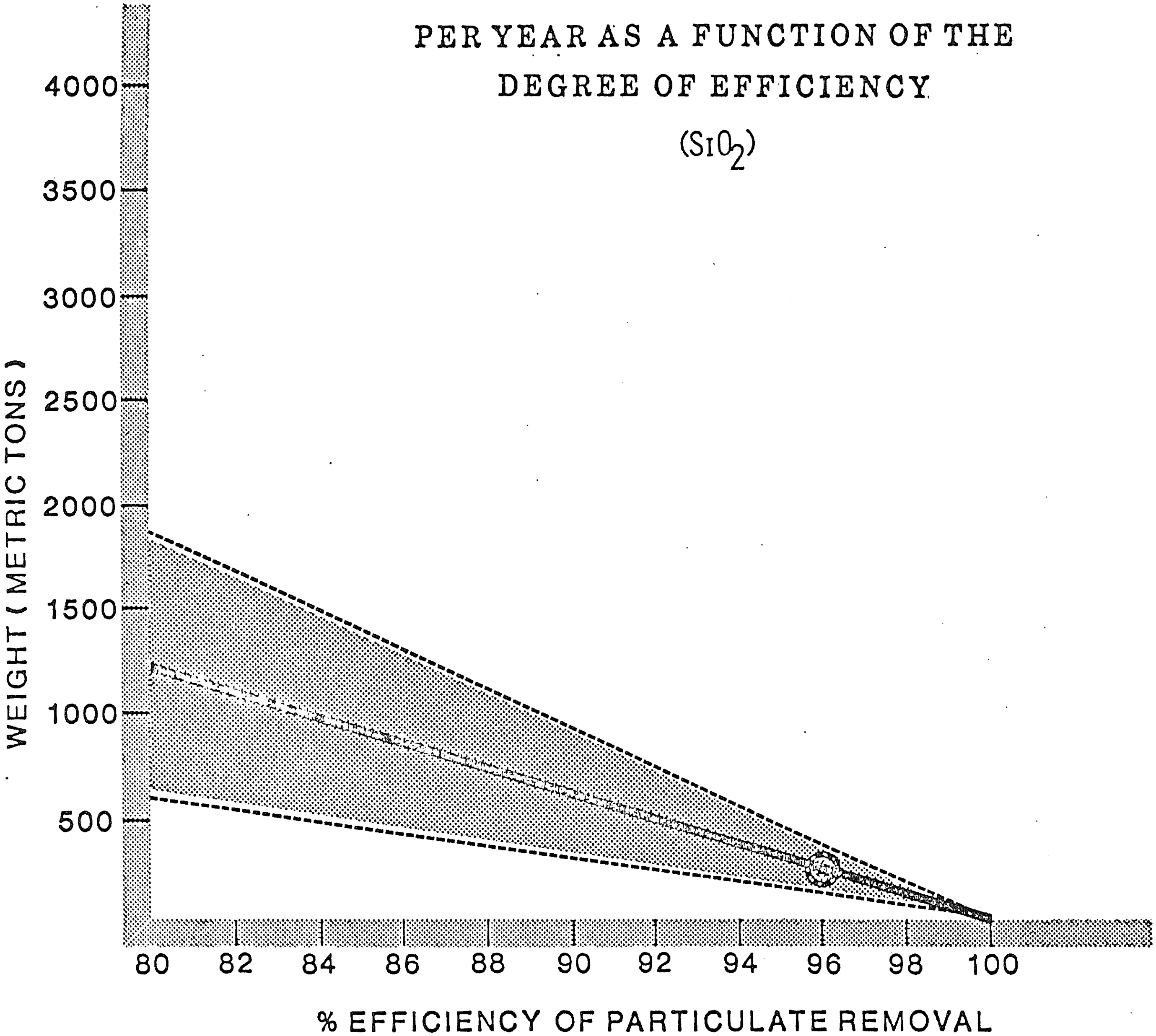
BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THF MATERIAL BALANCE CHANGE.

Figure 101.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY

(SiO_2)



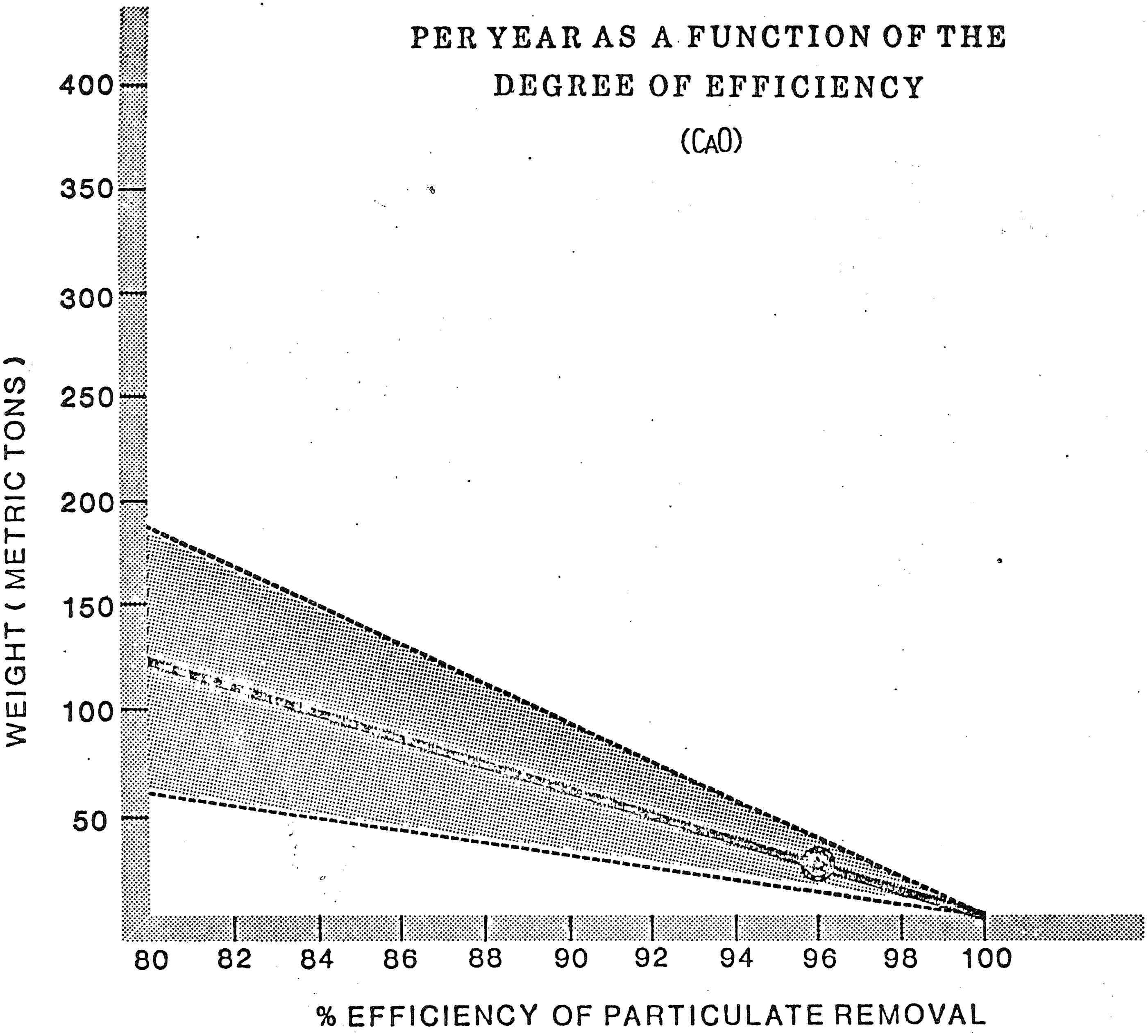
BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 102.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY

(CAO)



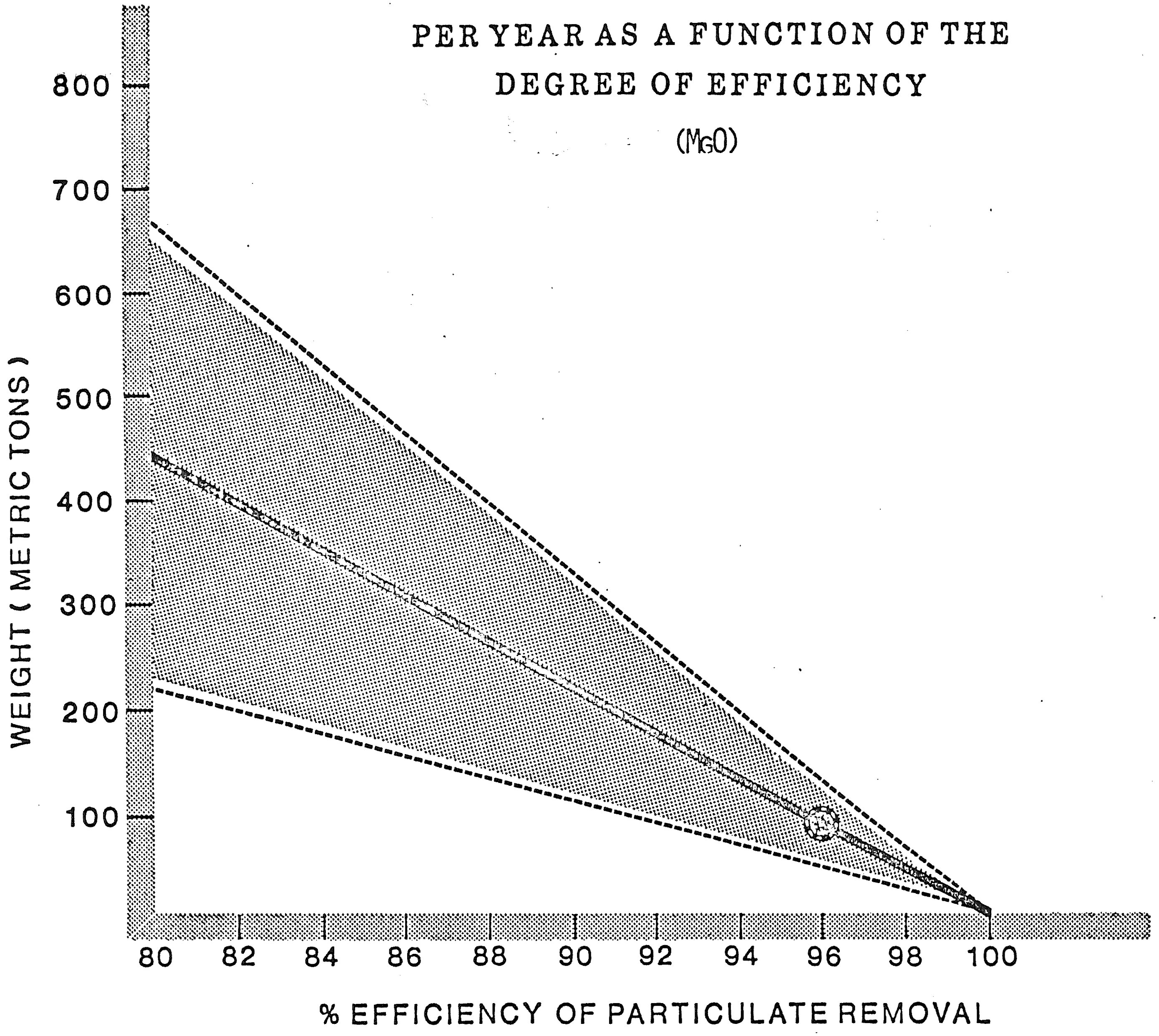
BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 103.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY

(MgO)



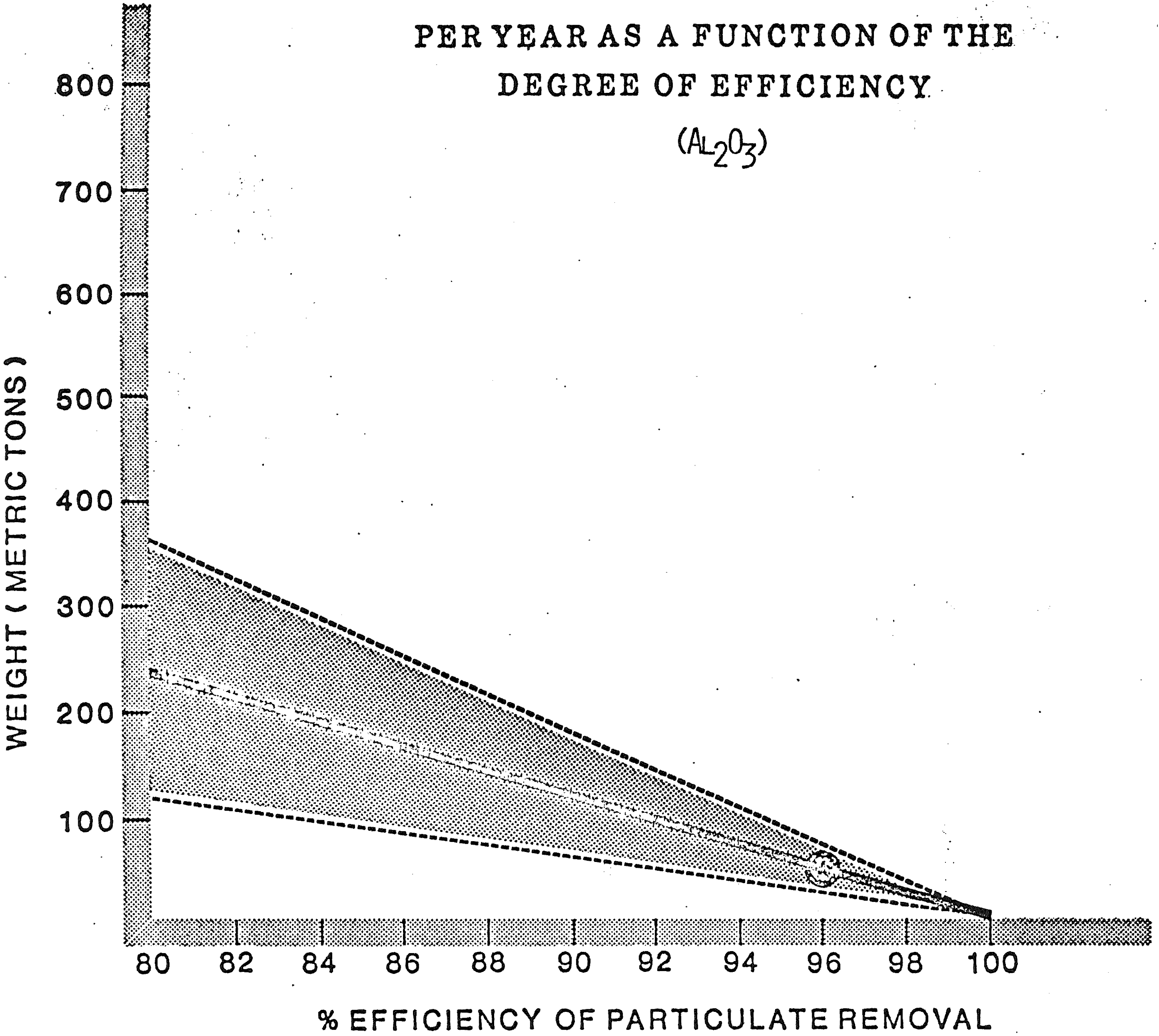
BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 104.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY

(Al_2O_3)

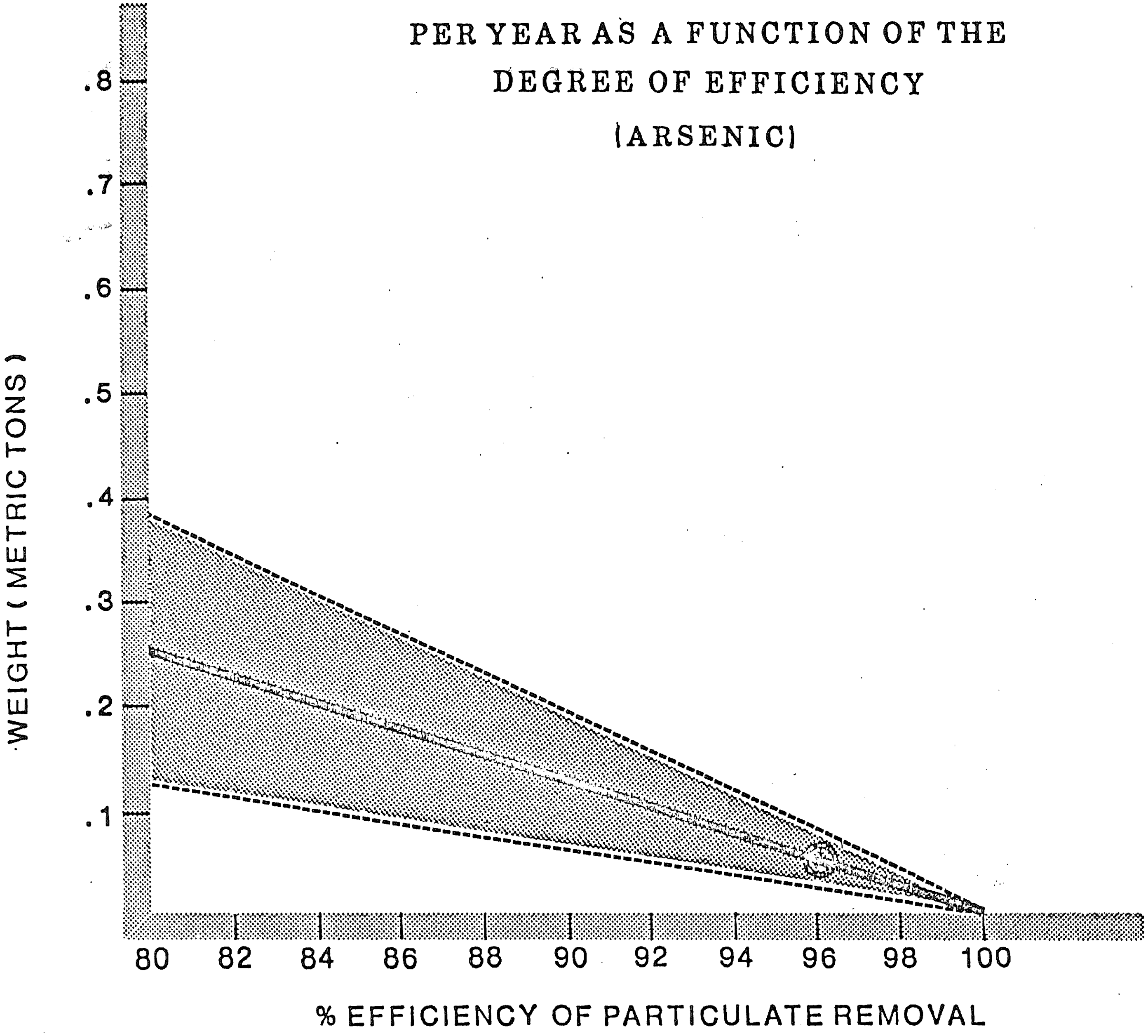


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 105.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY (ARSENIC)

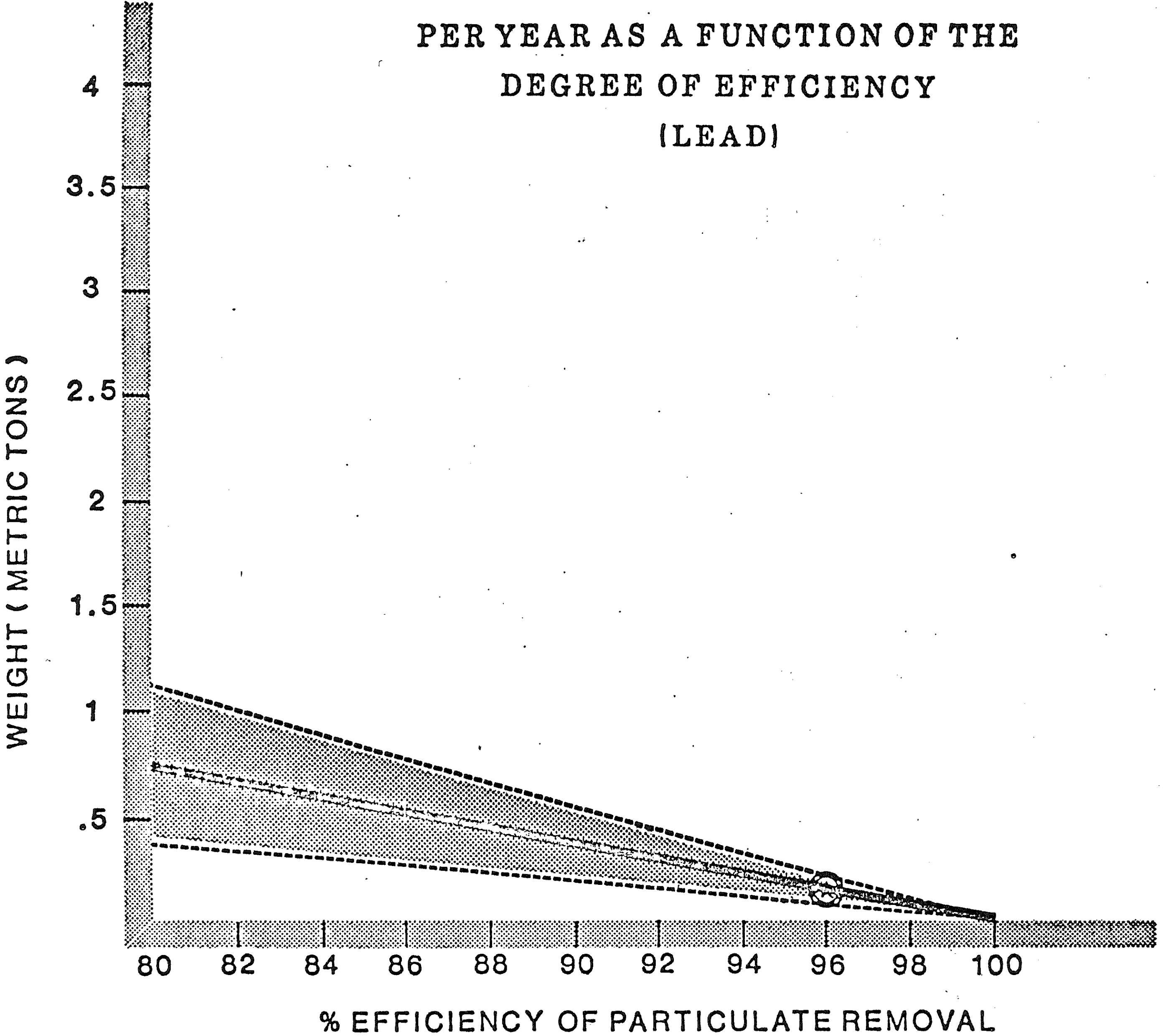


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 106.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY (LEAD)



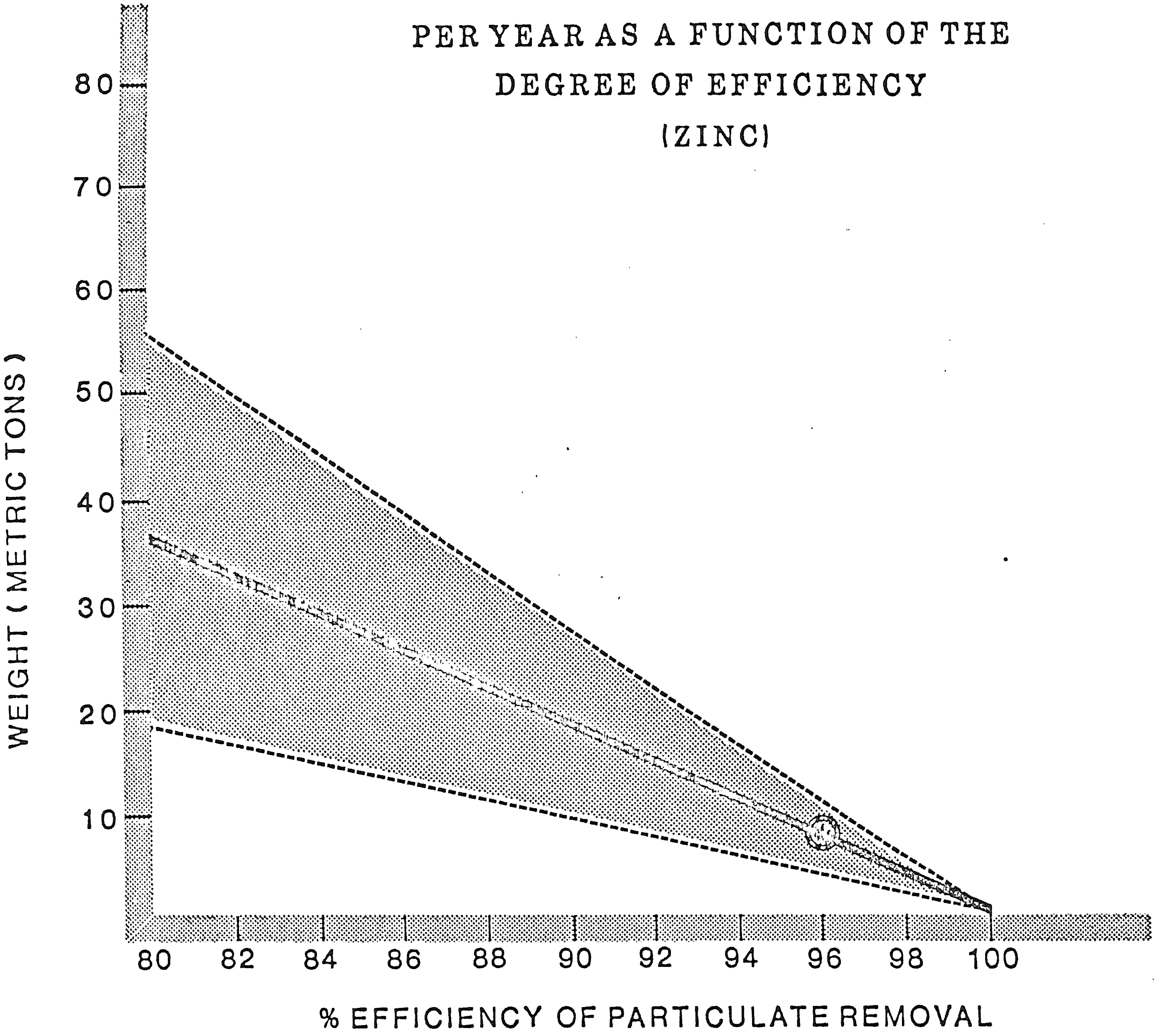
BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 107.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY

(ZINC)

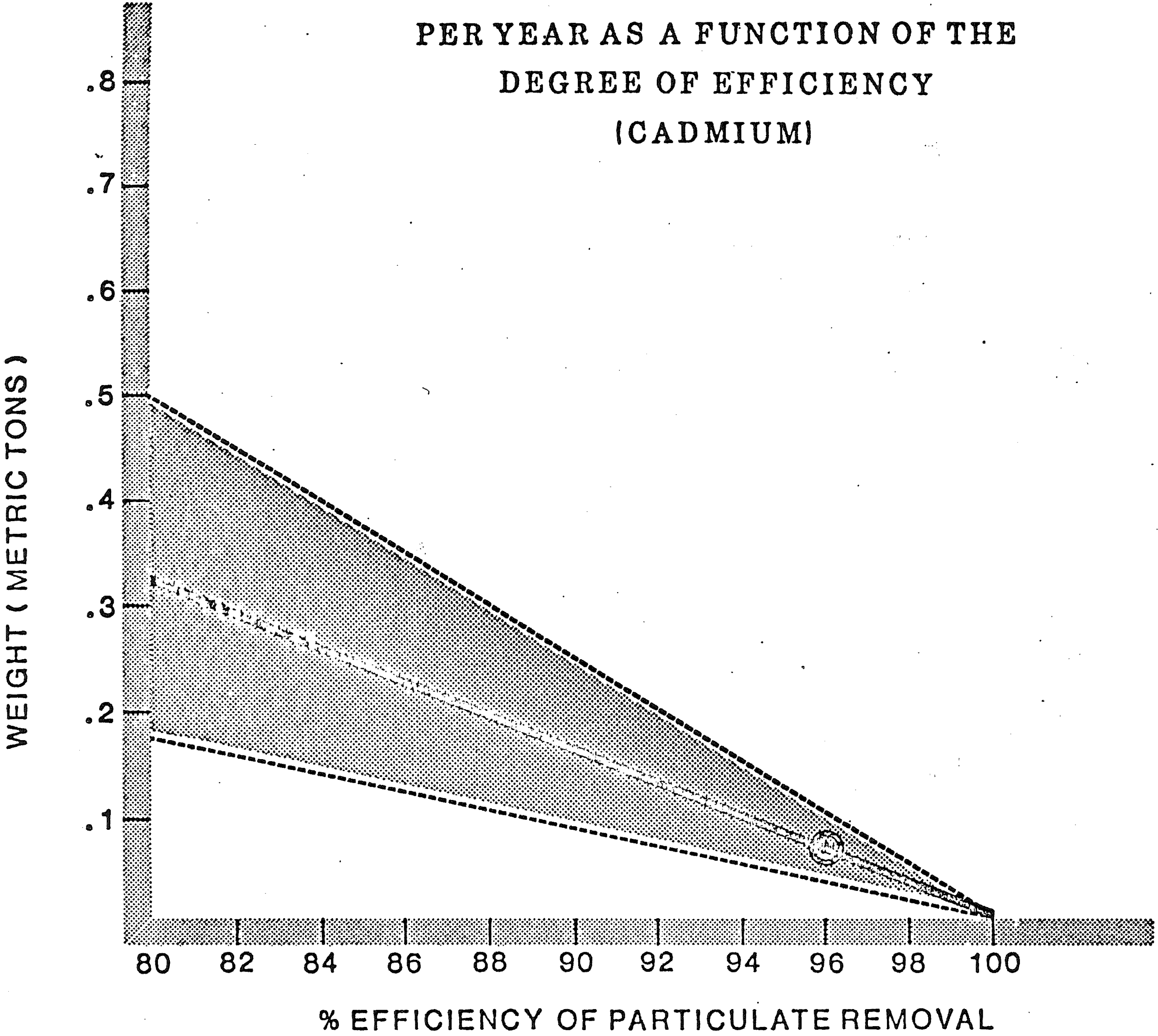


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE
AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 108.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY (CADMIUM)

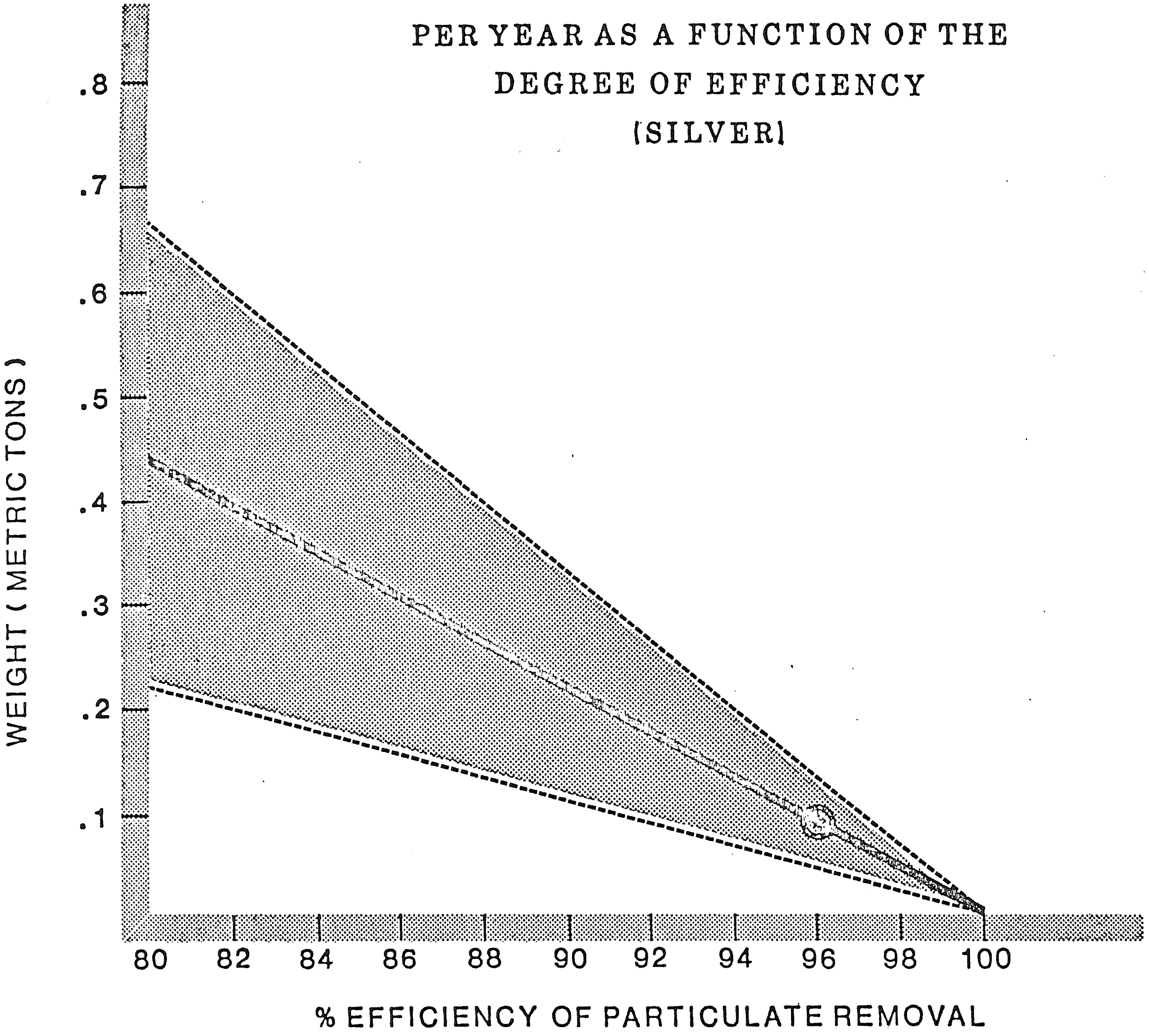


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 109.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY (SILVER)

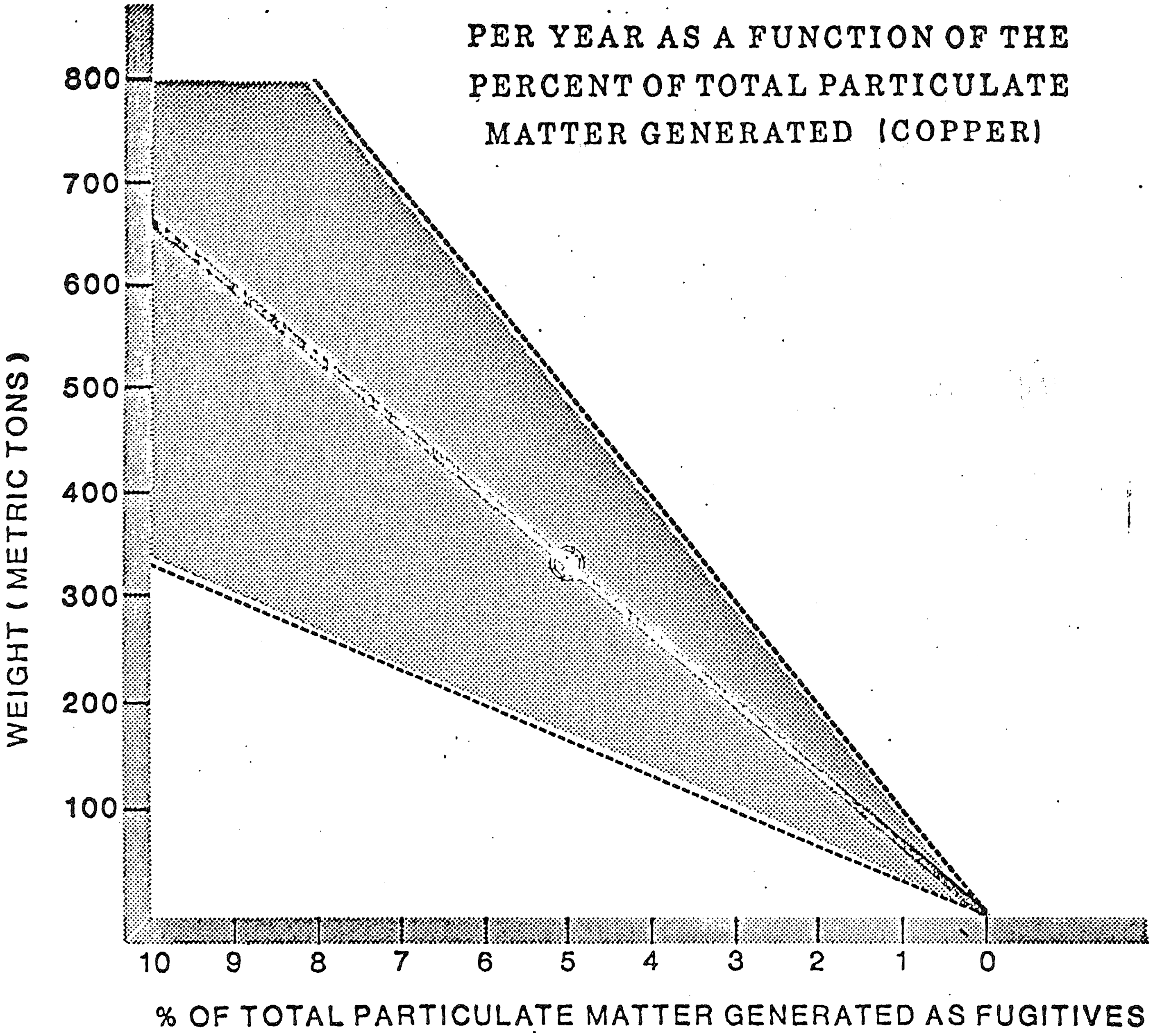


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE
AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 110.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (COPPER)

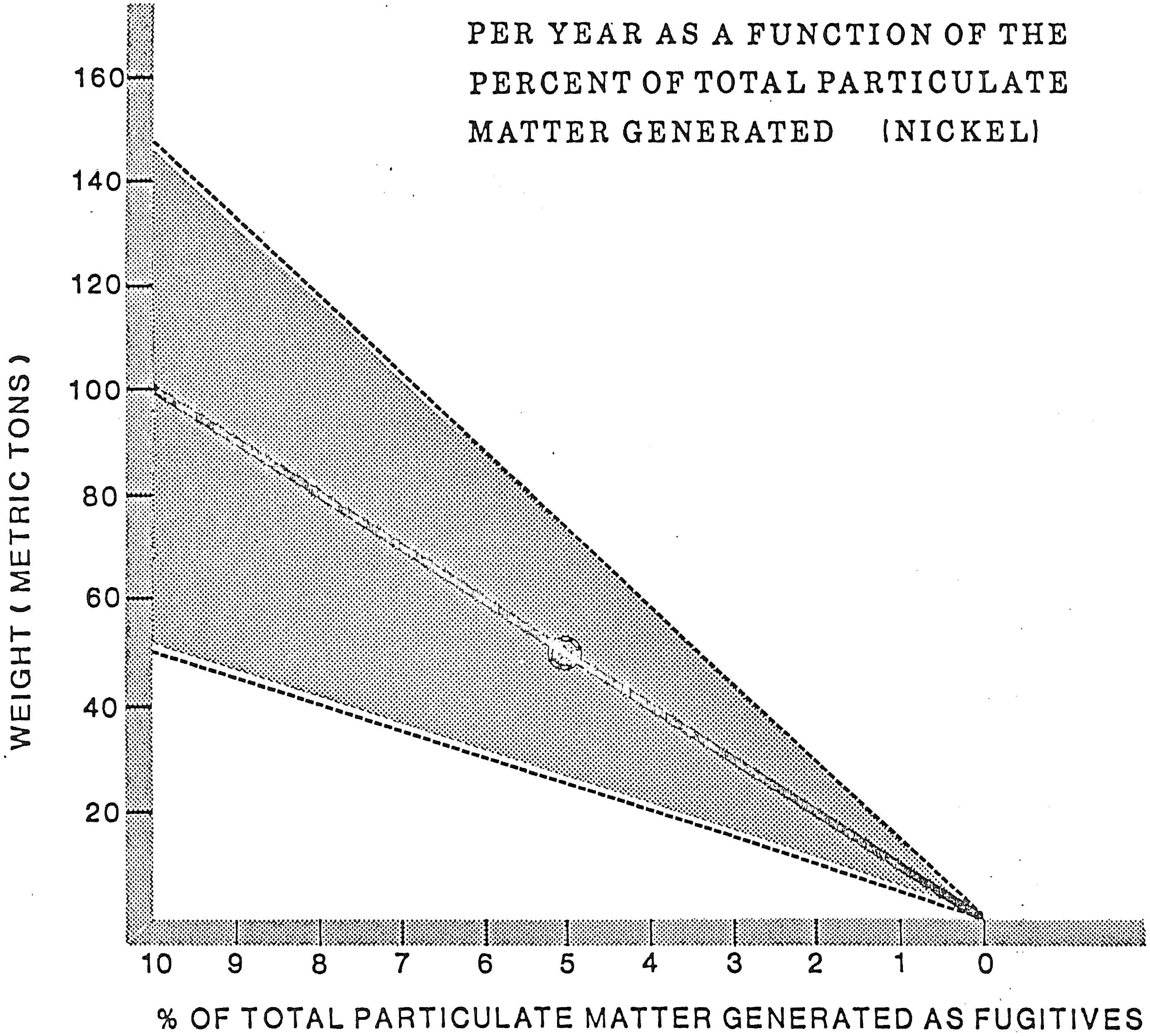


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE
AS THE ASSUMPTIONS TO T[illegible] MATERIAL BALANCE CHANGE.

Figure 111.

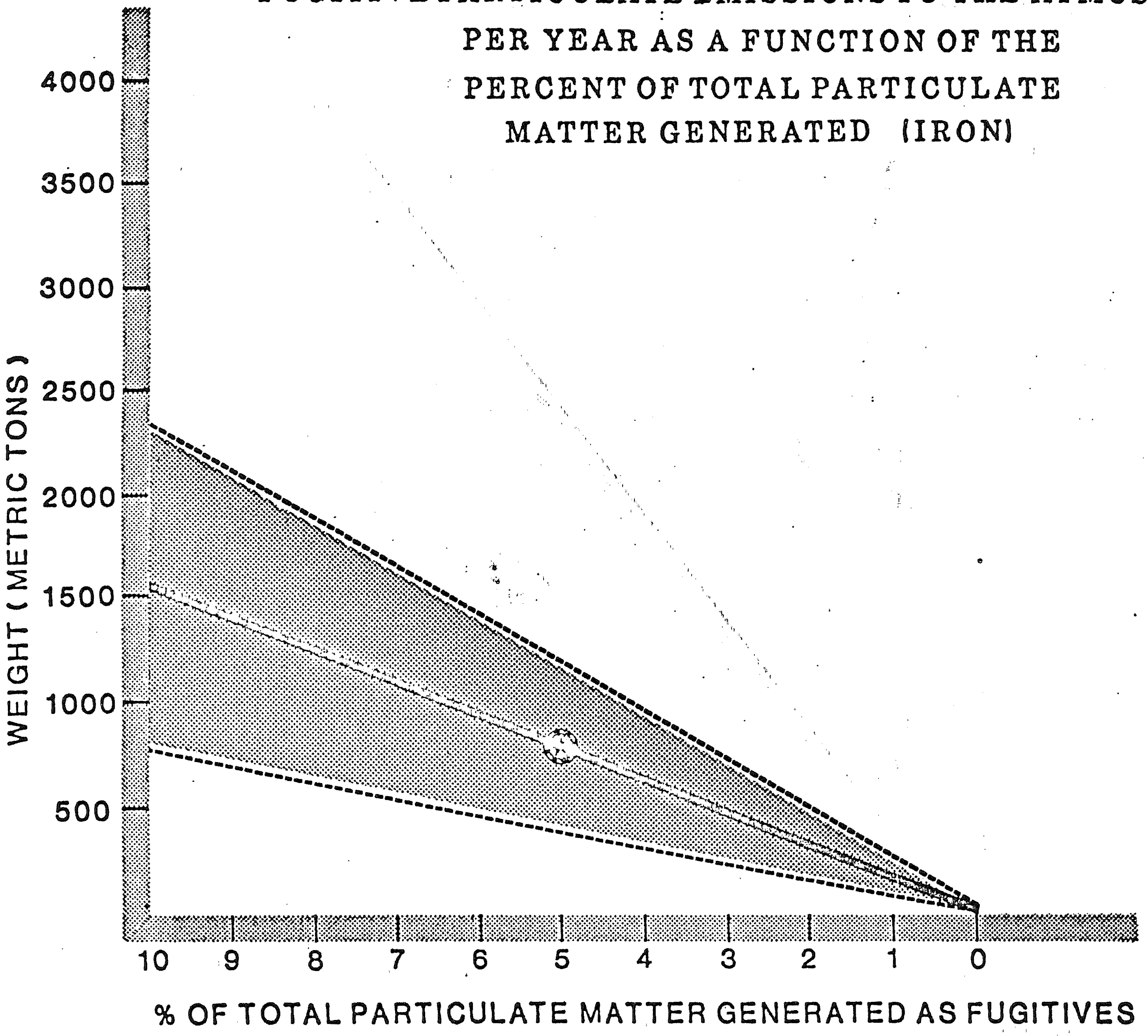
FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (NICKEL)



BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 112.

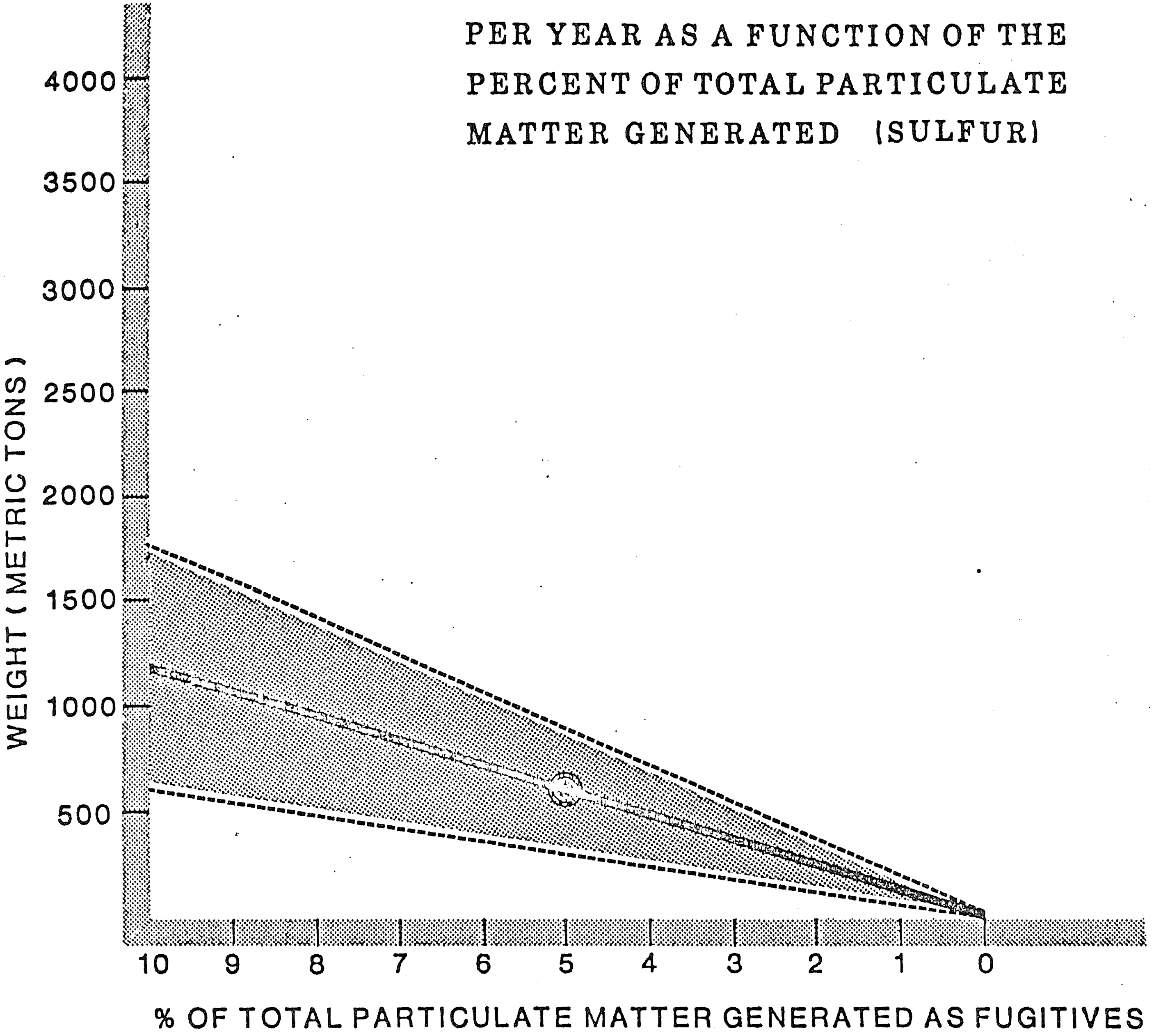


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 113.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (SULFUR)

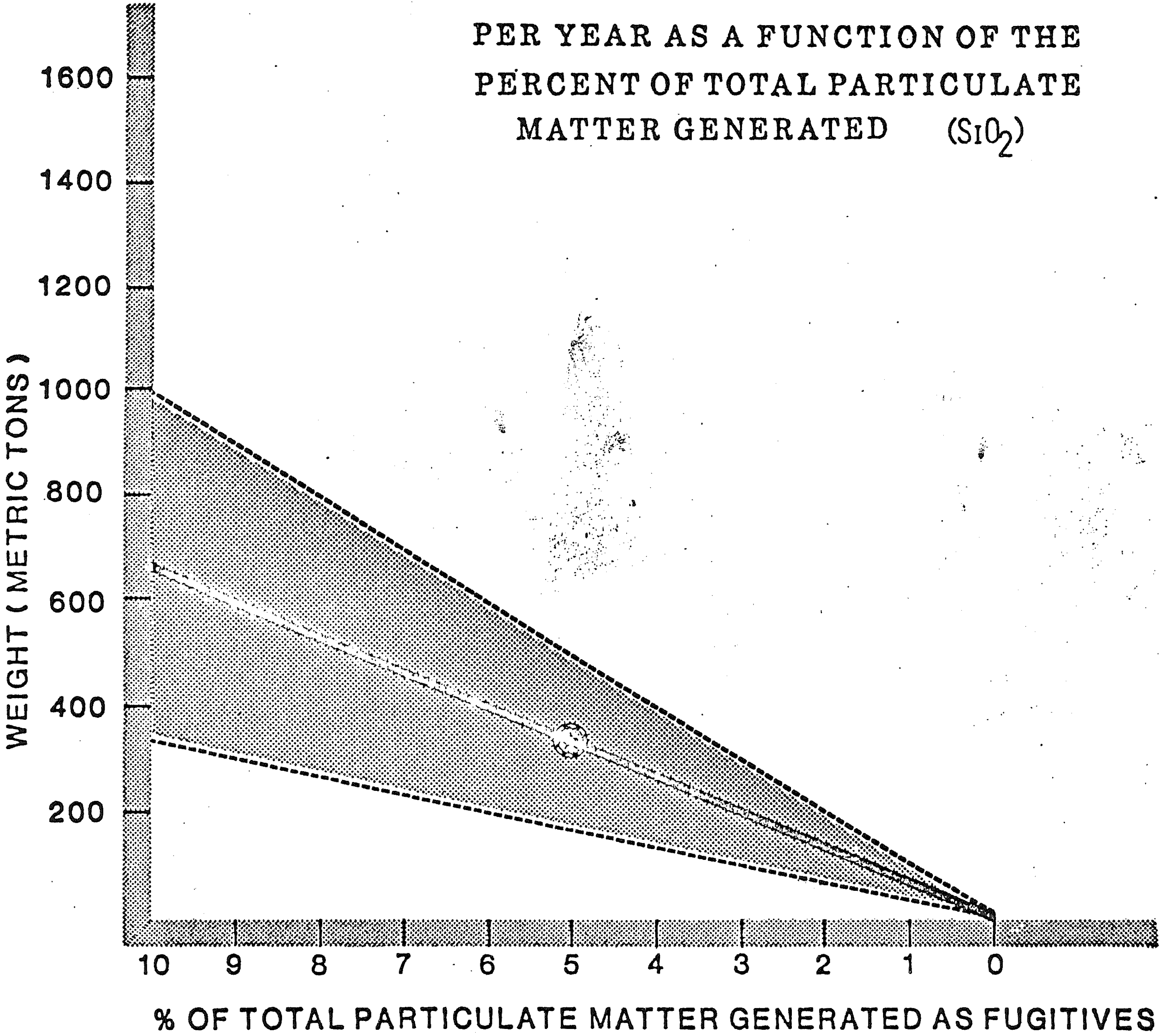


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE
AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 114.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (SiO_2)

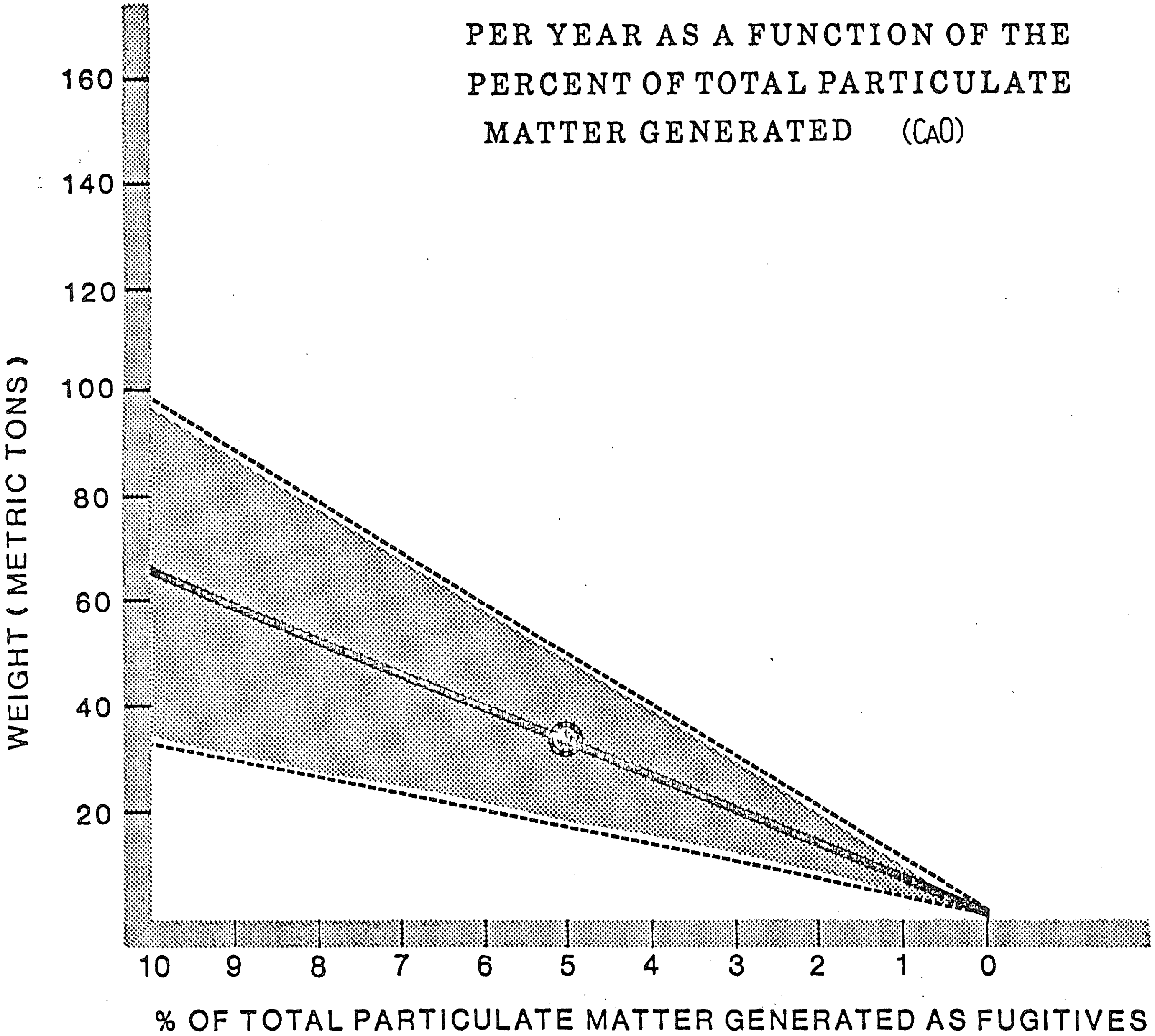


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 115.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (CaO)

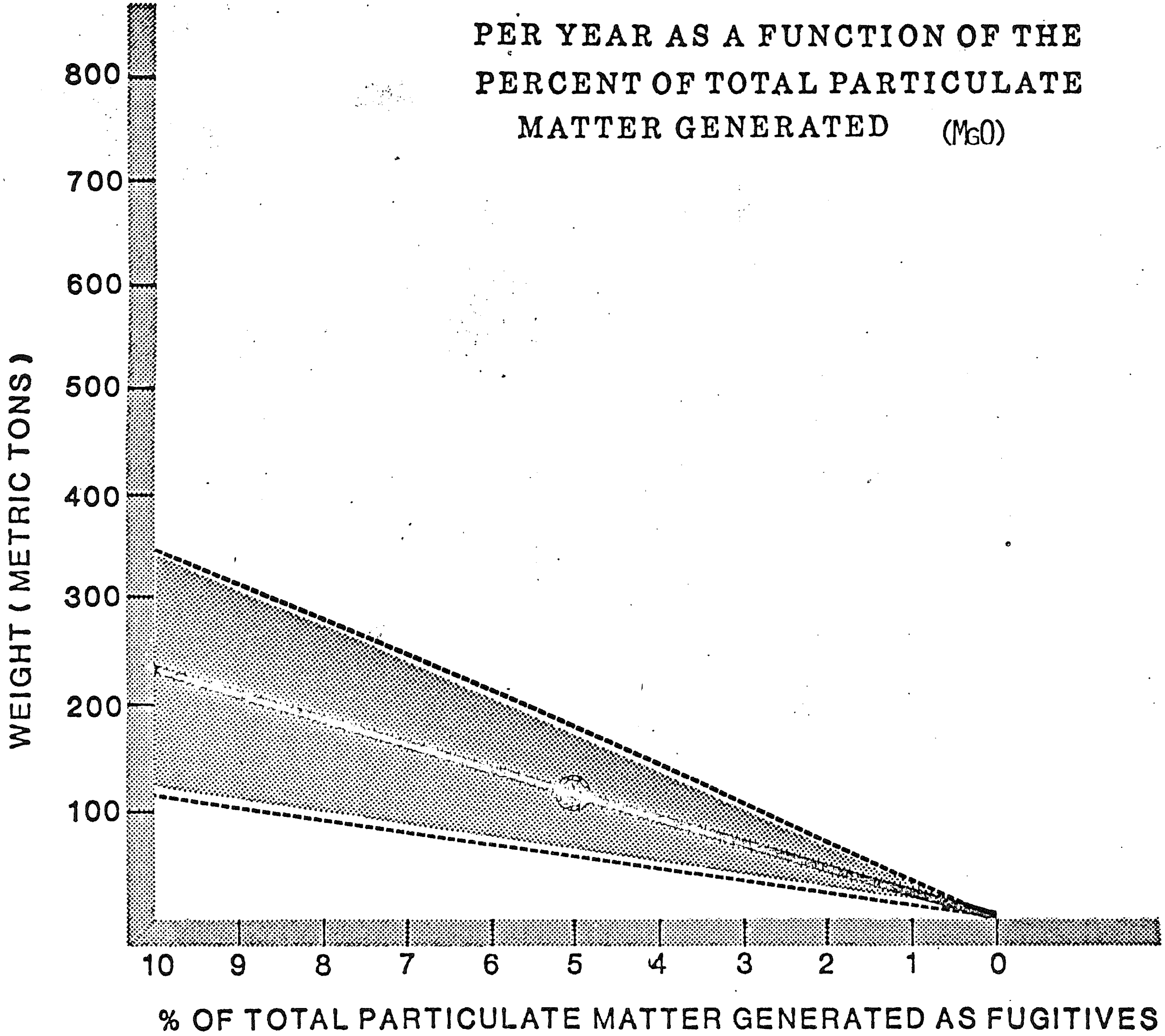


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 116.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (MgO)

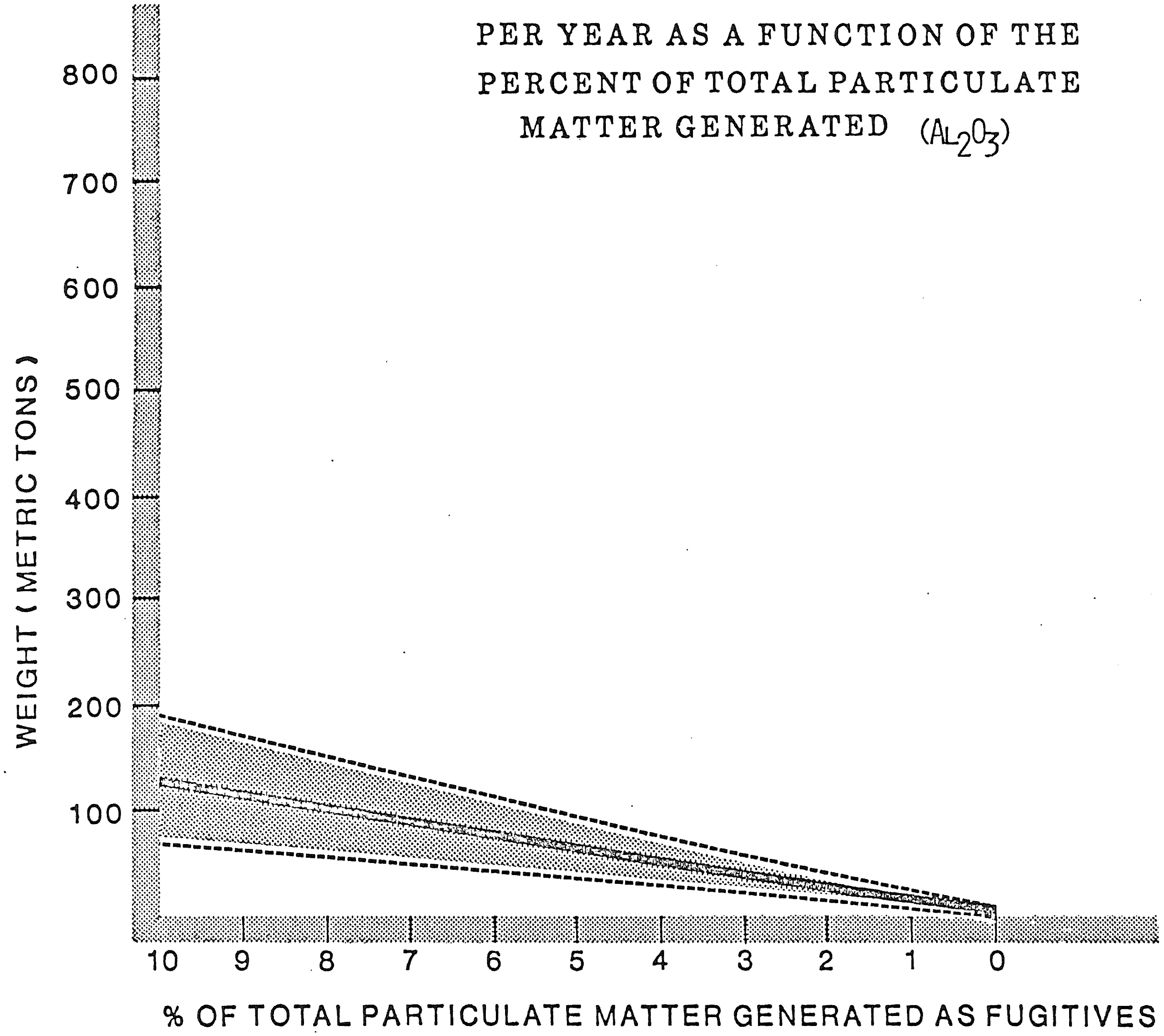


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO TH[illegible] MATERIAL BALANCE CHANGE.

Figure 117.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (AL_2O_3)

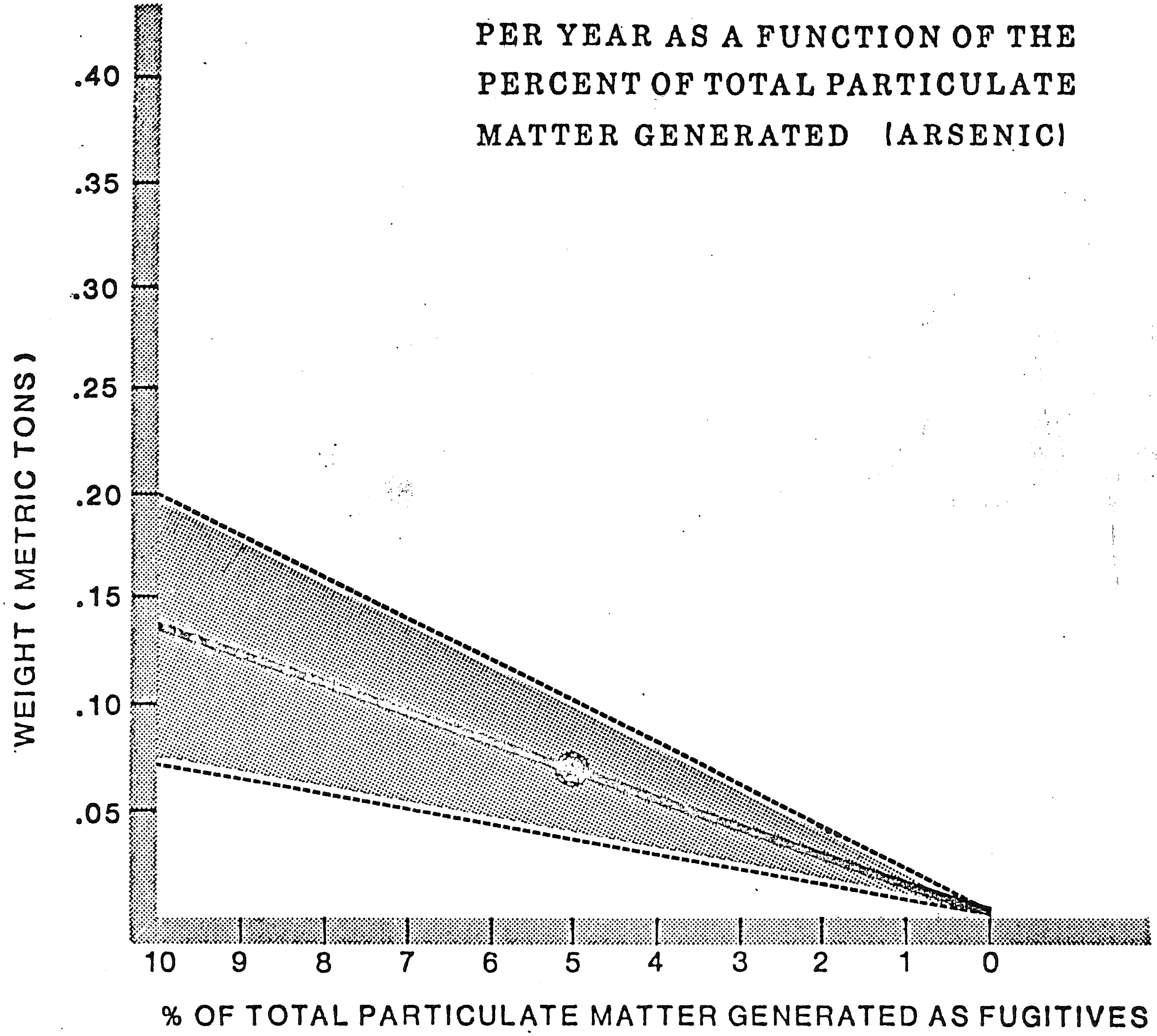


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 118.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (ARSENIC)

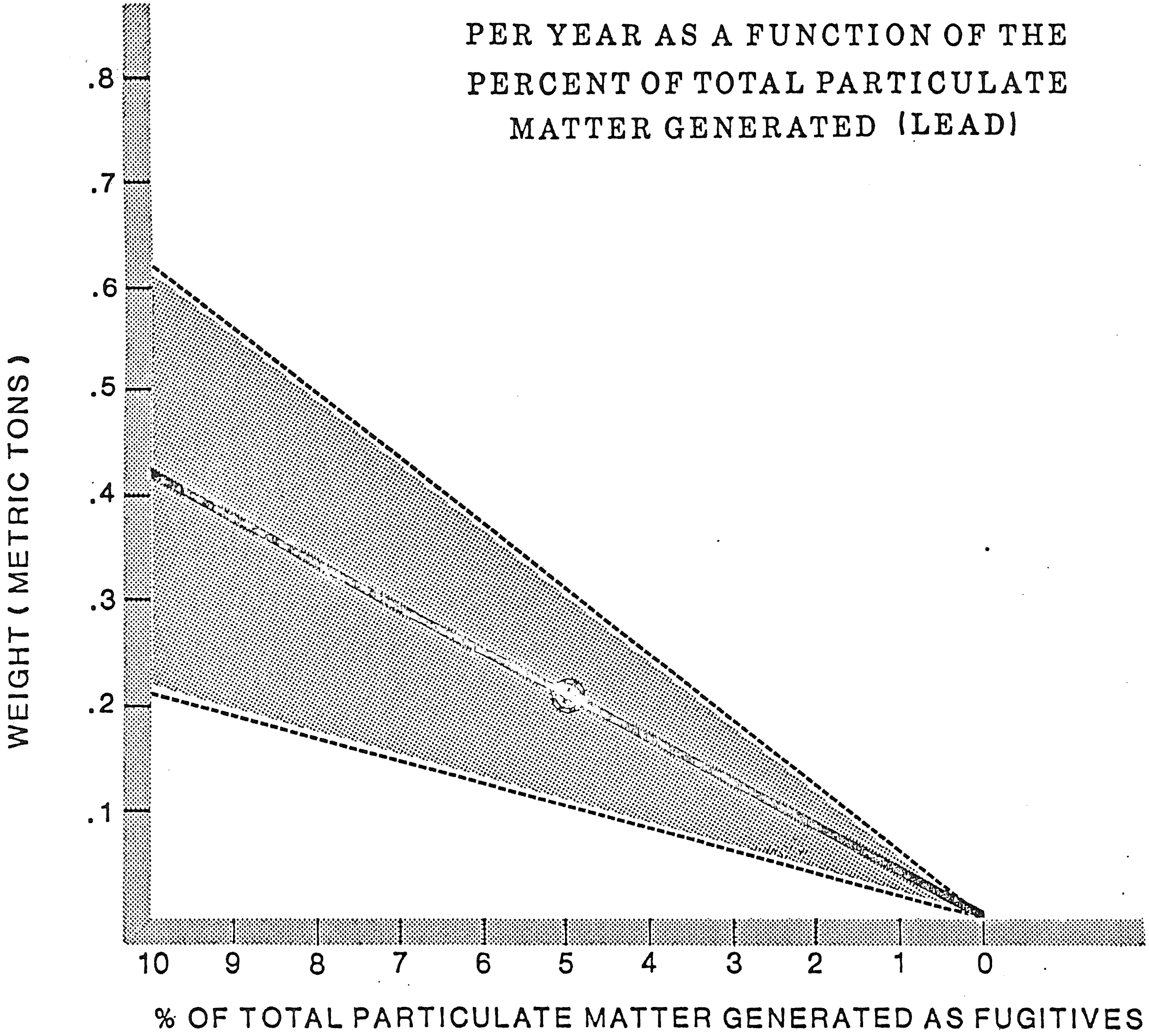


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO T MATERIAL BALANCE CHANGE.

Figure 119.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (LEAD)

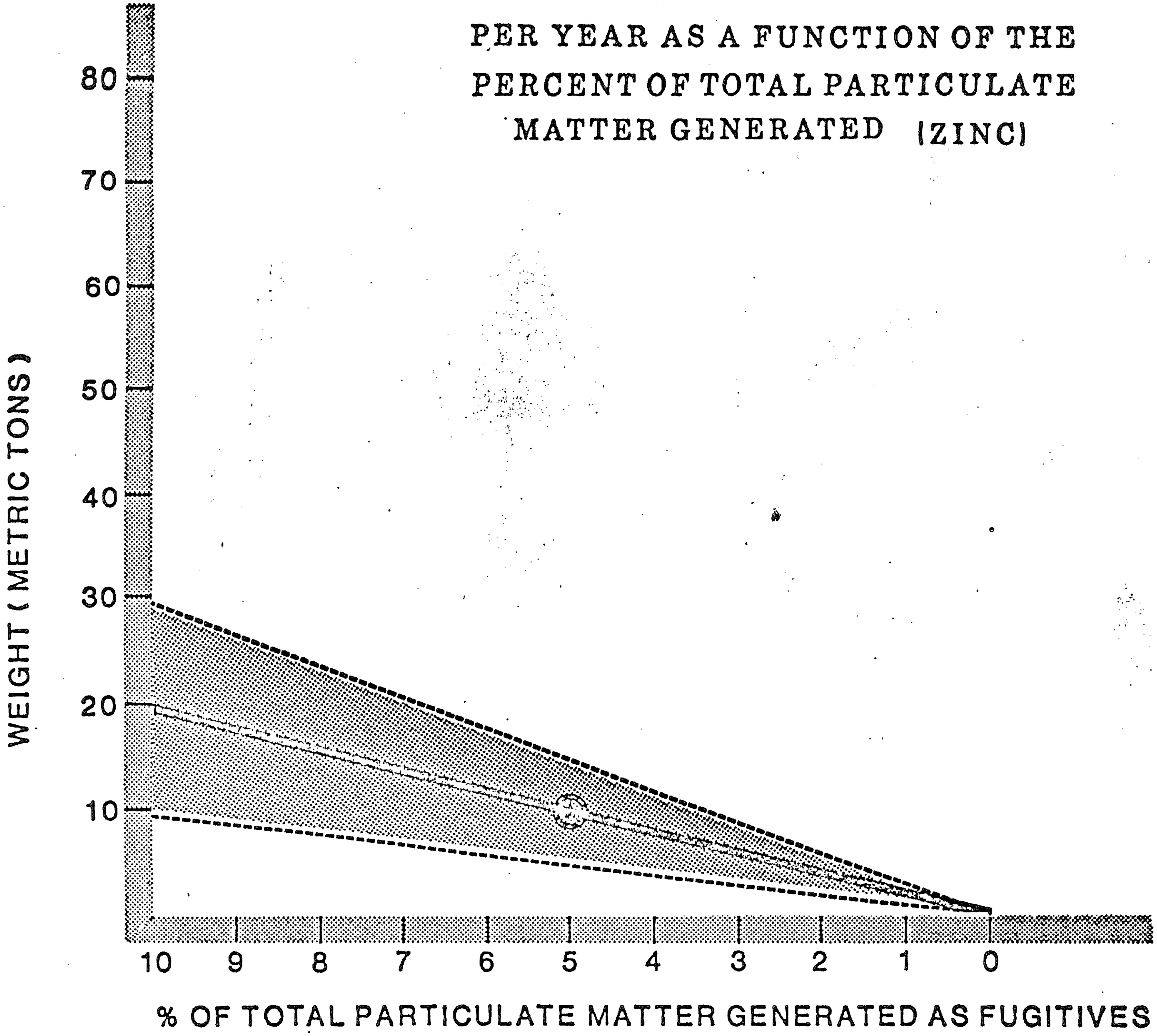


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 120.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (ZINC)

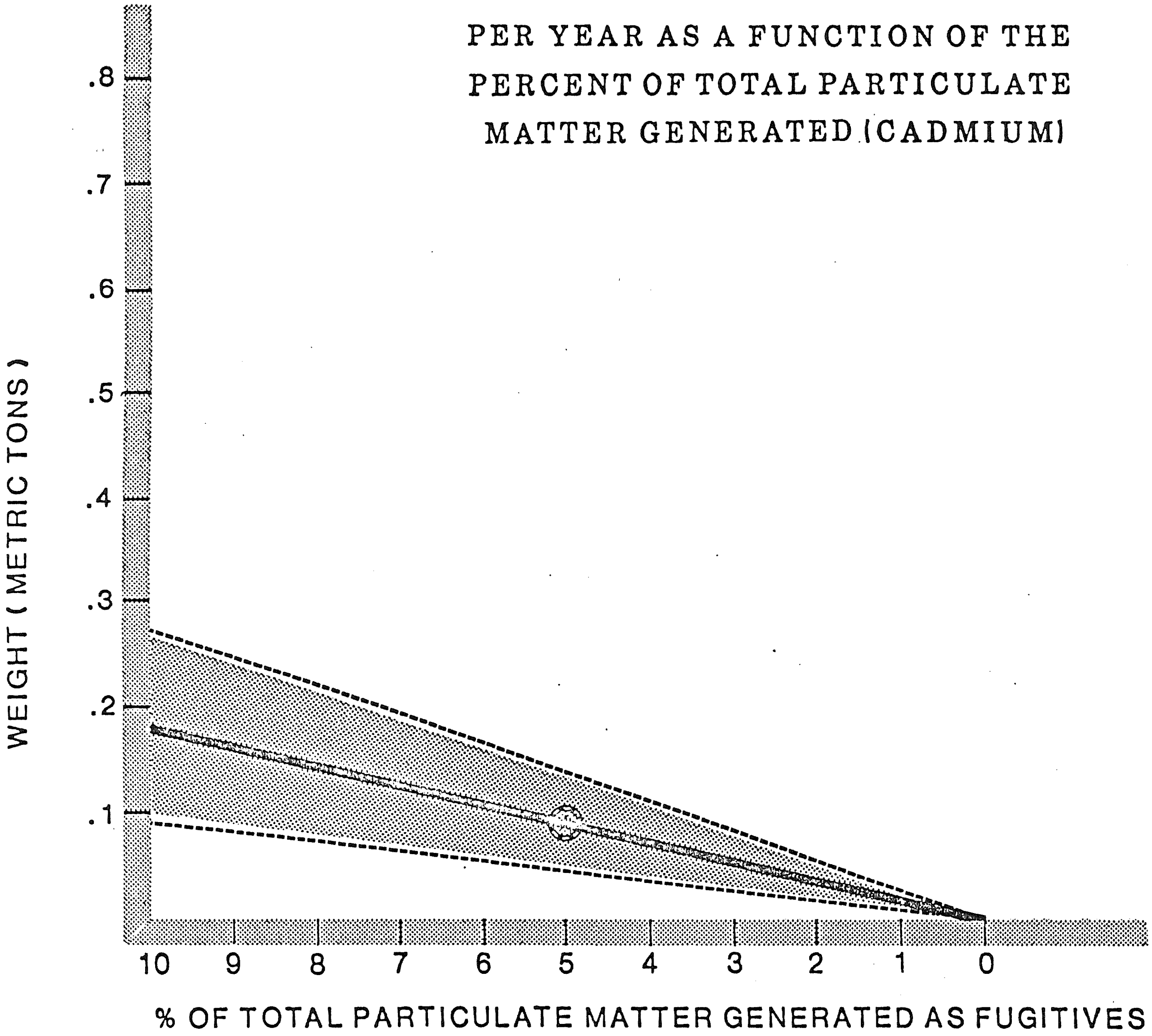


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE
AS THE ASSUMPTIONS TO T MATERIAL BALANCE CHANGE.

Figure 121.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (CADMIUM)

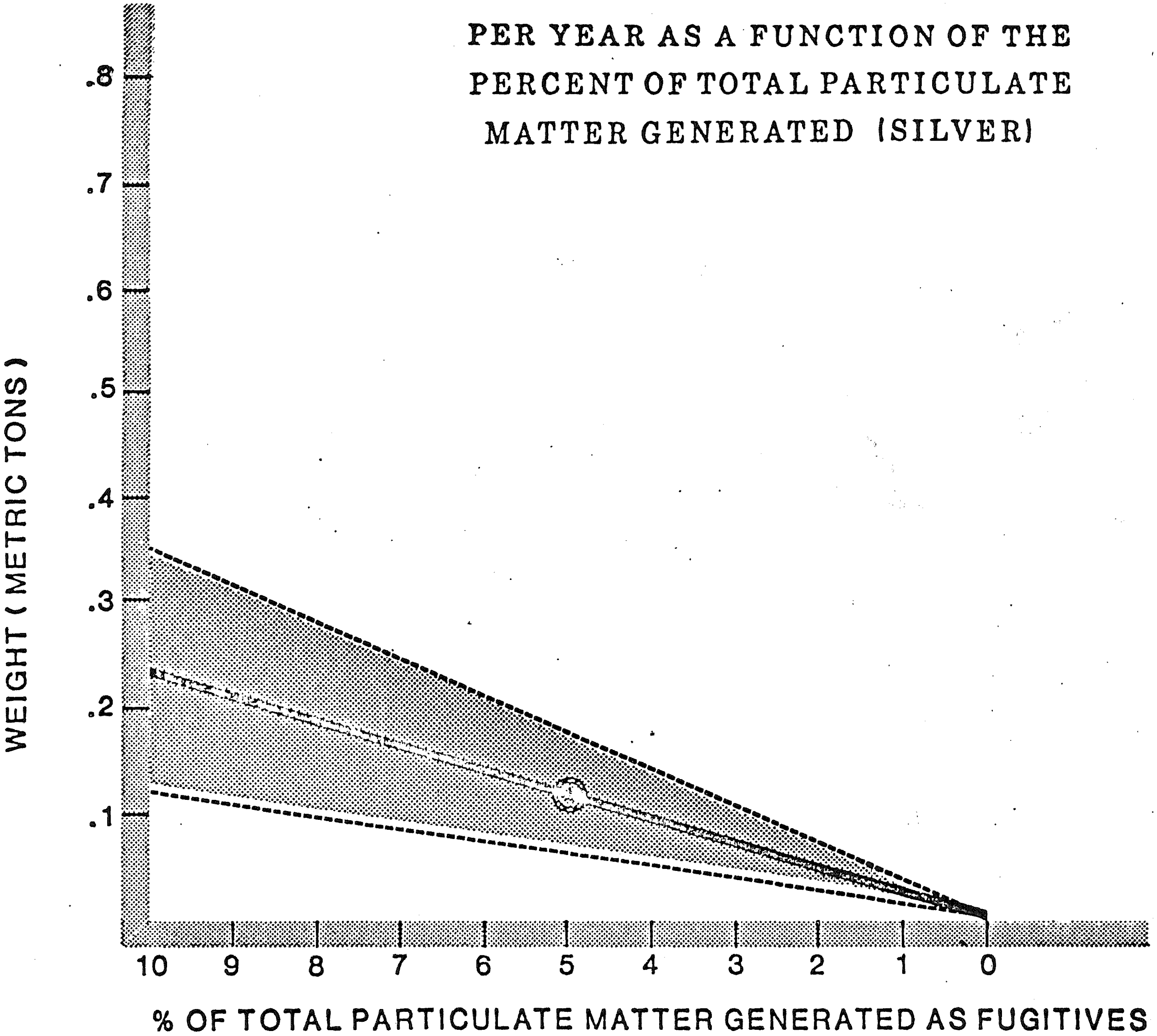


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 122.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (SILVER)



BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Concentrate Smelting

Figure 123 outlines in generalized form the flowsheet for the smelting of copper-nickel concentrates. Concentrate drying is included in this flowsheet so as to show its relationship to concentrate smelting. Figure 124 takes the flow of liquid-solid material from drying, through smelting and intermediate mineral processing, to refining. All numbers (both quantities and chemical analyses) were derived hypothetically using existing information as a guide. Three principal material balances in Figure 124 have been identified as being critical from a material balance point of view, and these balances are included individually in Table 23, and summarized collectively in Table 24. Please refer to Table 25 for a list of the specific assumptions which have been made here. Figures 125 through 144 indicate quantitatively the principal contaminants to the atmosphere as a consequence of smelting copper-nickel concentrates. Fugitive and stack particulate and gaseous emissions are the obvious environmental contaminants. Figures 145, 146, and 147 summarize the combined discharges from the drying and smelting of copper-nickel concentrates (both fugitive and stack, particulate and gas emissions) as a function of copper and nickel production. As the quantity of mined crude ore increases, the undesirable discharges will necessarily also increase.

Sulfur-rich gases discharged from the smelting complex will be treated for sulfur removal. Figure 148 shows how the total SCFM of cleaned discharge gas will vary with: 1) the concentration of SO_2 in the discharge gas; and 2) the total quantity of SO_2 discharged.

Figure 123. Generalized Flowsheet for Concentrate Smelting in the Production of Commercial Grade Copper and Nickel.

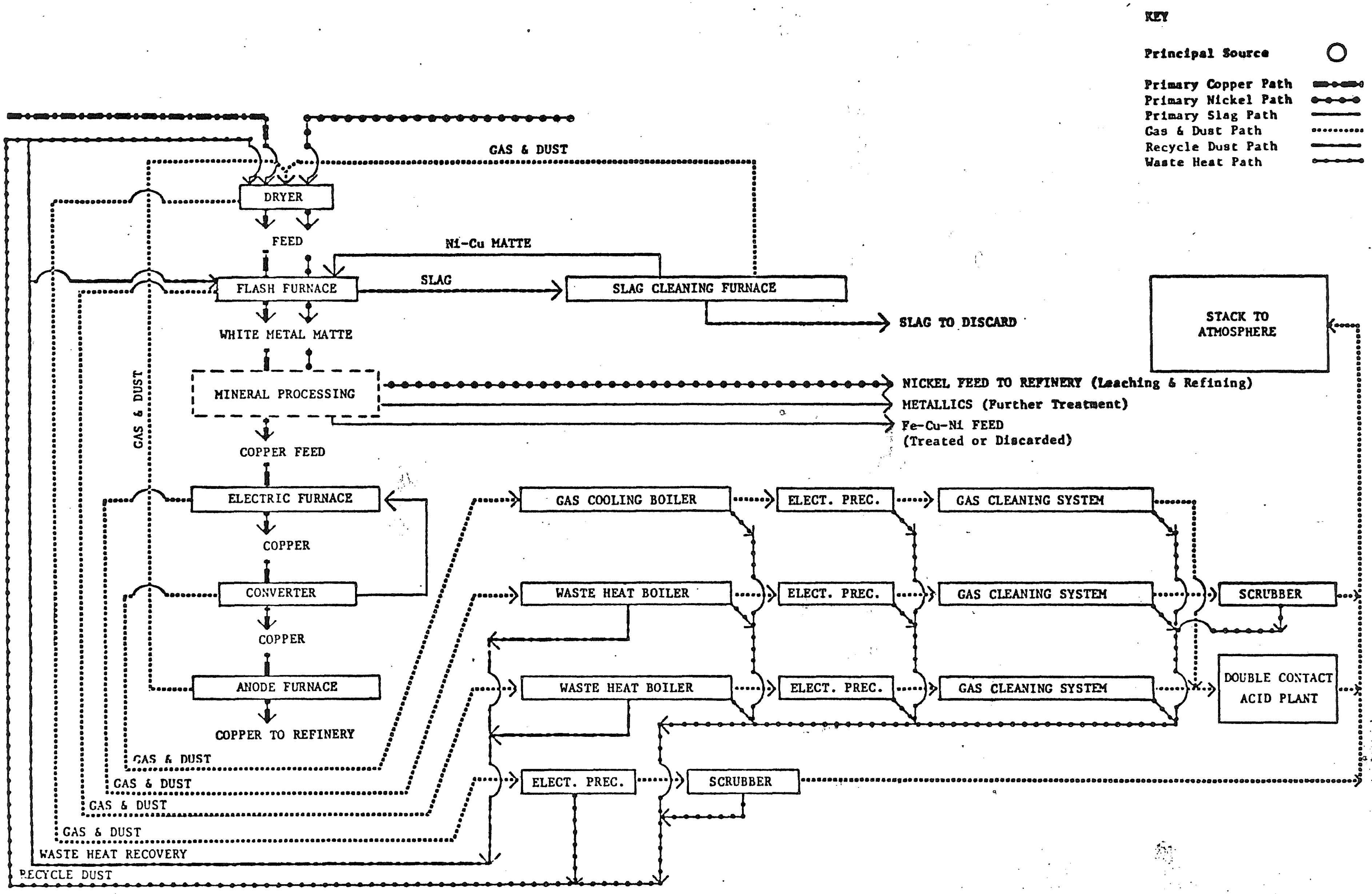
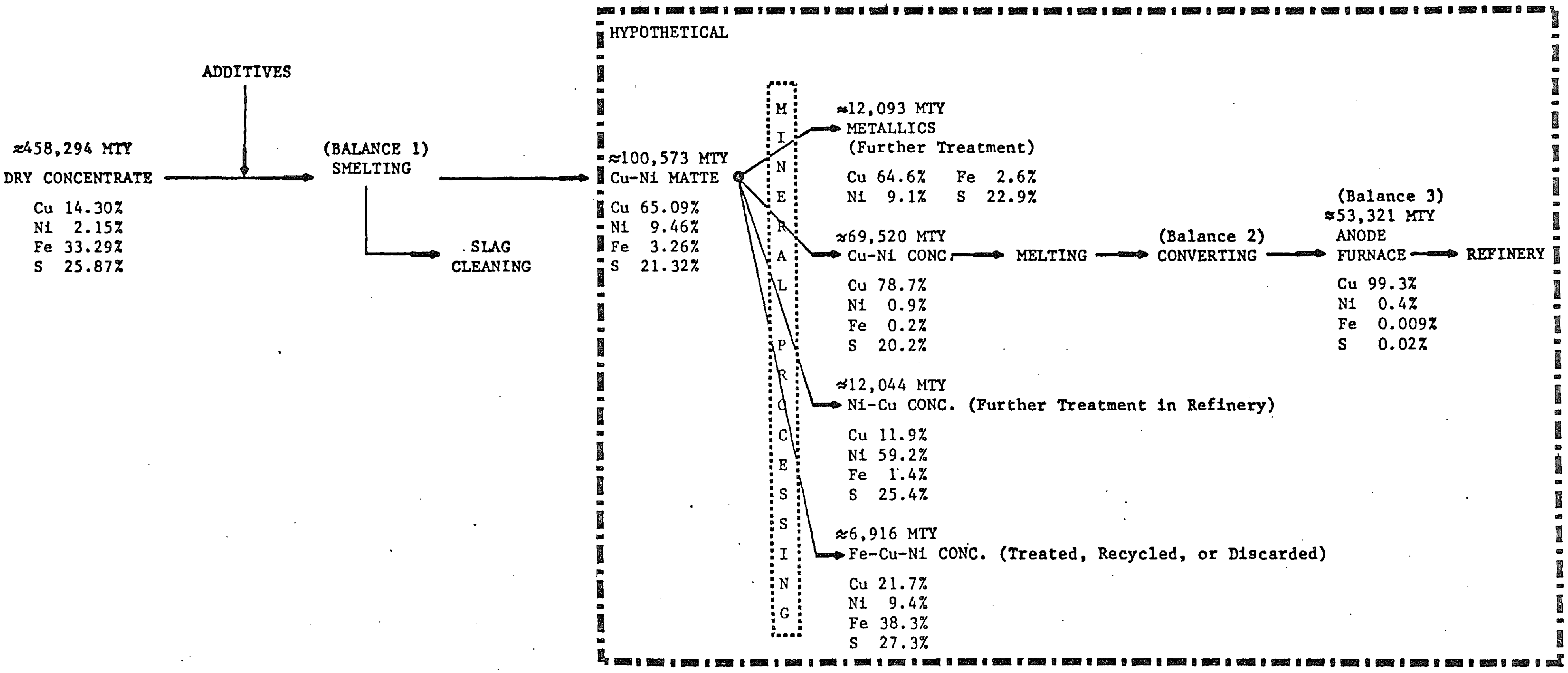


Figure 124. Material Products Flowsheet.



BASIS: ≈100,000 Metric Tons Per Year Production of White Metal Matte.

	Cu	Ni	Fe	S	SiO_2	CaO	MgO	Al_2O_3	As	Pb	Zn	Cd	Ag	Au	Rh	Pt	Pd
INPUT																	
Concentrate	65,536	9,842	152,357	118,560	66,000	6,488	23,221	12,766	13.40	41.25	1,947.3	17.56	23.06	0.71	0.0087	0.88	2.09
Recycle Dust	6,779	1,400	11,193	2,490	9,208				2.64	0.77	54.7	3.45					
Recycle Mat.	7,470	5,850	1,530	2,657													
Flux					61,092												
FEED	79,785	17,092	165,280	123,707	136,300	6,488	23,221	12,766	16.04	42.02	2,002	21.01	23.06	0.71	0.0087	0.88	2.09
MATERIAL BALANCE 1																	
OUTPUT																	
White Metal Matte	65,462	9,516	3,275	21,442					0.64	26.50	681	1.68	22.26	0.69	0.008	0.85	2.01
Recycle Mat.	7,217	5,710	1,421	2,472													
Dump Slag	1,207	604	148,337	1,730	126,203	6,488	23,221	12,766	0.97	11.32	1,021	0.42	0.80	0.02	0.0007	0.03	0.08
Recycle Dust	5,380	1,151	11,170	2,130	9,208				2.64	0.77	54.7	3.45					
Contained Dust									10.52	3.06	218.9	13.79					
STACK EMISSIONS																	
Particulate	224	48	465	89	384				0.55	0.16	11.4	0.72					
Gas				4,757													
FUGITIVE EMISSIONS																	
Particulate	295	63	612	117	505				0.72	0.21	15.0	0.95					
Gas				961													
MATERIAL BALANCE 2																	
INPUT																	
Cu Concentrate	54,712	626	139	14,043													
OUTPUT																	
Blister Copper	53,196	470	27	54													
Recycle Mat.	253	140	109	185													
Recycle Dust	1,155	14.6	3.4	321													
STACK EMISSIONS																	
Particulate	45	0.4	0.2	13													
Gas				666													
FUGITIVE EMISSIONS																	
Particulate	63	1	0.2	17													
Gas				134													
MATERIAL BALANCE 3																	
INPUT																	
Blister Copper	53,196	470	27	54													
OUTPUT																	
Anode Copper	52,929	212	5	11													
Recycle Dust	243.8	235.2	20.06	39.2													
STACK EMISSIONS																	
Particulate	10.2	9.8	0.84	1.6													
FUGITIVE EMISSIONS																	
Particulate	13.0	13.0	1.1	2.2													

BASIS: 100,000 Metric Tons Per Year Production of White Metal Matte.

Table 23. Metallurgical Material Balance of the Principal Constituents from the Flash Smelting of Cu-Ni Concentrate (Quantities are in Metric Tons Per Year).

	Cu[1]	Ni	Fe	S	SiO_2	CaO	MgO	Al_2O_3	As	Pb	Zn	Cd	Ag	Au	Rh	Pt	Pd
<u>INPUT</u>																	
Concentrate	65,536[2] 82.1[3] 14.30[4]	9,842 57.6 2.15	152,557 92.3 33.29	118,360 95.8 25.87	66,000 48.4 14.40	6,488 100 1.42	23,221 100 5.07	12,766 100 2.79	13.40 83.5 0.0029	41.25	1,947.2	17.56	23.06 100	0.71 100	0.0087 100 0.0000019	0.88 100 0.00019	2.09 100 0.00045
Recycle Dust	6,779 8.5	1,400 8.2	11,193 6.8	2,490 2.0	9,208 6.8				2.64 16.5	0.77 1.8	54.7 2.7	3.45 16.4					
Recycle Mat.	7,470 9.4	5,850 34.2	1,530 0.9	2,657 2.2													
Flux					61,092 44.8												
<u>INPUT FEED</u>	79,785 100 14.0	17,092 100 3.0	165,280 100 29.0	123,707 100 21.71	136,300 100 23.92	6,488 100 1.14	23,221 100 4.07	12,766 100 2.24	16.04 100 0.003	42.02 100 0.01	2,002. 100 0.35	21.01 100 0.004	23.06 100 0.004	0.71 100 0.0001	0.0087 100 0.000002	0.88 100 0.0002	2.09 100 0.0004
<u>OUTPUT</u>																	
Metallics	7,812 9.8 64.6	1,103 6.5 9.1	320 0.2 2.6	2,765 2.2 22.9							89 4.5 0.74			0.70 98.6 0.0058	0.0081 93.1 0.000067	0.86 97.7 0.0071	2.02 96.6 0.017
Anode Cu	52,929 66.3 99.3	212 1.2 0.4	5 0.003 0.009	11 0.009 0.02					0.64 4.0 0.0012	13.0 30.9 0.024	138 6.9 0.26	0.80 3.8 0.0013	11.13 48.3 0.021				
Ni-Cu Conc.	1,439 1.8 11.9	7,134 41.7 59.2	165 0.01 1.4	3,054 2.5 25.4						13.0 30.9 0.11	227 11.3 1.9	0.80 3.8 0.0066	11.13 48.3 0.092				
Fe-Cu-Ni Conc.	1,499 1.9 21.7	653 3.8 9.4	2,650 1.6 38.3	1,885 1.5 27.3						0.5 1.2 0.0072	227 11.3 3.3	0.08 0.4 0.016	0.1 0.4 0.0014				
Slag	1,207 1.5 0.3	604 3.5 0.15	148,337 89.7 36.8	1,730 1.4 0.43	126,203 92.6 31.31	6,488 100 1.61	23,221 100 5.76	12,766 100 3.17	0.97 6.0 0.0002	11.32 26.9 0.003	1,021 51.0 0.25	0.42 2.0 0.0001	0.7 3.0 0.0002	0.01 1.4 0.000002	0.0006 6.9 0.0000001	0.02 2.3 0.000005	0.07 3.4 0.00002
Fugitive Emis.																	
Particulate	371 0.5	77 0.5	614 0.4	136 0.1	505 0.4				0.72 4.5	0.21 0.5	15.0 0.8	0.95 4.5					
Gas				1,087 0.9													
Stack Emis.																	
Particulate	279 0.3	58.2 0.3	466 0.3	104 0.08	384 0.3				0.55 3.4	0.16 0.4	11.4 0.6	0.72 3.4					
Gas				542.5 4.4													
Recycle Dust	6,799 8.5	1,400 8.2	11,193 6.8	2,490 2.0	9,208 6.7				2.64 16.5	0.77 1.8	54.7 2.7	3.45 16.4					
Contained Dust									10.52 65.6	3.66 7.4	218.9 10.9	13.79 65.7					
Recycle Mat.	7,470 9.4	5,850 34.3	1,530 0.9	2,657 2.2													
Sulfur Converted to Acid				102,363 82.7													

BASIS: 100,000 Metric Tons Per Year Production of White Metal Matte.

[1] Element
[2] Metric Tons of Element
[3] Element Distribution
[4] Chemical Analysis of Input-Output Product

Table 24. Metallurgical Material Balance Summary (Quantities are in Metric Tons Per Year).

Table 25. Table of "Smelting" Assumptions.

- The input feed concentrate has the following chemical composition:

Element	Wt %	Element	Wt %
Cu	14.3	Pb	≈ 0.000
Ni	2.14	Zn	≈ 0.42
Fe	33.2	Cd	≈ 0.0058
S	25.8	Ag	≈ 0.005
SiO_2	14.3	Au	≈ 0.00015
CaO	1.4	Rh	≈ 0.0000019
MgO	5.0	Pt	≈ 0.00019
Al_2O_3	2.8	Pd	≈ 0.00045
As	≈ 0.0029		

- Converter slag and portions of captured flue dust are recycled and added as feed to smelter.

- Exhaust gases contain some molten or semi-molten dust particles. Volatile compounds tend to be concentrated in the dust. Gases plus dust are assumed to be a homogeneous mixture at the exit port of each emission location.

- Individual dust particles emitted at the various point source locations within a smelting complex assume the same chemical composition as the material from which it is derived. Dusts generated from the flue bed reactors assume the same chemical analysis as the produce calcine, dusts generated from the smelting furnace assume the same chemical analysis as the metal matte, as so on.

- Zinc: major portion of this constituent is oxidized and distributed into the slage. A portion is partly volatilized.

85% goes into the slag	(true for concentrates high in zinc)
15% removed as dust	(volatilization rate is ≈20-30%)

- Lead: major portion goes into the matte. PbS in the matte is then oxidized partly to PbO in the converting furnace and distributed into the slag.

90% goes into the matte	(true for concentrates high in lead)
10% removed as dust	(volatilization rate is ≈60%)

- CaO, MgO, Al_2O_3: these constituents are principally distributed into the slag.

- Arsenic: 90 percent of the total amount present is removed as dust or particulates (volatilization rate is ≈10%).

- Cadmium: 90 percent of the total amount present is removed as dust or particulates (volatilization rate is ≈10%).

- Cobalt: approximately 70 percent of the total amount present goes into the matte. Thirty percent is removed as dust or particulates (volatilization rate is ≈10%).

- It is assumed that some flue dust is more easily collected then others. The ease of collection is dependent upon where dust accumulates in the system. Also, it is assumed that some dusts or particulates are specially treated for removal of certain constituents. It is assumed that ≈73 percent of the As, Pb, Zn, and Cd is either:

  1) contained or isolated in the system where collection occurs less frequently, or

  2) removed by special treatment prior to recirculation.

- Five percent of the total dust generated in the smelting system(s) is considered to be fugitive, as as such, is never treated for removal.

- Ninety-six percent of the dust generated (excluding fugitive dust) is treated and recovered and returned for use in the smelter. The remaining four percent is discharged as a stack emission.

- One percent of the total sulfur generated as SO_2 is considered to be fugitive, as as such is never treated for removal.

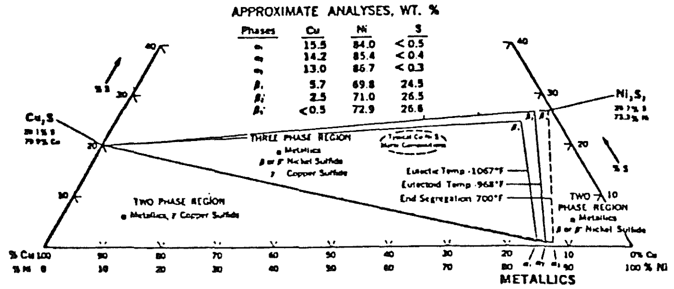
- Ninety-five percent of the sulfur generated as SO_2 (excluding fugitive sulfur emissions as SO_2) is treated for removal as sulfuric acid and sold as a byproduct. The remaining five percent is discharged as a stack emission.

- Ninety-six percent (96%) removal efficiency on particulate cleaners is assumed. Deviations from this efficiency will dramatically change particulate emissions.

- Ninety-five percent (95%) removal efficiency of SO_2 in the sulfuric acid plant is assumed. Deviations from this efficiency will dramatically change gas emissions.

- Assume Cu, Ni, Fe, S in the matte are as Cu_2S, Ni_3S_2, Fe_7S_8, and Fe. Assume in flotation 95 percent recovery of Cu_2S in a Cu-concentrate, 85 percent recovery of Ni_3S_2 in a Ni-concentrate, and 85 percent recovery of Fe_7S_8 in a Fe-concentrate. The remainder of each constituent is split equally among the three concentrates. The analysis of the three concentrates is determined after summation. Assume 12,000 metric tons per year of metallics will form from every 100,000 metric tons of white metal matte averaging 63 to 68 percent Cu. The analysis of the metallics is determined by difference (% $\text{Element}_{\text{metallics}}$ = % $\text{Element}_{\text{white metal}}$ - % $\text{Element}_{\text{Cu conc.}}$ - % $\text{Element}_{\text{Ni conc.}}$ - % $\text{Element}_{\text{Fe conc.}}$) (Figure 3).

Figure 3. Cu-Ni Equilibrium Phase Diagram. (Source: Boldt, 1967)



APPROXIMATE ANALYSES, WT. %

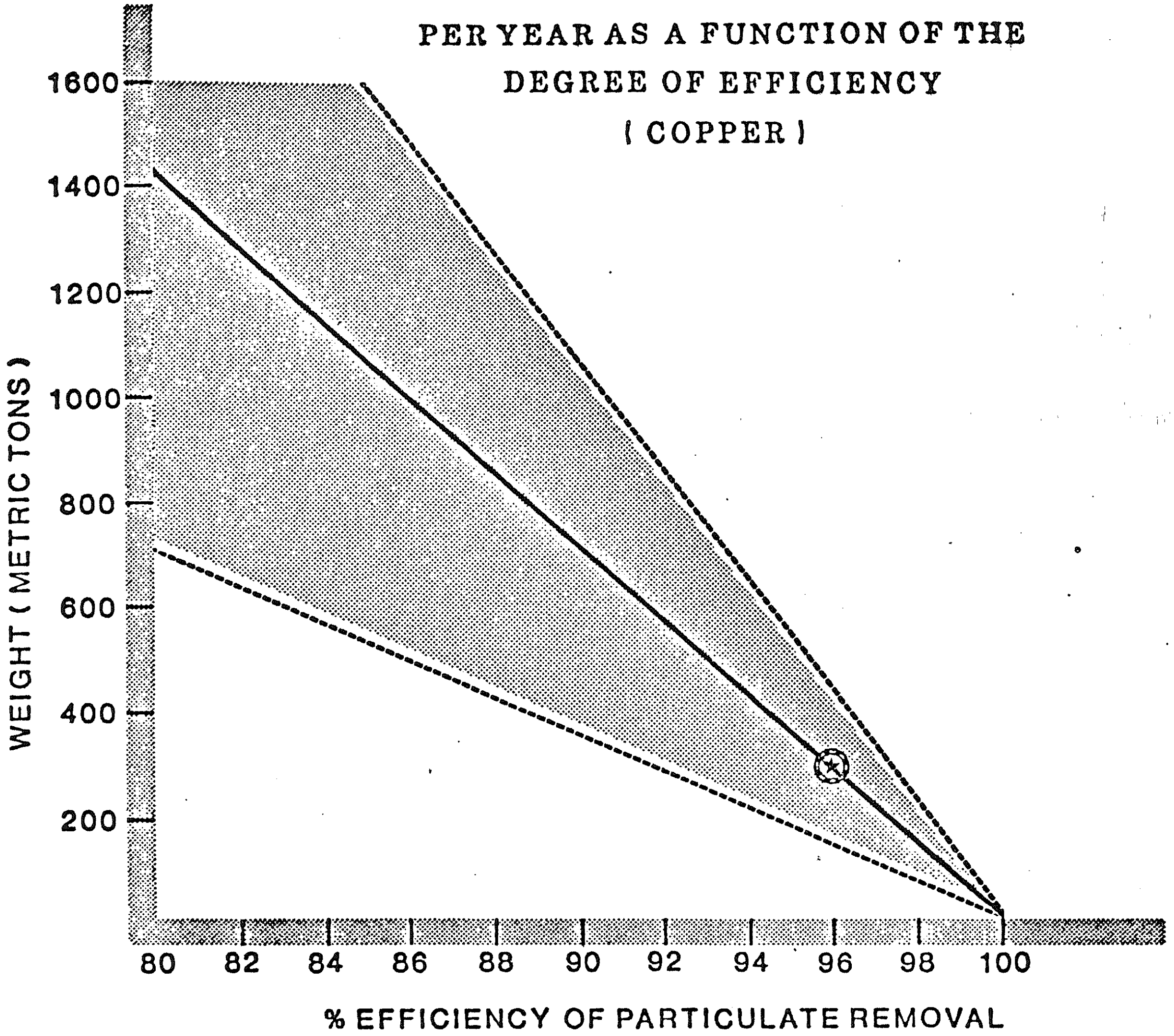
Phases	Cu	Ni	S
α_1	15.5	84.0	< 0.5
α_2	14.2	85.4	< 0.4
α_3	13.0	86.7	< 0.3
β_1	5.7	69.8	24.5
β_2'	2.5	71.0	26.5
β_3'	<0.5	72.9	26.6

This equilibrium phase diagram is a concise graphical representation of the amounts and compositions of the solid phases that crystallize from a nickel-copper-sulfur melt held at a particular temperature until all change ceases. All mattes with compositions that fall within the boundaries of the center triangles will contain three separate distinct phases, the compositions of which may be read from the corners of the appropriate triangle. At the ternary eutectic temperature of 1067°F the three solid phases co-existing are: copper sulfide of 20 per cent sulfur and less than 0.5 per cent nickel; metallic phase (α) of about 15.5 per cent copper, less than 0.5 per cent sulfur, and about 84 per cent nickel; and nickel sulfide (β) of about 6 per cent copper. At the eutectoid temperature of 968°F the copper sulfide and metallic phases have changed little, but the nickel sulfide undergoes a transformation to a low temperature form (β') in which the solubility of copper is only about 2.5 per cent. At 700°F, β' contains less than 0.5 per cent copper.

Figure 125.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY

(COPPER)



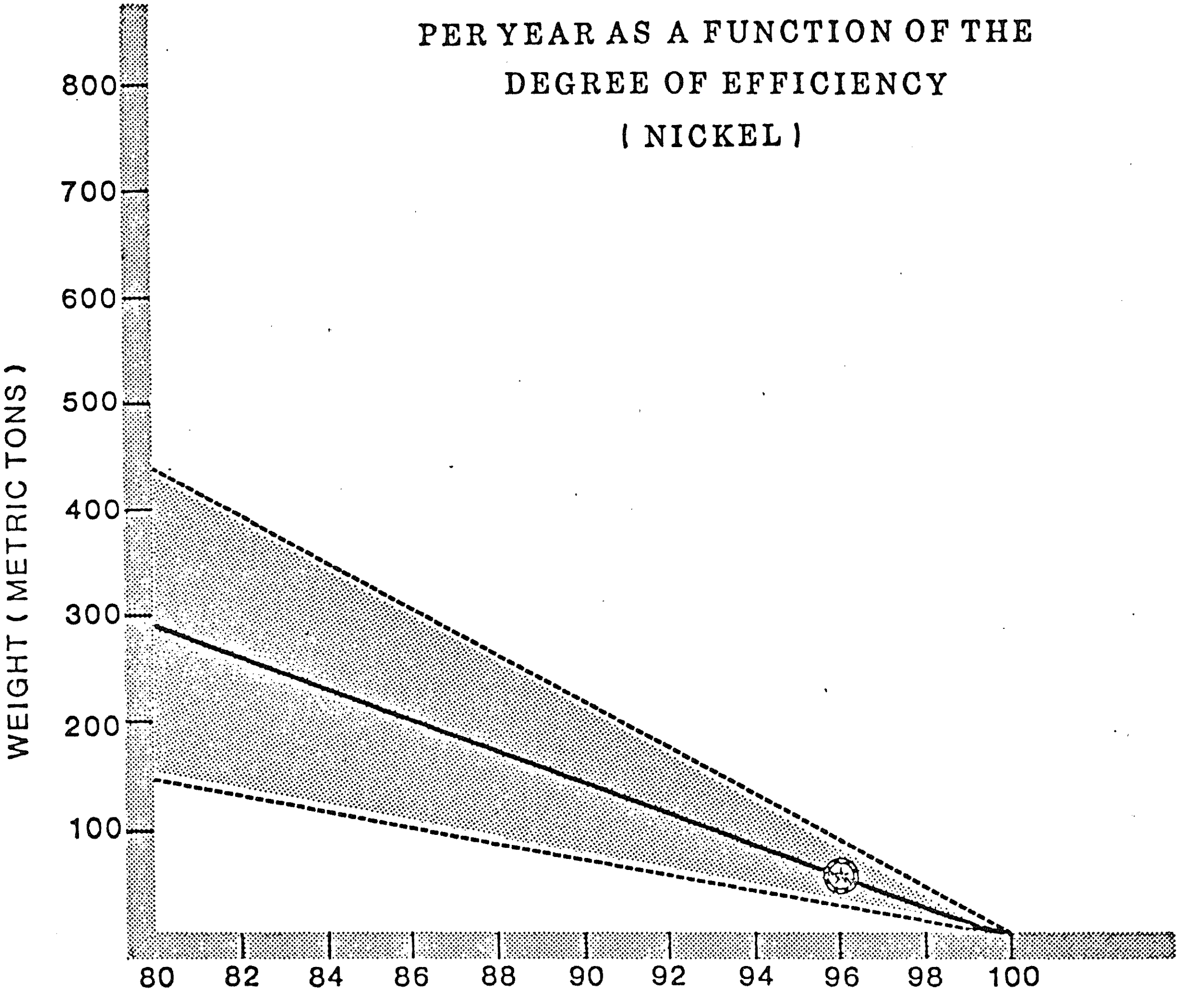
BASIS: 100.000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 126.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY

(NICKEL)



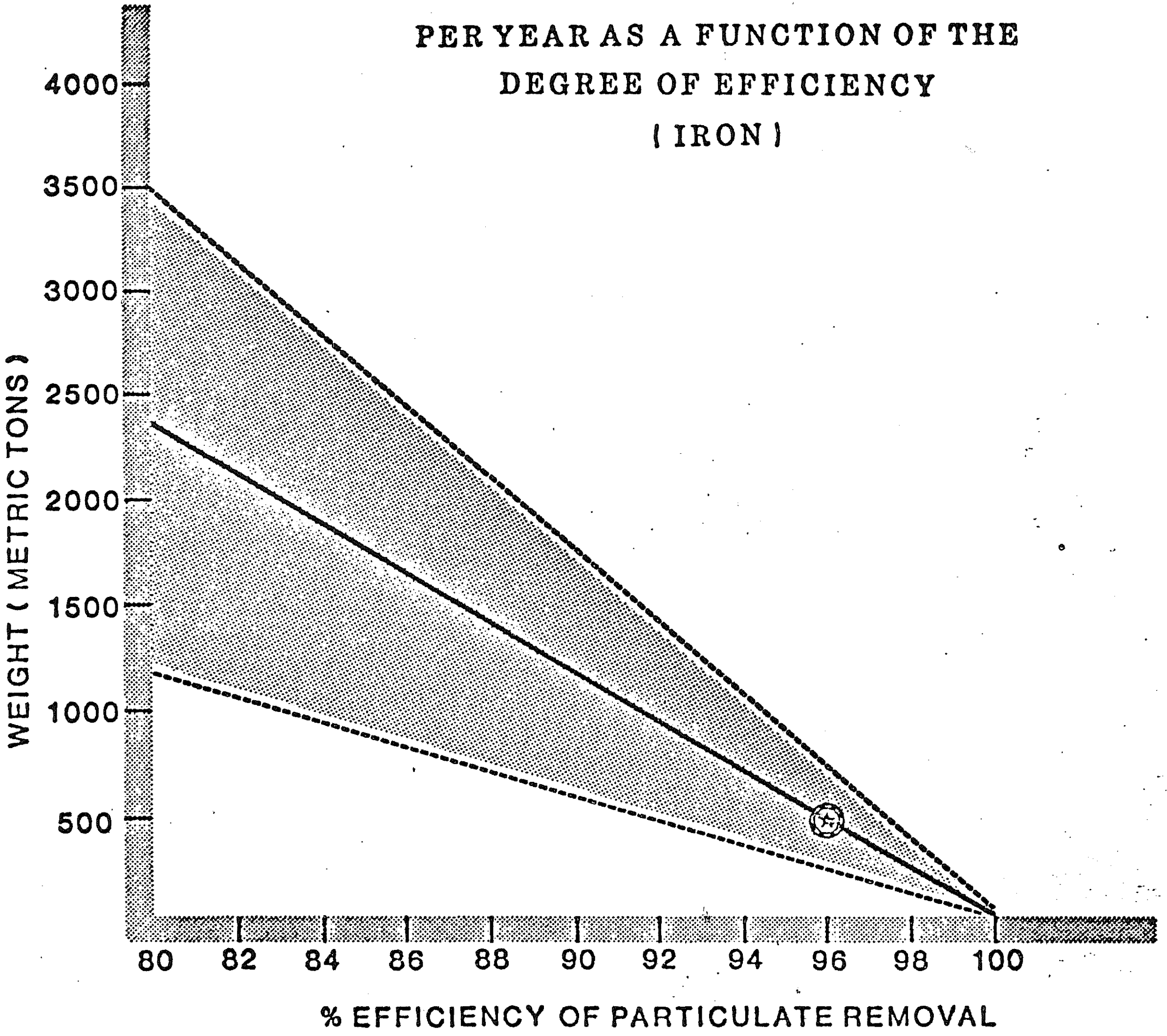
% EFFICIENCY OF PARTICULATE REMOVAL

BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 127.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY (IRON)

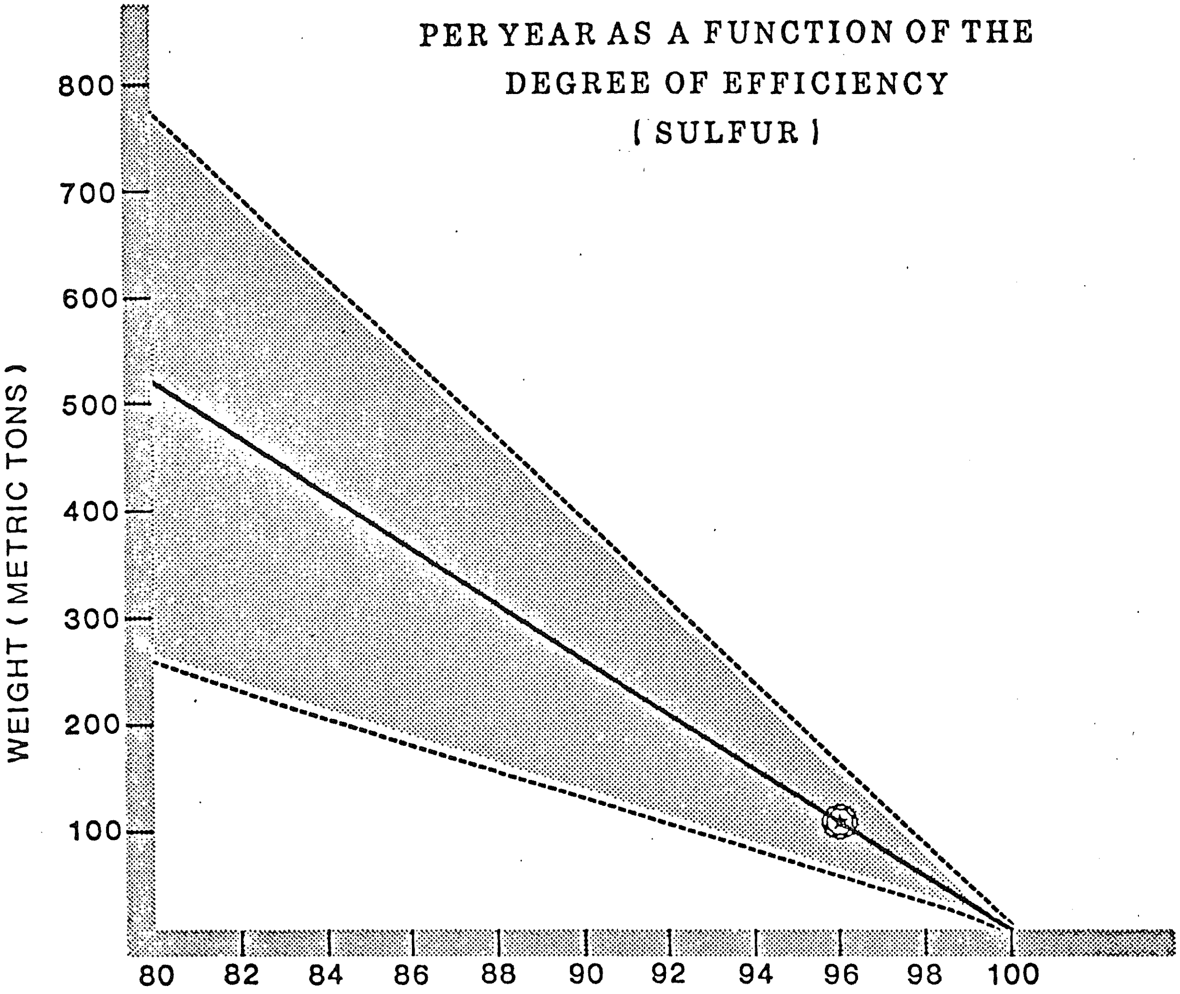


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 128.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY (SULFUR)



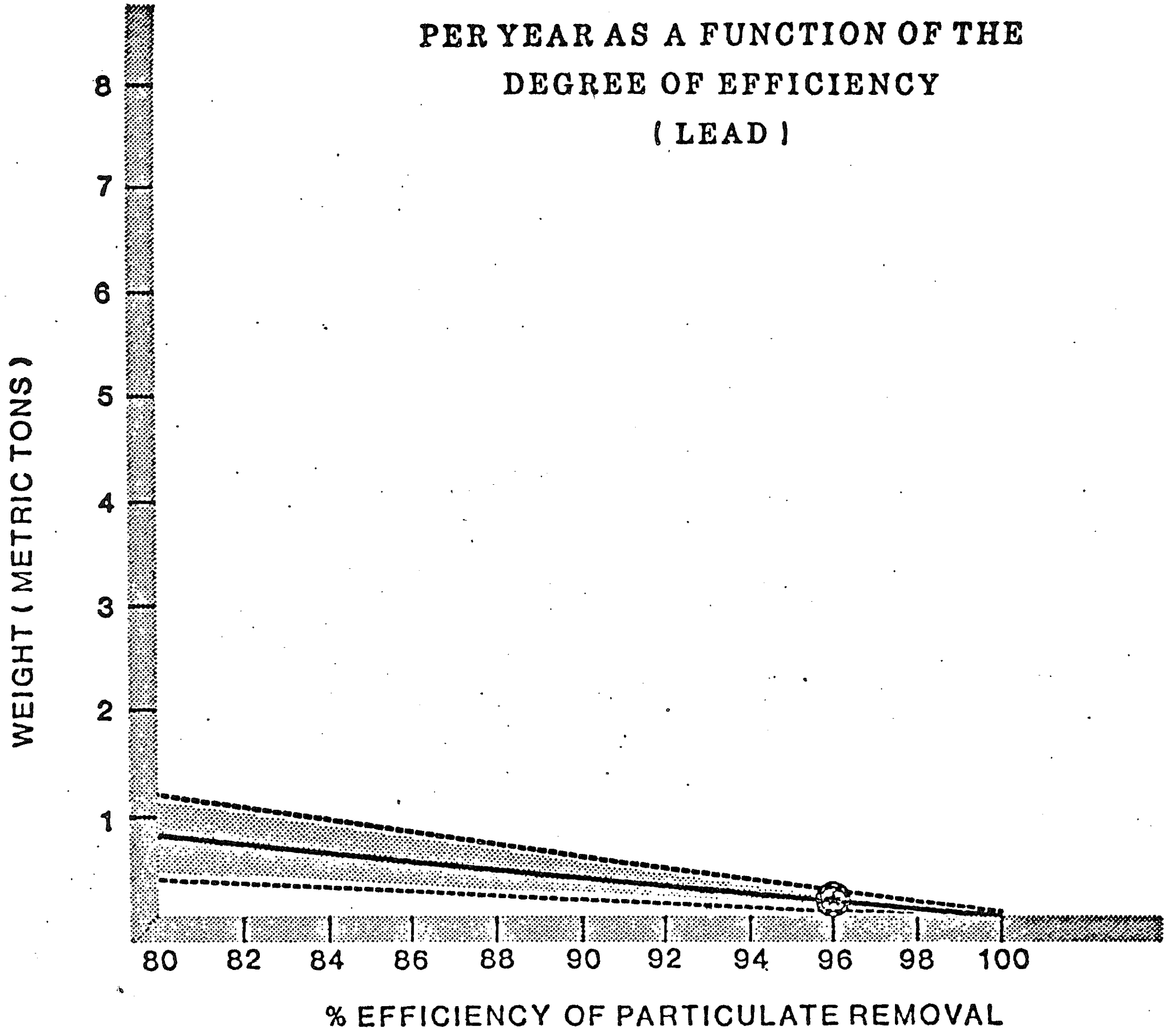
% EFFICIENCY OF PARTICULATE REMOVAL

BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 129.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY
(LEAD)

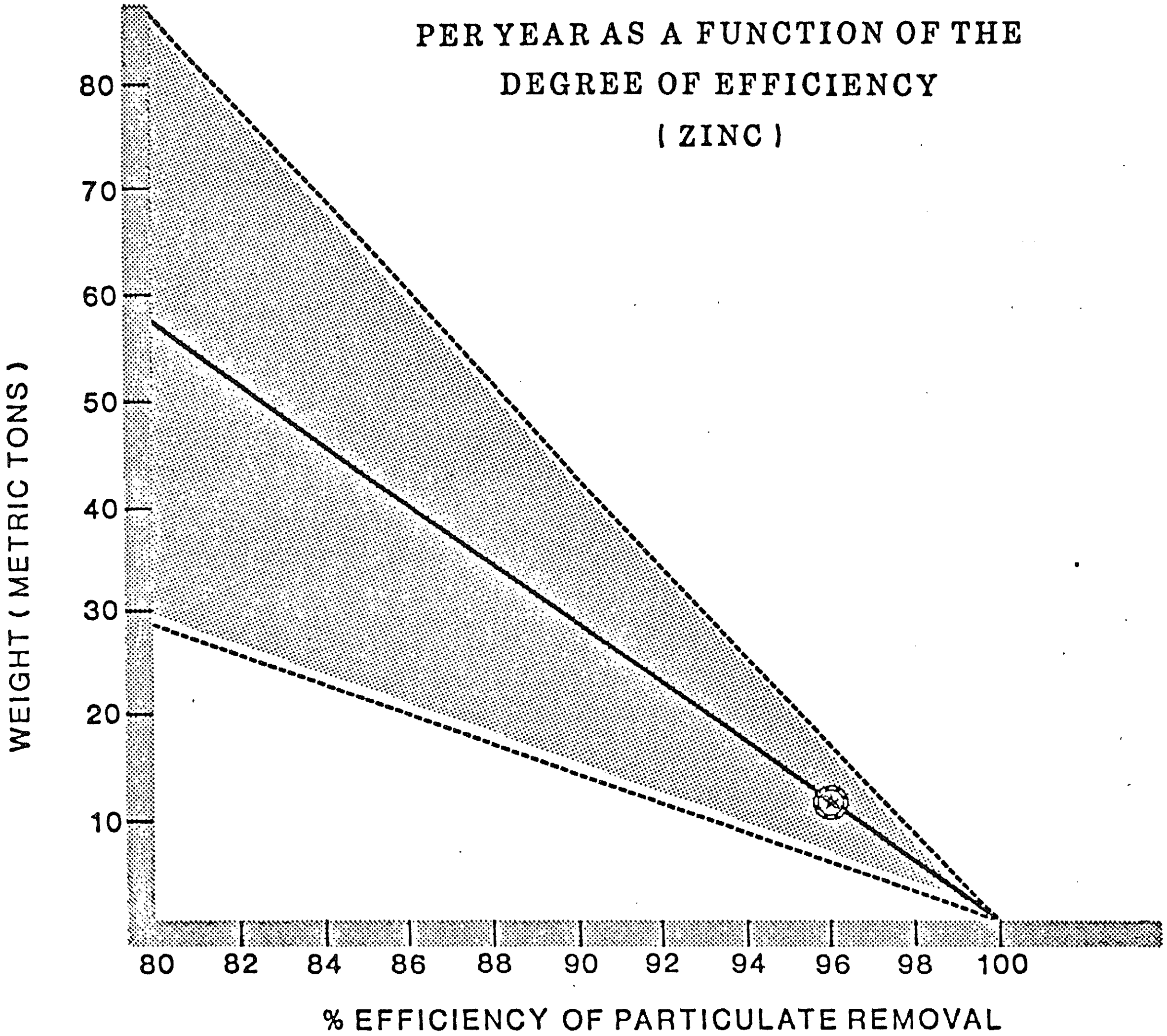


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 130.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY (ZINC)

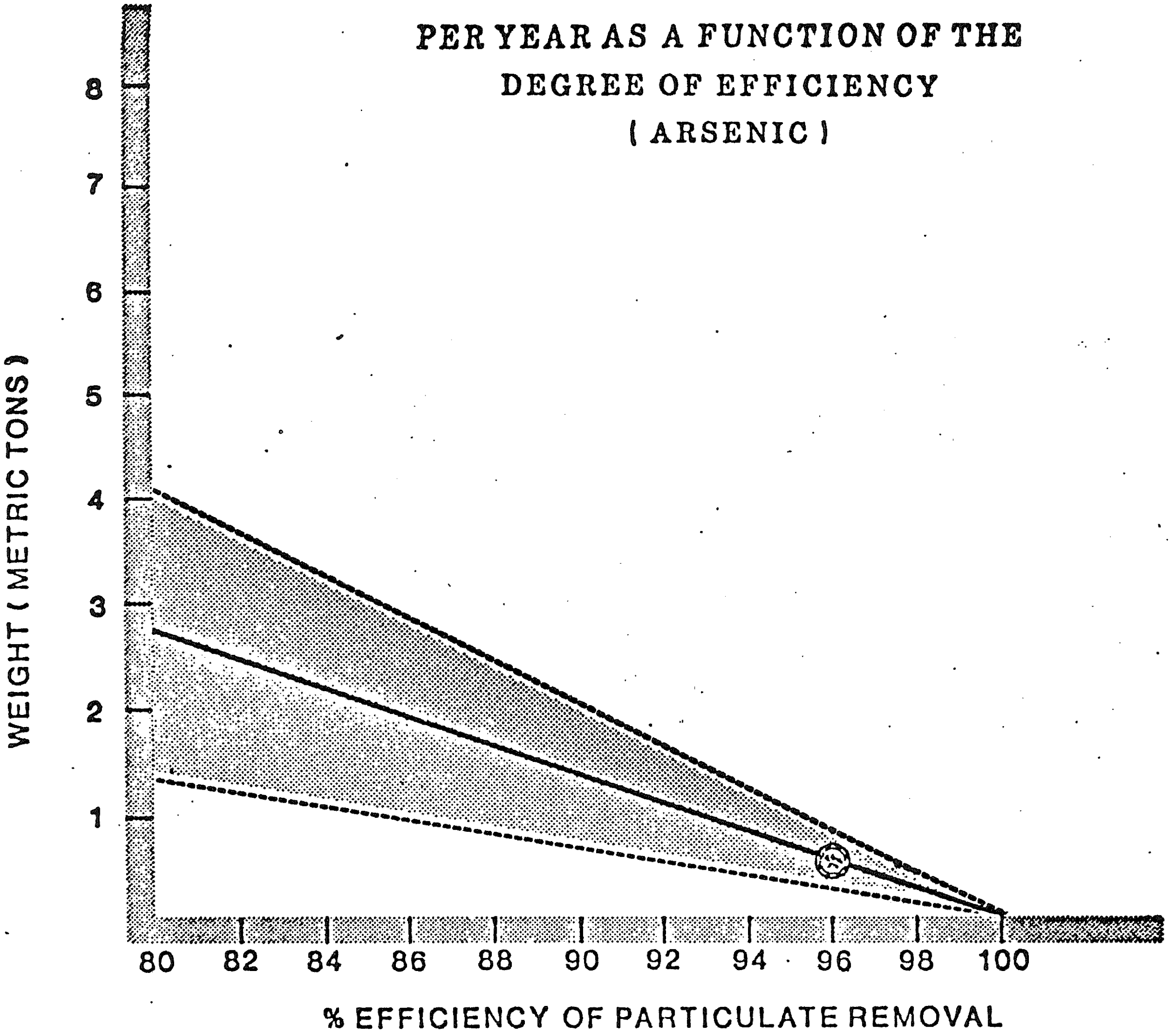


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 131.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY (ARSENIC)

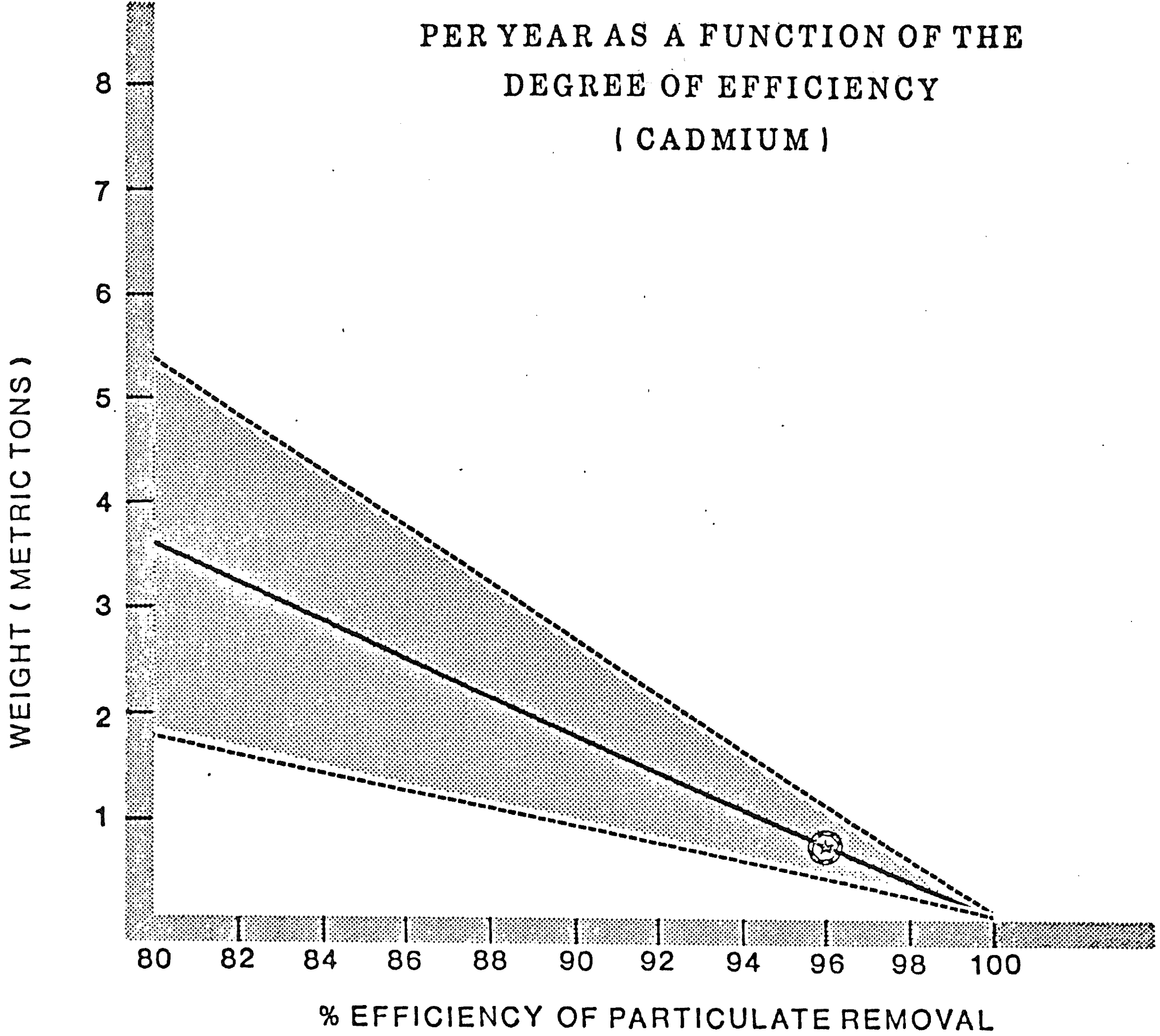


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE
AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 132.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY (CADMIUM)



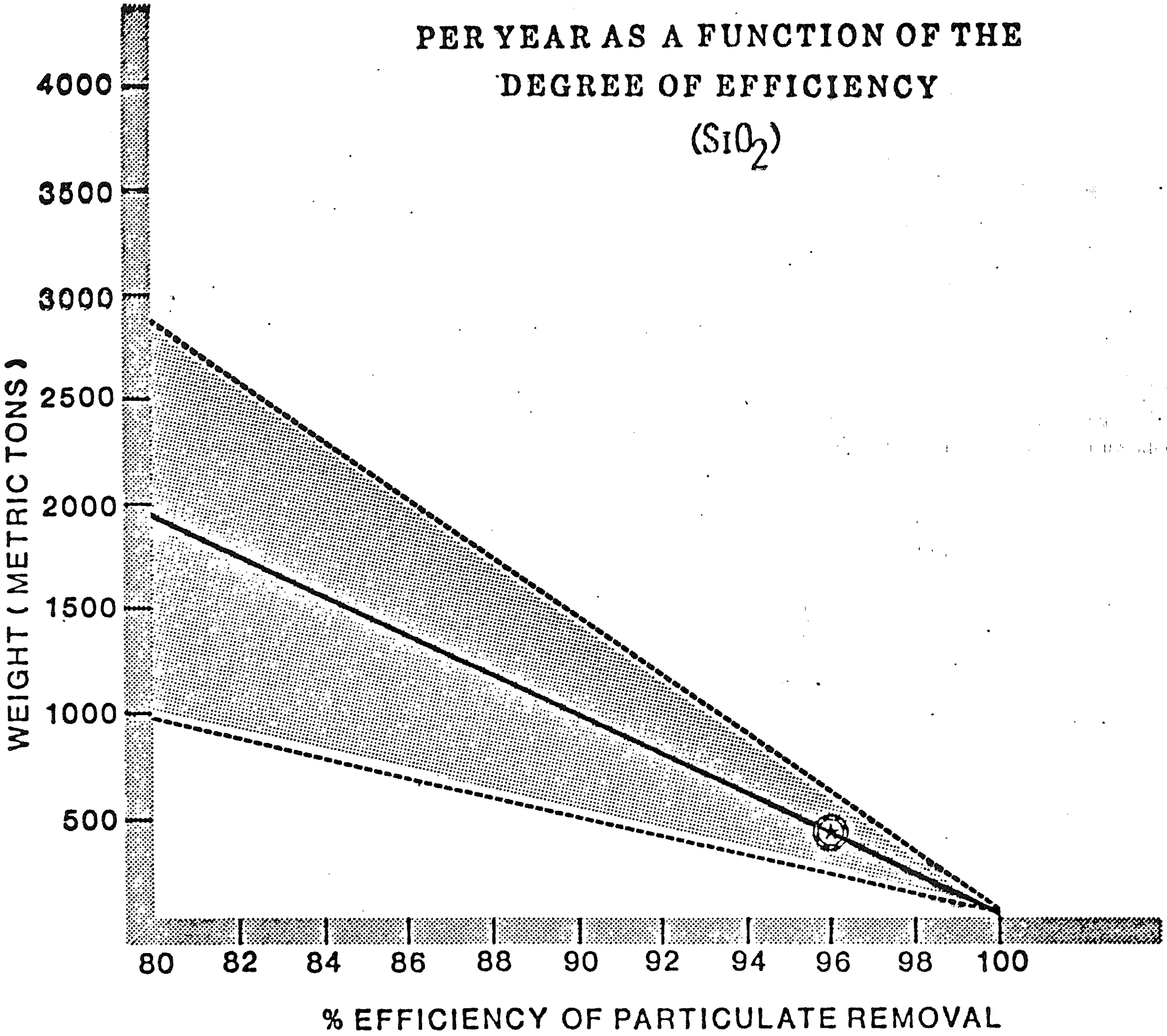
BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE
AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 133.

STACK PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY

(SiO_2)

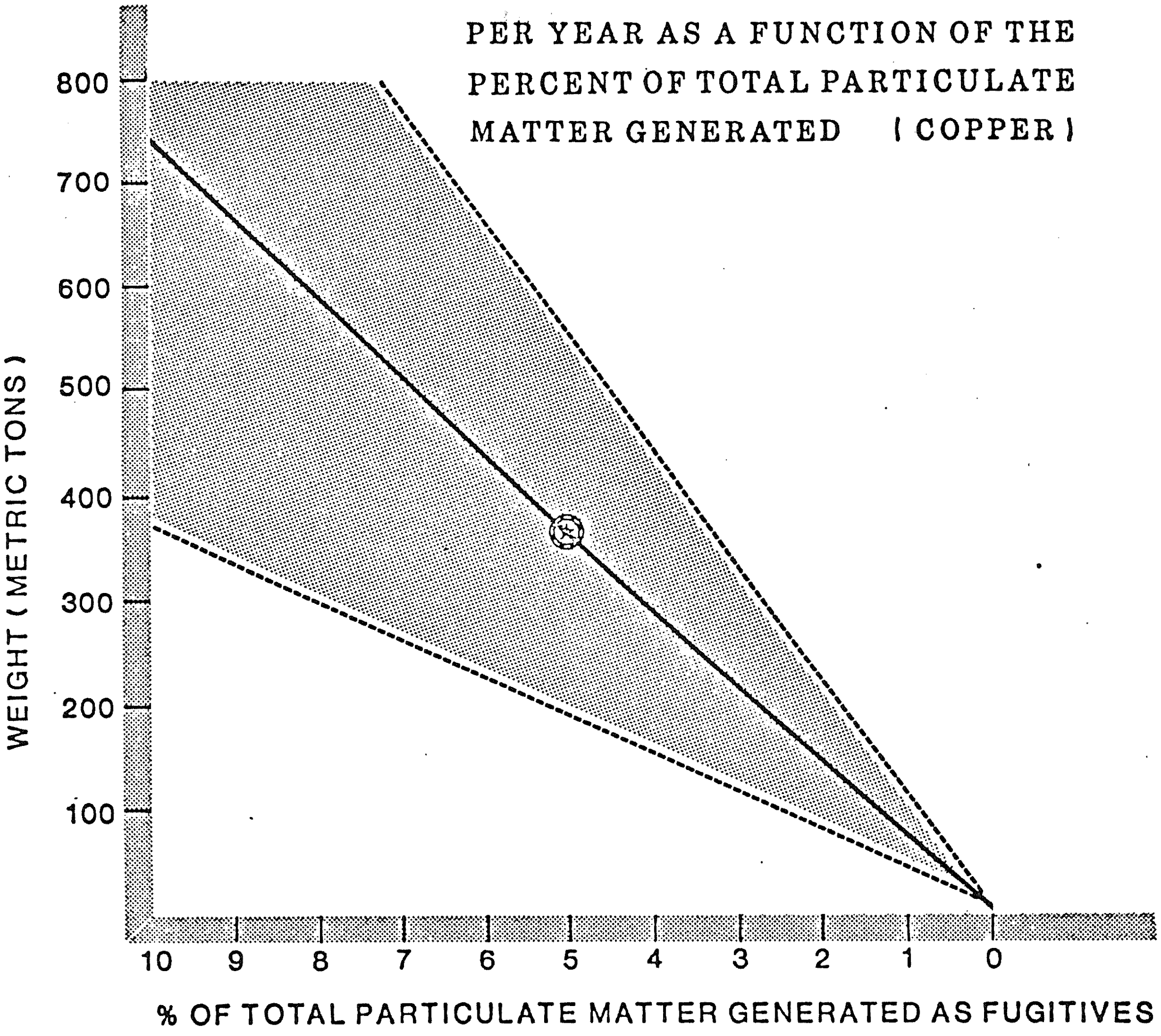


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 134.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (COPPER)

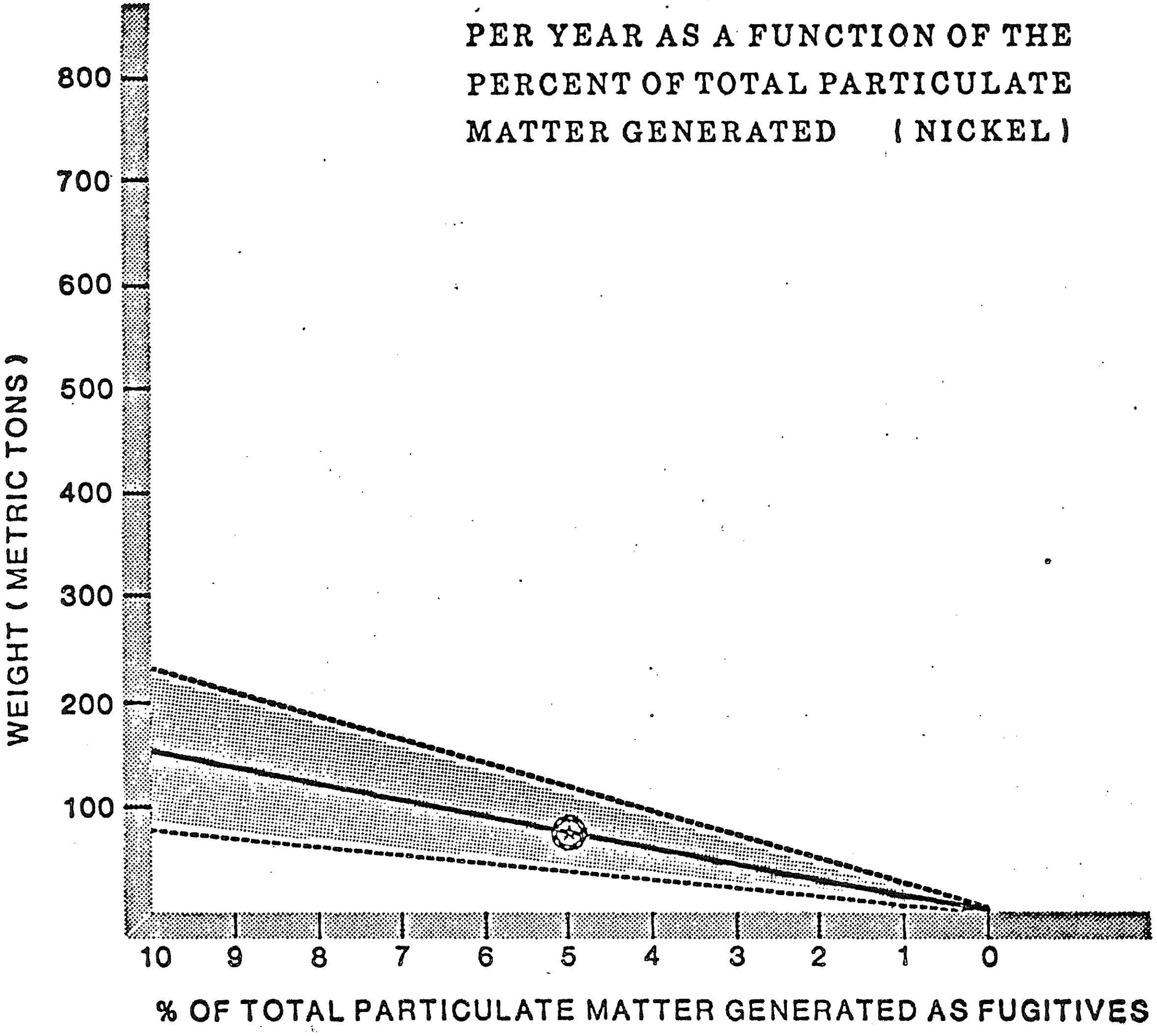


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 135.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (NICKEL)

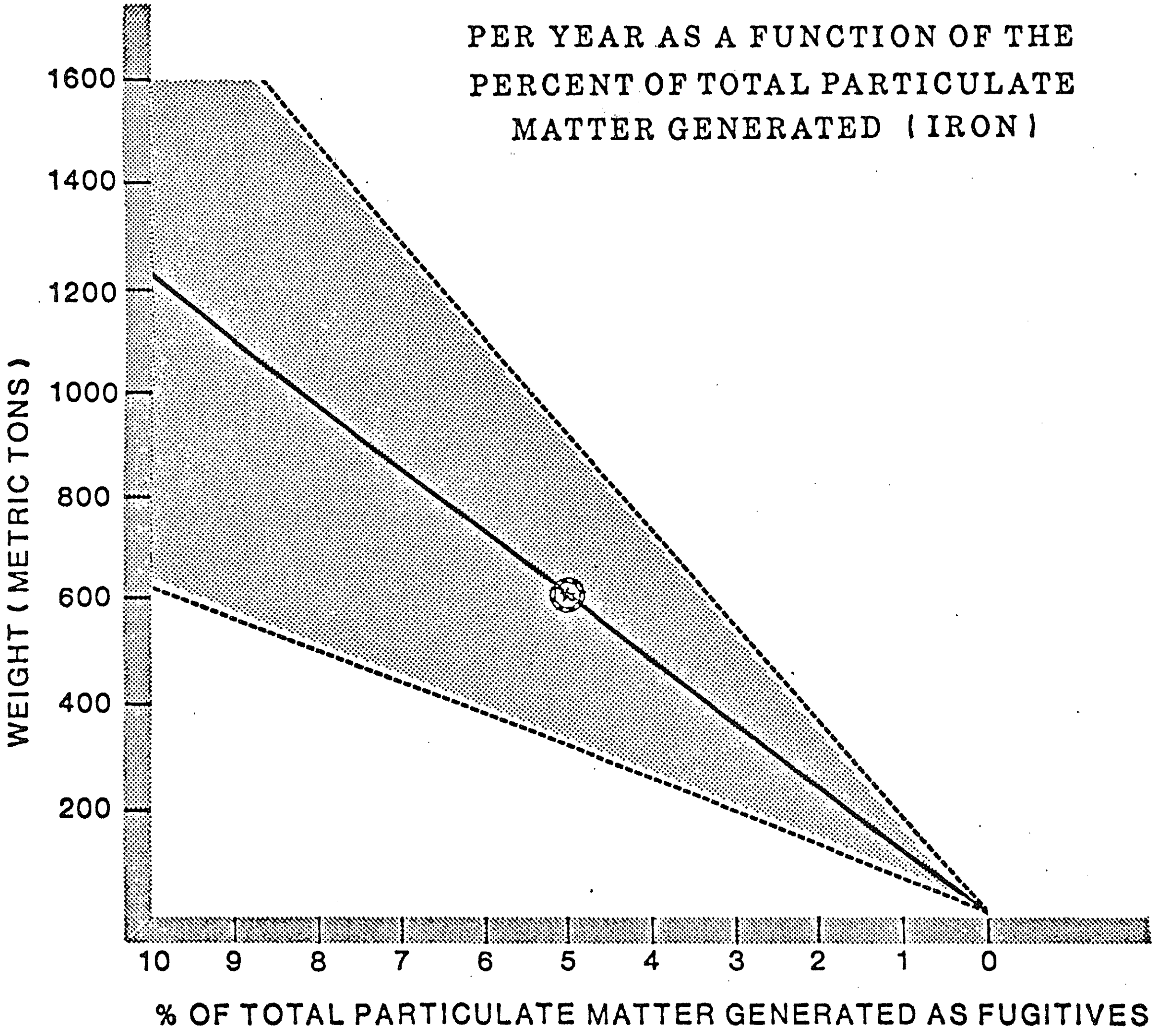


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 136.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (IRON)

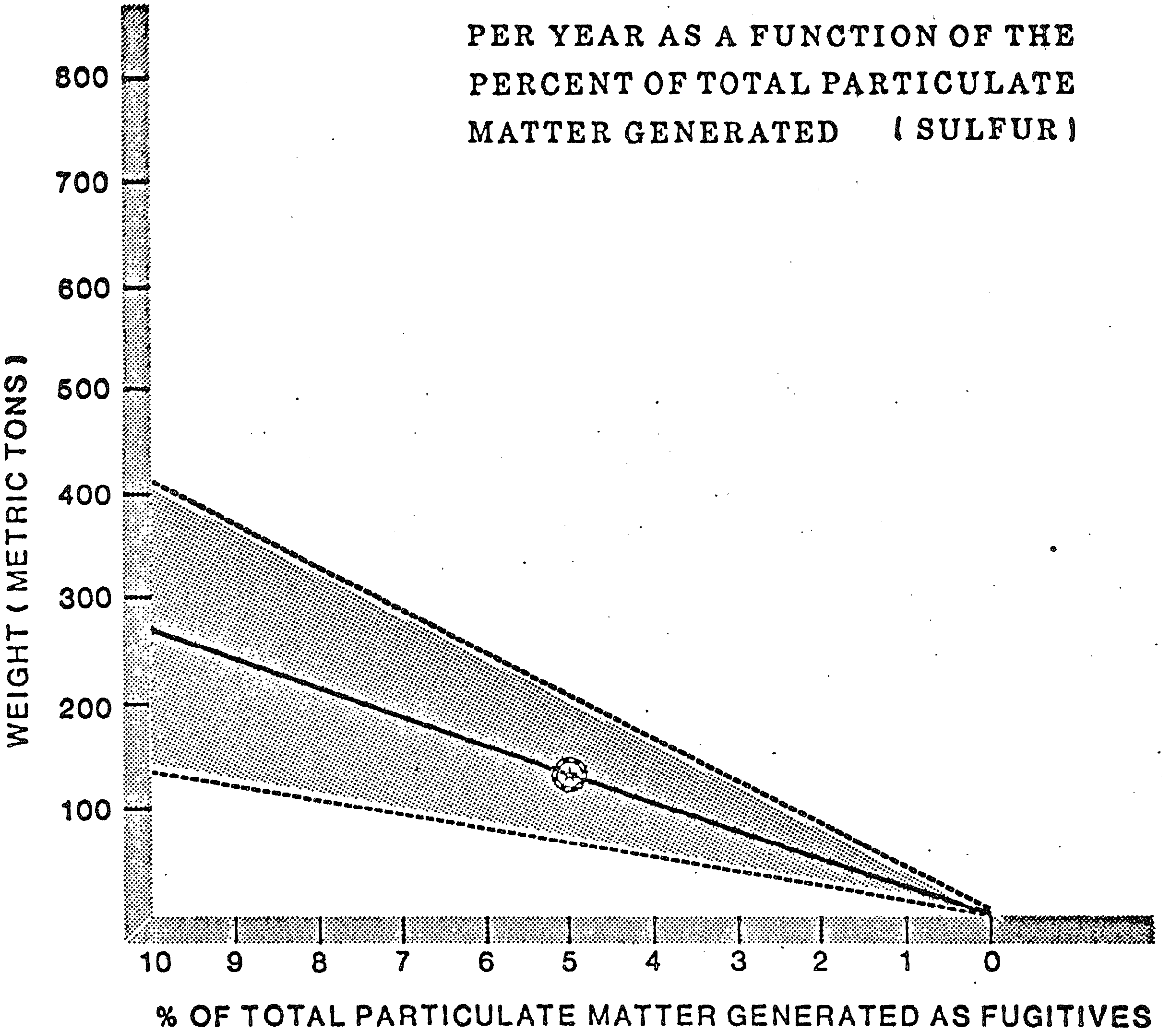


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 137.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (SULFUR)

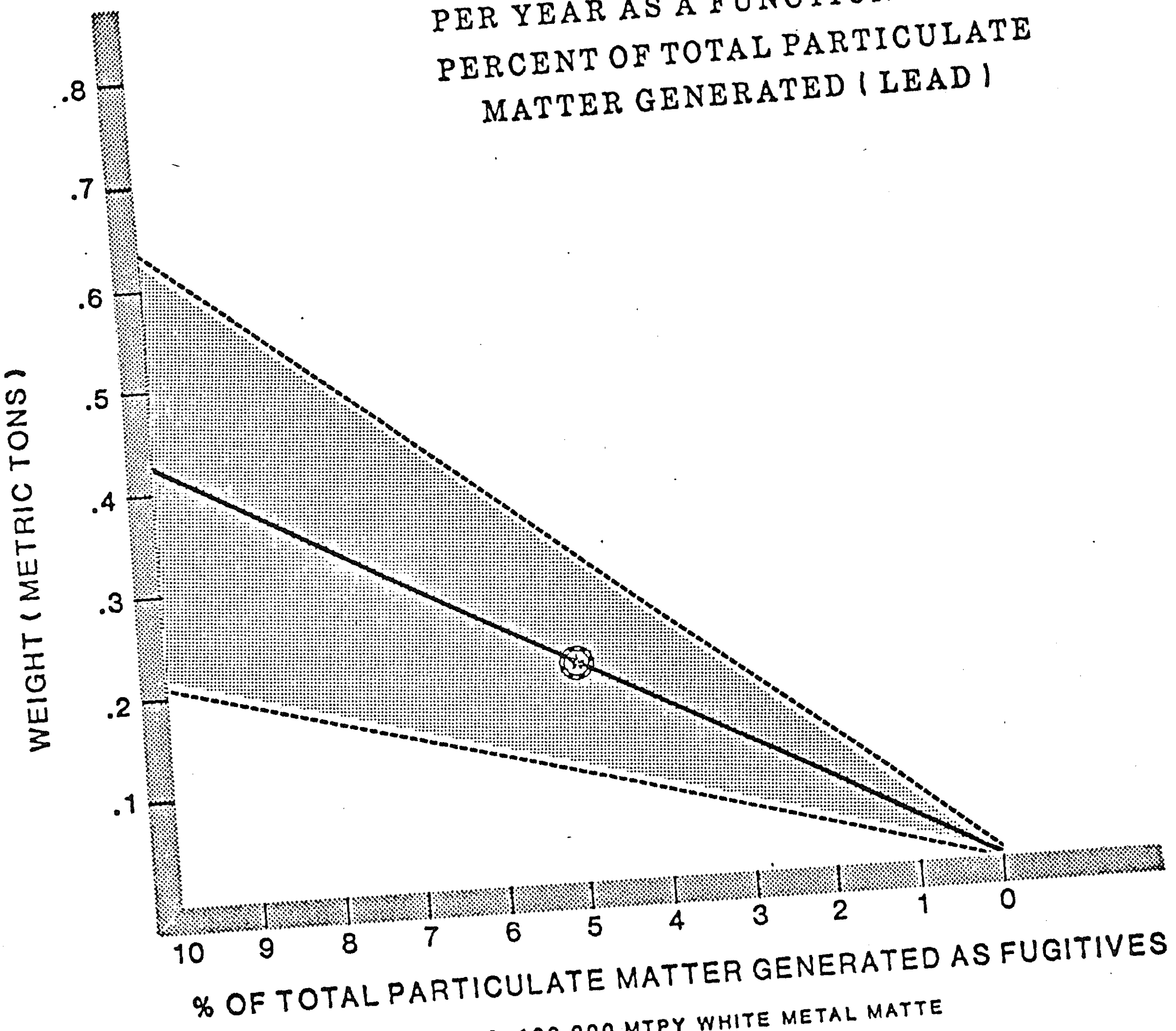


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 138.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (LEAD)

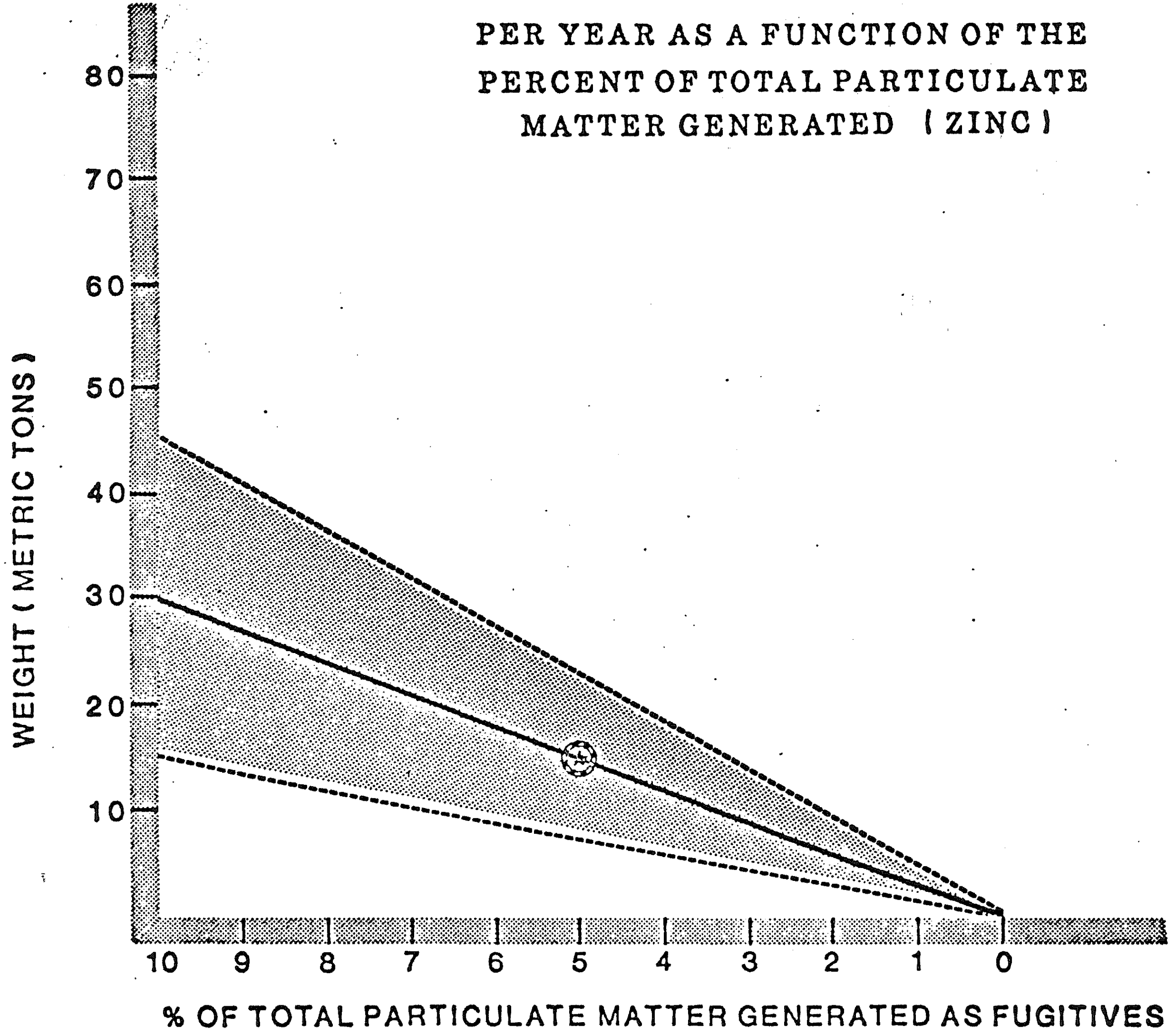


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 139.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (ZINC)

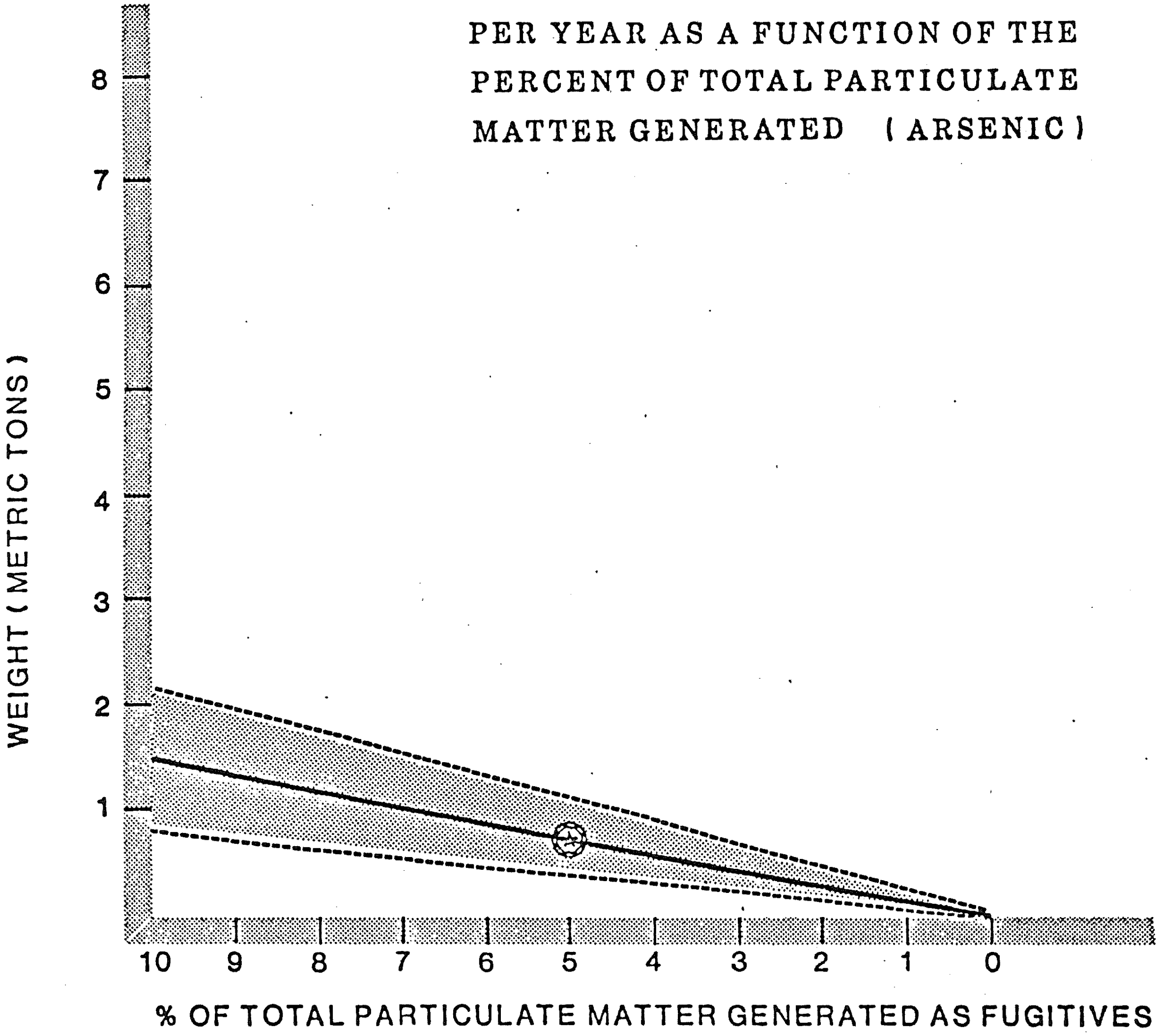


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE
AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 140.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (ARSENIC)

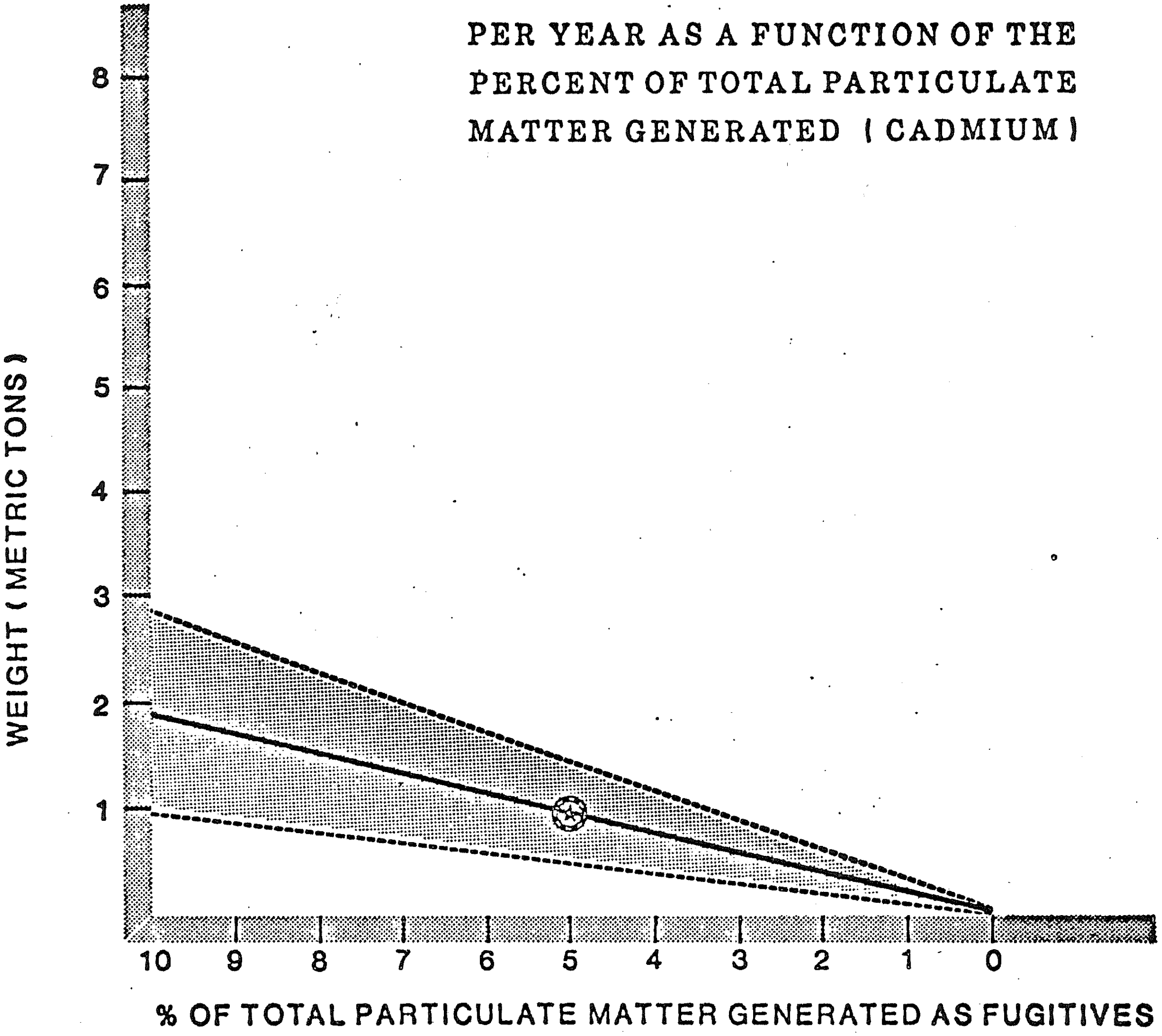


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 141.

FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (CADMIUM)

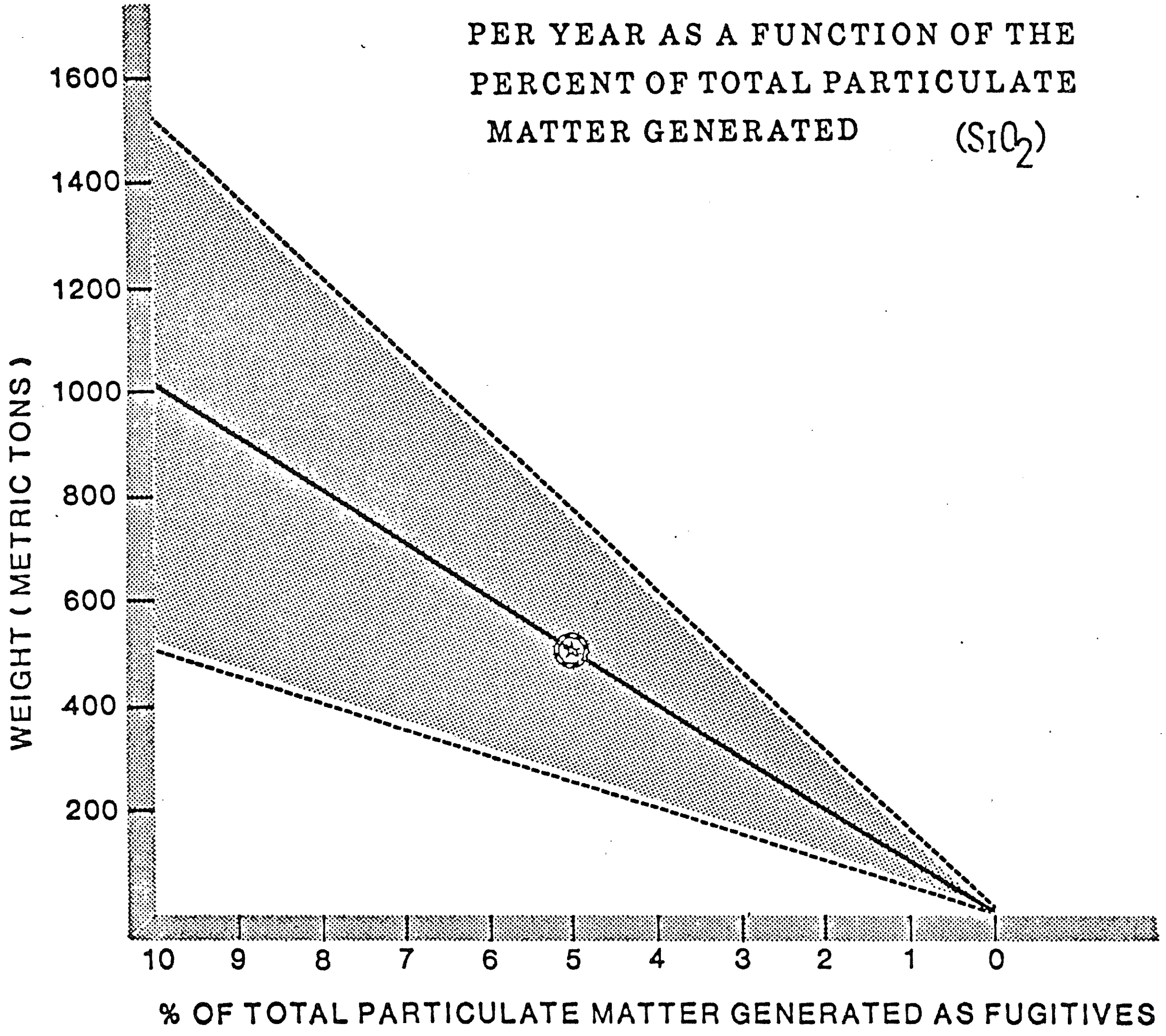


BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 142.

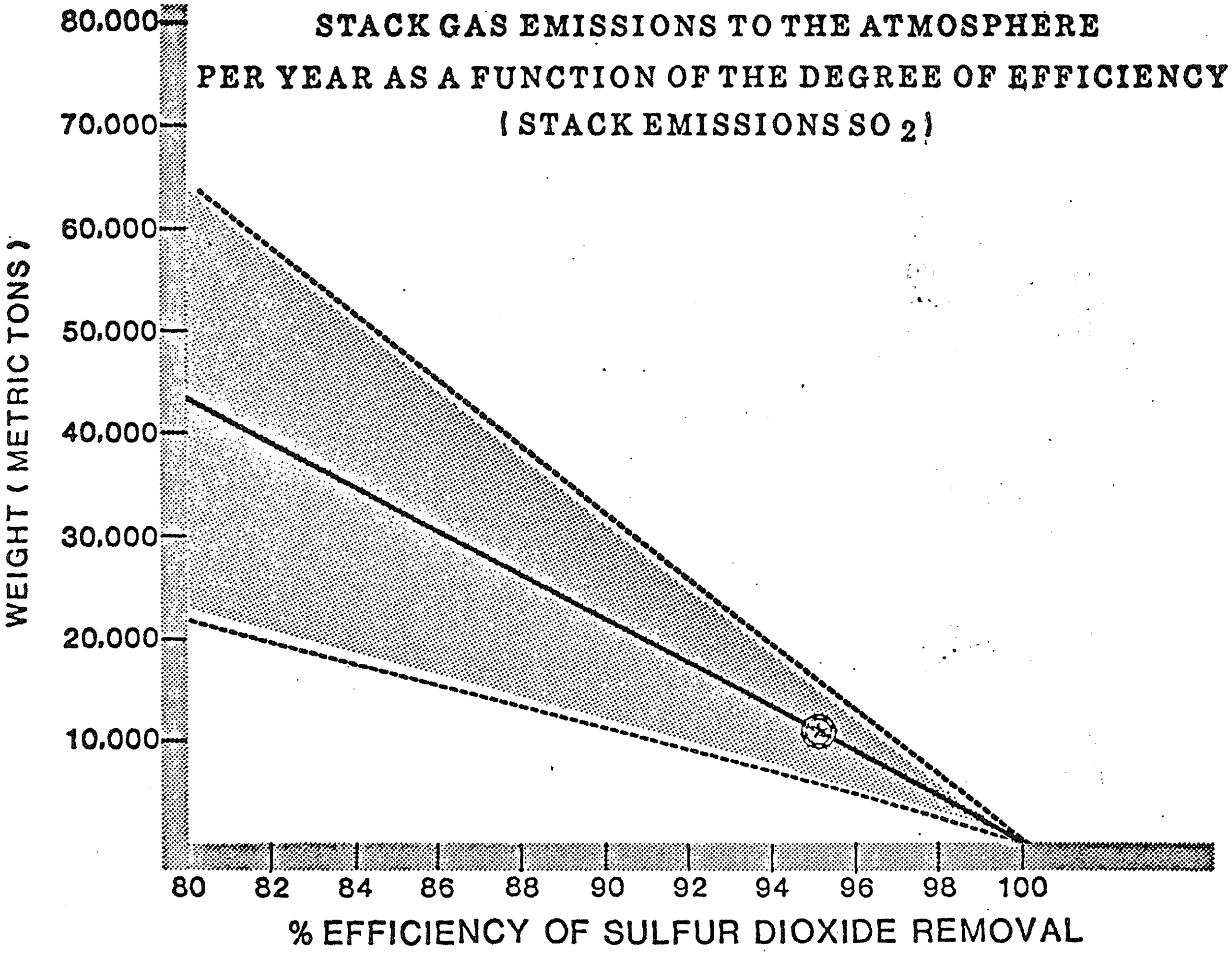
FUGITIVE PARTICULATE EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL PARTICULATE MATTER GENERATED (SiO_2)



BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 143.

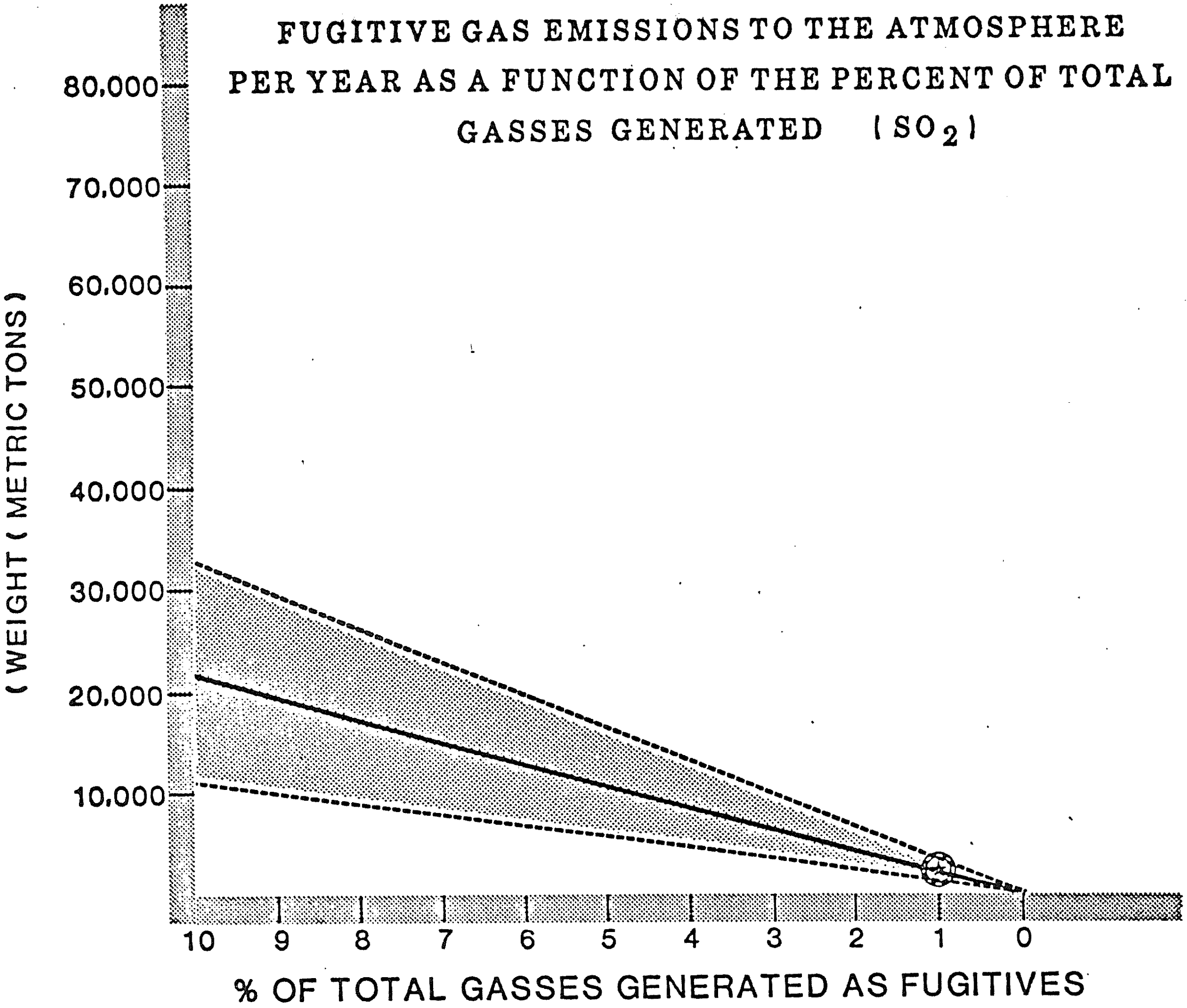


STACK GAS EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE DEGREE OF EFFICIENCY (STACK EMISSIONS SO_2)

BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 144.



FUGITIVE GAS EMISSIONS TO THE ATMOSPHERE PER YEAR AS A FUNCTION OF THE PERCENT OF TOTAL GASSES GENERATED (SO_2)

BASIS: 100,000 MTPY WHITE METAL MATTE

ABSOLUTE QUANTITIES OF A PARTICULAR ELEMENT WILL CHANGE AS THE ASSUMPTIONS TO THE MATERIAL BALANCE CHANGE.

Figure 145. Fugitive Particulate Emissions to the Atmosphere Per Year as a Function of Copper and Nickel Production.

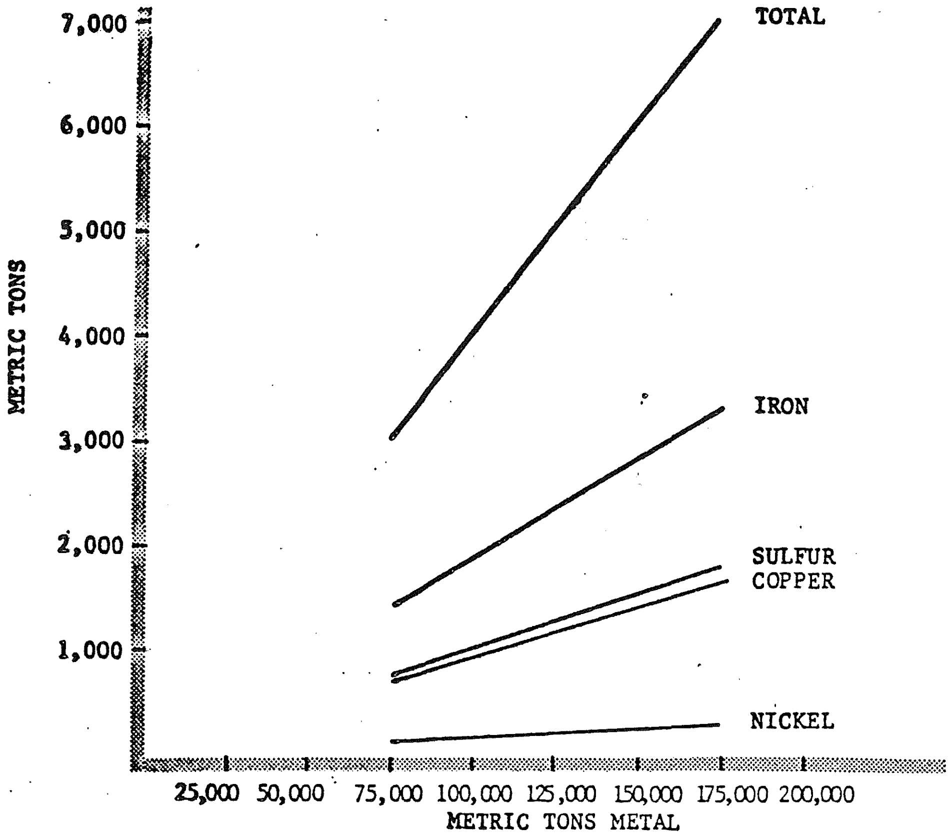


Figure 146. Stack Particulate Emissions to the Atmosphere Per Year as a Function of Copper and Nickel Production.

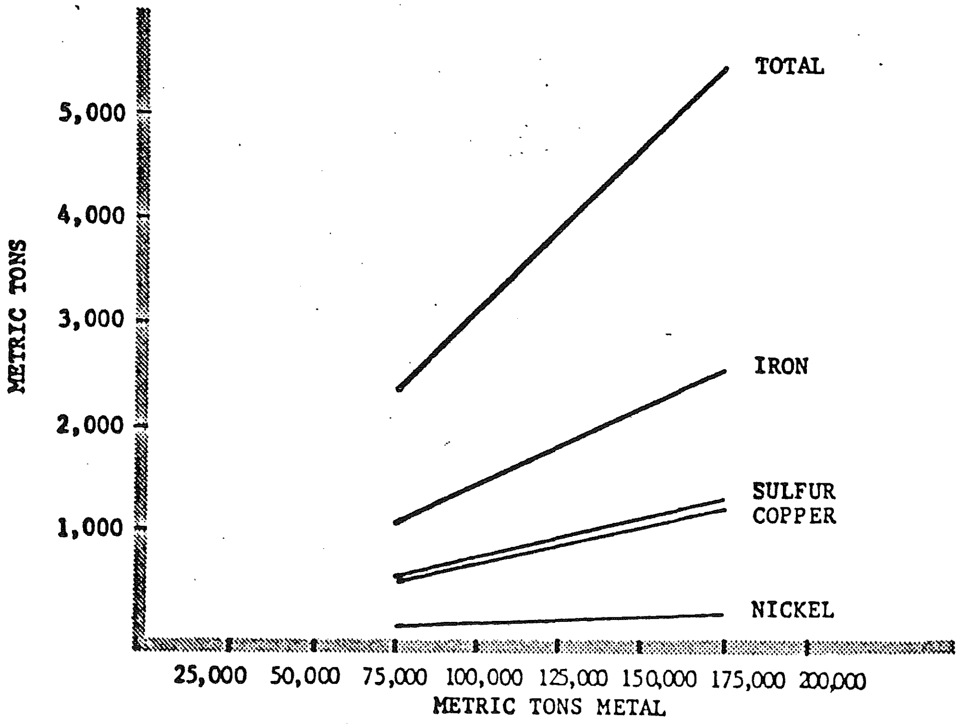


Figure 147. Fugitive and Stack SO_2 Emissions to the Atmosphere Per Year as a Function of Copper and Nickel Production.

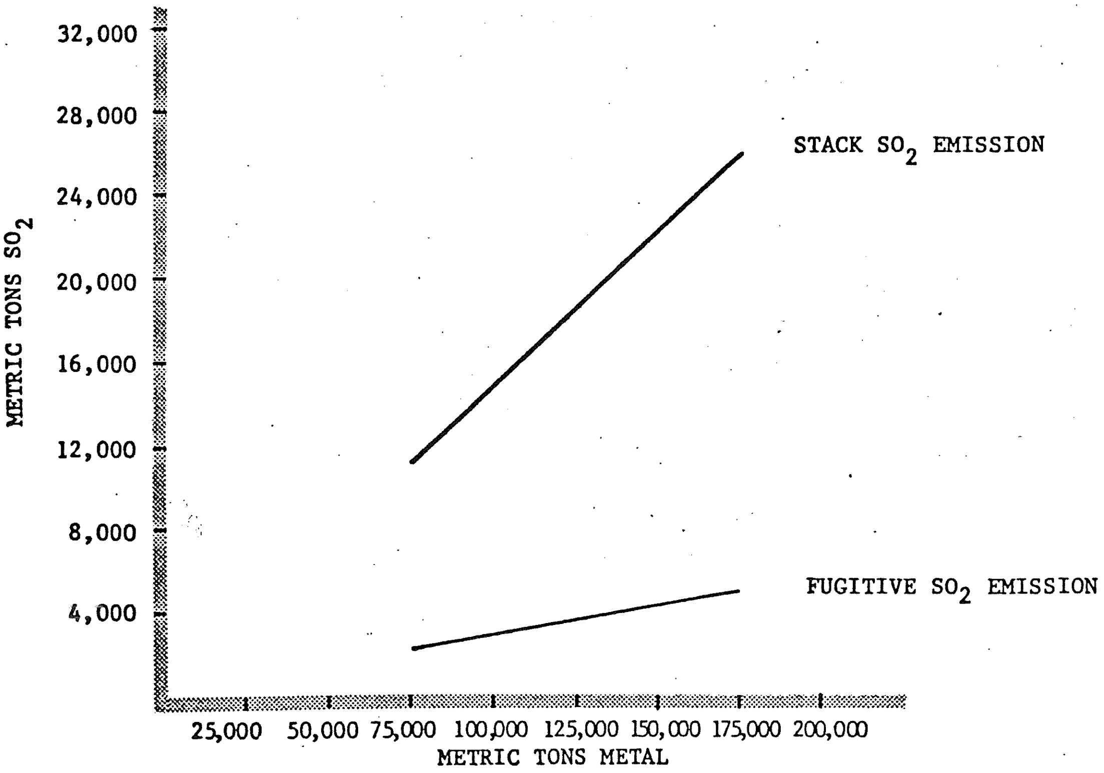
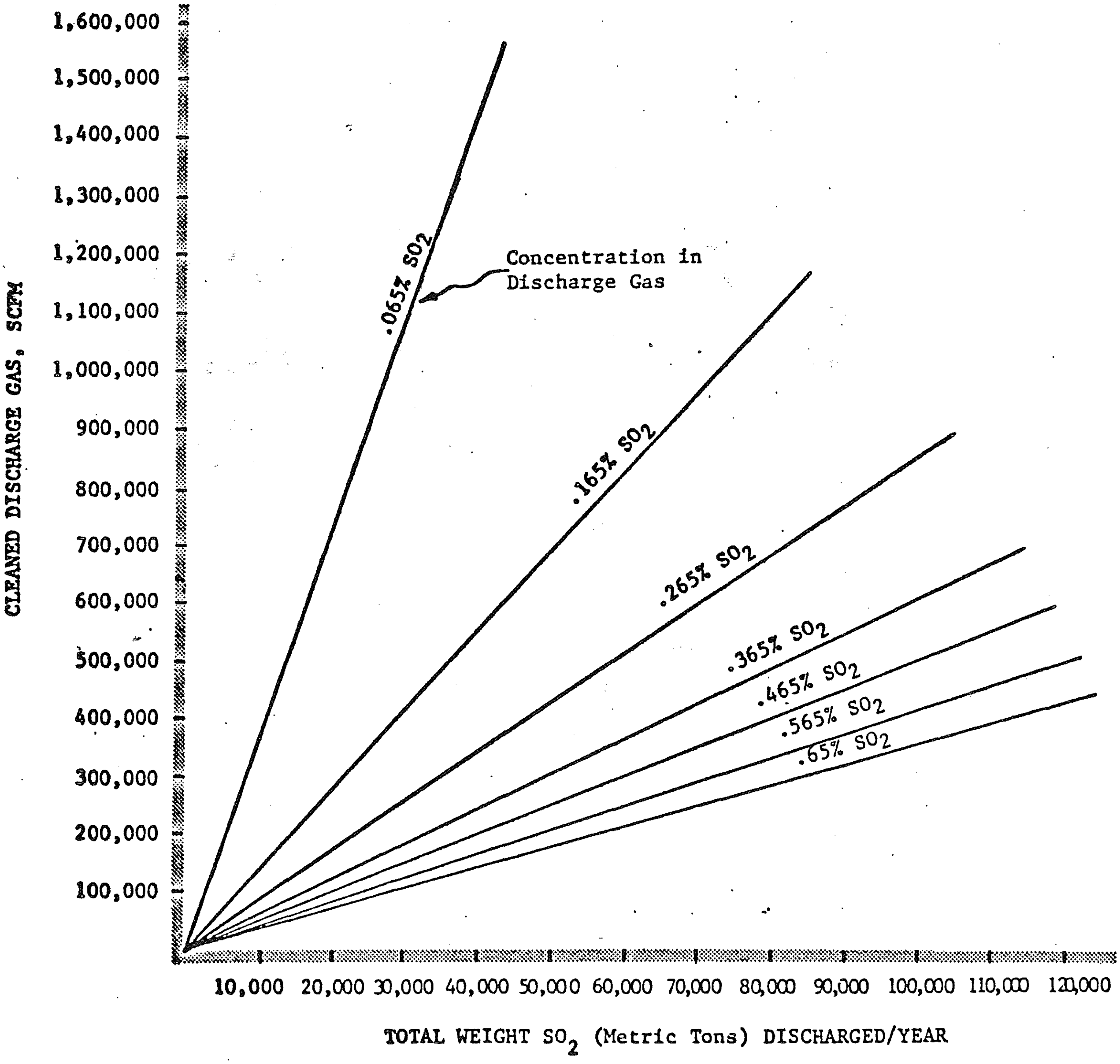


Figure 148. Cleaned Exit Stack Gas Flows as a Function of Quantity and Percentage Concentration of SO_2.



REFINERY MODEL

Copper Refining

Figure 149 illustrates in generalized form the flowsheet for the refining of blister type copper. Table 26 summarizes the material balance for the above flowsheet, and includes the major metals and trace elements. Refer to Table 27 for a list of the specific assumptions made here.

Nickel Refining

Figure 150 illustrates in generalized form the flowsheet for the refining of nickel concentrate. Table 28 summarizes the material balance for the above flowsheet, and includes the major metals and trace elements. Refer to Table 29 for a list of the specific assumptions made here.

SIZE CONSIDERATIONS IN NONFERROUS SMELTING

Smelters, unlike mining and milling operations, are generally large in their throughput requirements. The minimum economic output is that of a single fully-used smelting furnace, generally on the order of 225 to 275 metric tons of copper per day. In this situation, in addition to the smelting furnace itself, two or three converters, an anode furnace, and an anode casting machine would be required.[4]

Figure 151 summarizes the major worldwide copper smelters (and electrowinning plants) by number and capacity. The worldwide average copper smelter size is computed at 87,200 metric tons per year; the USA average is 93,000 metric tons per year. Additional sources define the minimum economic size of smelters as ranging from 55,000 to 94,000 metric tons of copper per year (refer to Figure 151). It is concluded that the average size of a copper-

Figure 149. Generalized Flowsheet for a Copper Refinery.

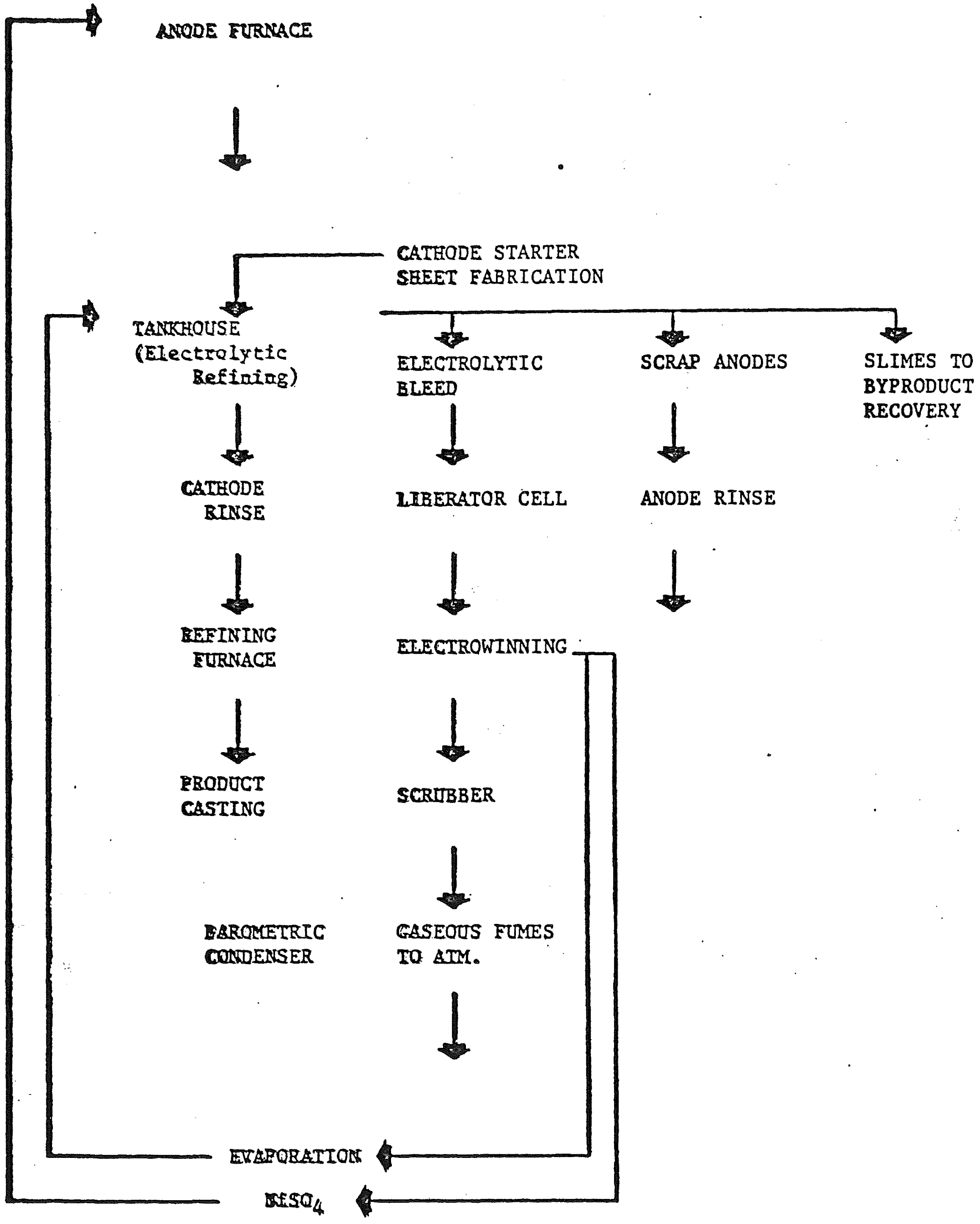


Table 26. Metallurgical Material Balance of the Principal Constituents in Copper Electrolytic Refining (Quantities are in Metric Tons Per Year).

	Cu	Ni	Fe	S	As	Pb	Zn	Cd	Ag
INPUT									
Anode Cu	52,929	212	5	11	.64	13	138	0.80	11.13
OUTPUT									
*Cathode Cu	48,560	29.91	.47	.41	.12	.37	3.89	?	0.26
Scrap Anode	3,142	12.58	.30	.65	.04	.77	8.19	0.05	0.66
*Solution	374	149.00	3.73	—	.26	—	120.56	?	.005
*Slime	52	19.94	.47	9.94	.12	11.86	5.16	?	10.20
Discharge Waters	0.045	0.08	0.02	—	0.1	—	0.1	?	0.005
Cathode Copper (from electrolyte purification)(copper sulfate conversion to metallic copper)	801	.49	.01	—	.002	—	0.6	?	—

BASIS: 100,000 Metric Tons Per Year Production of High Grade Copper-Nickel Matte.

Table 27. Table of "Copper Refining" Assumptions.

1) Average values for waste effluents from the copper refinery are taken from the table below (50,000 metric tons cathode copper/year production is assumed).

Waste Effluents From Electrolytic Copper Plants.

PARAMETER	PLANT NUMBER 1 NET LOADING kg/kkg	 2 NET LOADING kg/kkg	 3 NET LOADING kg/kkg	 4 NET LOADING kg/kkg	 5 NET LOADING kg/kkg	 6 NET LOADING kg/kkg	AVERAGE VALUES kg/kkg	 metric tons/year
pH								
Alkalinity		0.044	0.04		0.017			
COD	0.006	0.472	0.024	0.119	0.305			
Total Solids	0.074	3.40	0.014	0.310				
Dissolved Solids	0.068	1.63	0.006	0.173	0.94			
Suspended Solids	0.006	1.00	0.008	0.137	0.10			
Oil and Grease					0.05	0.003		
Sulfate (as S)	0.011	0.066	0.002	0.080		16.1	0.03975	2.0
Chloride		1.14	0.003	0.037	0.22			
Cyanide								
Aluminum		0.0004			0.010	0.004		
Arsenic		0.0001		0.003		0.174	0.002	0.1
Cadmium		<0.0001						
Calcium	0.040	0.28	0.001		0.06			
Chromium					<0.0001	0.0002		
Copper	0.001	0.0005	0.0001		0.002	0.43	0.0009	0.045
Iron	0.0008	0.0003	<0.0001	0.027	0.0006		0.00045	0.0225
Lead	0.0002		<0.0001		<0.0001	0.005	0.00135	0.0675
Magnesium		0.115						
Mercury					0.0001	<0.0001		
Molybdenum		4.72						
Nickel	0.0004	0.0001	0.0001	0.006	0.001		0.00152	0.076
Potassium		0.013						
Selenium	0.0002	<0.0001	0.0002	0.0004		0.042		
Silver		<0.0001				0.0001	0.0001	0.005
Sodium		0.91	0.0001					
Tellurium			0.001			0.052		
Zinc	0.0002		<0.0001	0.003	<0.0001	0.008	0.00228	0.114

SOURCE: Development document for interim final effluent limitations guidelines and proposed new source performance standards for the primary copper smelting subcategory and the primary refining subcategory of the copper segment of the nonferrous metals manufacturing point source category, United States Environmental Protection Agency, February, 1975.

Not considered in computation of average values.

2) The volume of copper sulfate production converted to metallic copper amounts to 1.5-1.8% of the amount of copper produced. (1.6% of the amount of copper produced is assumed for the copper refinery balance.) Source: The Electrolytic Refining of Copper, V.T. Isakov, 1973.

3) The approximate distribution of the principal elements contained in the anodes between the cathodes, electrolyte, and slime is given below in percent.

	Cathodes	Solution	Slime
Gold	1.0-1.5	-	98.5-99.0
Silver	2.0-3.0	-	97.0-98.0
Copper	98.0	1.9	0.07
Selenium and tellurium	1.0-2.0	-	98.0-99.0
Lead	1.0-5.0	-	95.0-99.0
Nickel	15.0	75.0	10.0
Antimony and tin	20.0-30.0	20.0	50.0-60.0
Sulphur	3.0-5.0	-	95.0-97.0
Iron	10.0-20.0	80.0	10.0-20.0
Zinc	3.0	93.0	4.0
Aluminium	5.0	75.0	20.0
Silicon	-	-	100
Bismuth	-	21.8	78.2
Arsenic	20.0	60.0	20.0

SOURCE: The Electrolytic Refining of Copper, V.T. Isakov, 1973.

4) Assume 95 percent recovery of precious metals.

Table 27. (contd.)

5) Number of starter cells is determined by the following formula:

$H = Nn_1 Mn_3/(2an_2K + n_1 Mn_3) = 19$

where N = number of cells in plant (328)
n_1 = number of cathodes in production cell (45)
n_2 = number of blanks in cell (28)
K = yield of usable sheets (95%)
n_3 = number of sheets used for the production of one starting sheet (1.08)
a = average period for growth of cathodes in all the production cells (14)
M = machine time of production cells (93%)
H = number of starter cells

SOURCE: The Electrolytic Refining of Copper, V.T. Isakov, 1973.

6) Weight of anode is 0.35 metric tons.

7) Weight of cathode is 0.125 metric tons.

8) Number of days per refinery campaign is 28 days.

9) Number of anodes per cell is 44.

10) Number of cathodes per cell is 45.

11) Number of cathodes per campaign is 90.

12) Number of cells is 331

13) Number of cells per section is 24.

14) Number of sections is 14.

Assumptions 6-14
SOURCES: The Electrolytic Refining of Copper, V.T. Isakov, 1973;
Extractive Metallurgy of Copper, A.K. Biswas, 1976.

Figure 150. Generalized Flowsheet for a Nickel Refinery.

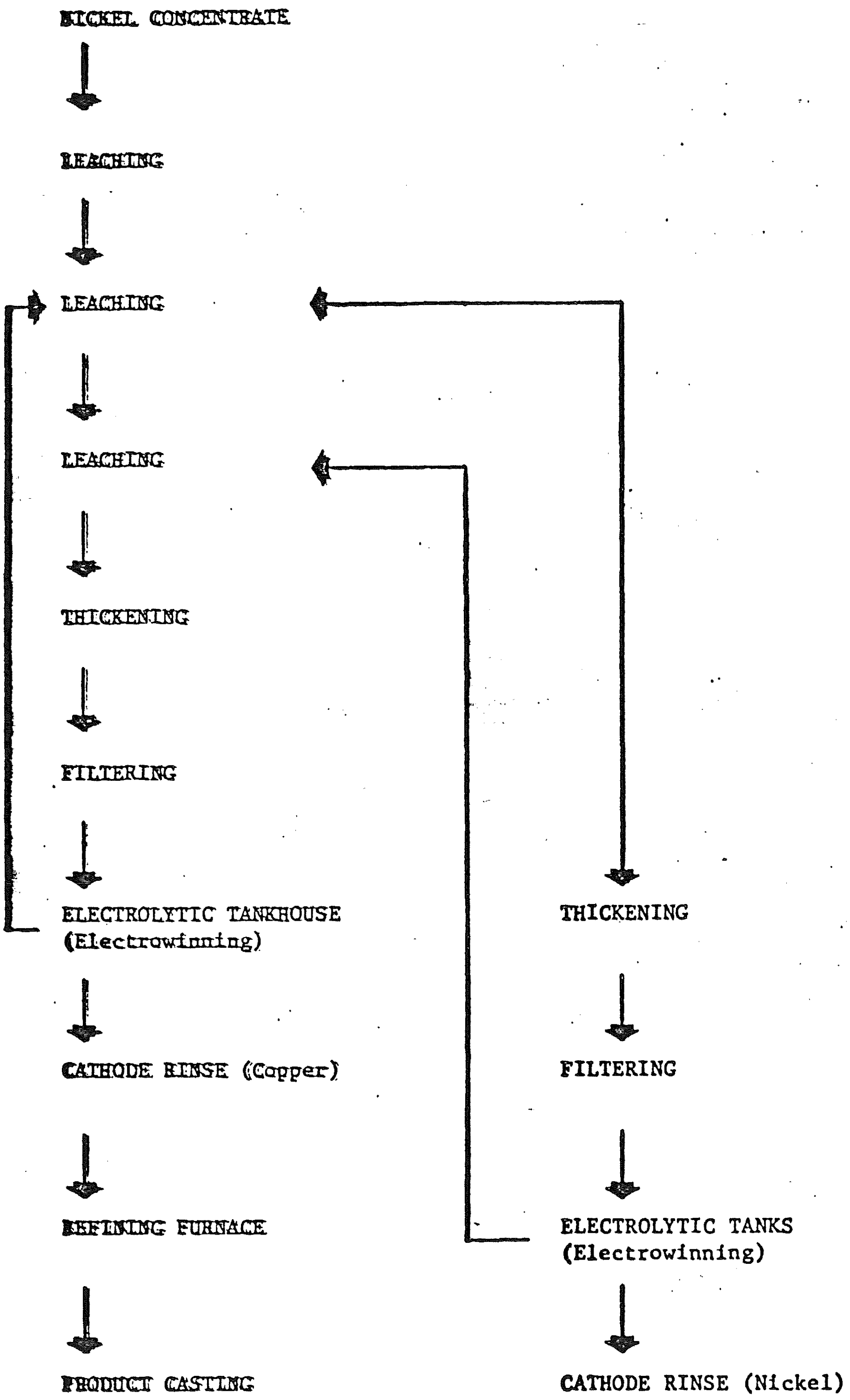


Table 28. Metallurgical Material Balance of the Principal Constituents in Nickel Electrowinning (Quantities are in Metric Tons Per Year).

	Cu	Ni	Fe	S	Pb	Zn	Cd	Ag
INPUT	1,439	7,134	165	3,054	13	227	.8	11.13
OUTPUT								
Cathode Ni	.18	6,064	.06	.06	.04	?	?	?
*Cathode Cu	750	.46	.01	.01	.01	.06	?	.004
*Residue + Anolyte (Returned to system)	688.82	1,069.54	169.93	3,053.93	12.95	226.94	?	11.1126

BASIS: 100,000 Metric Tons Per Year Production of High Grade Copper-Nickel Matte.

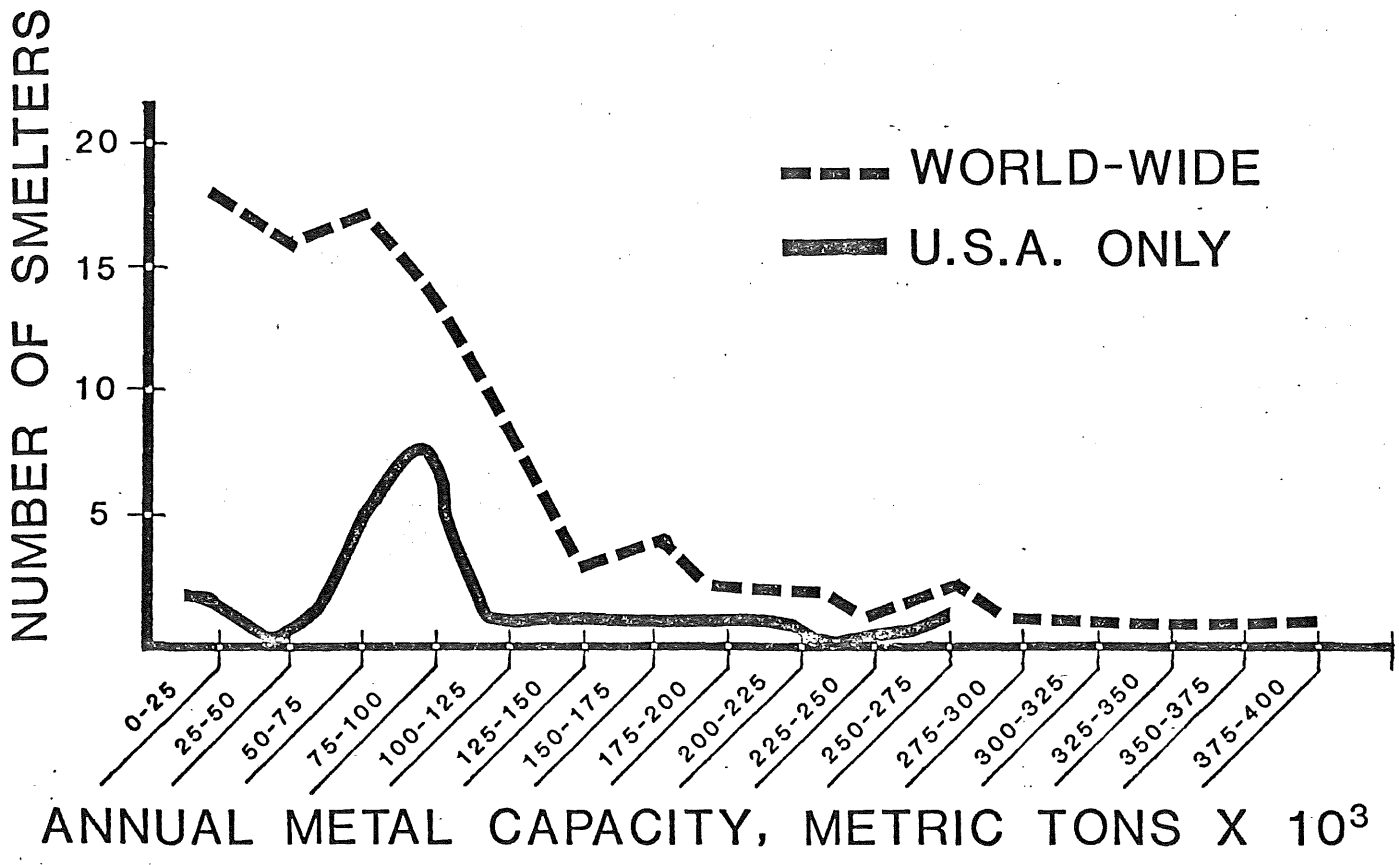
*Speculation.

Table 29. Table of "Nickel Electrowinning" Assumptions.

- Assume an overall nickel recovery if 98 percent.
- Nickel recovery as a nickel cathode is assumed to be 85 percent.
- Weight of nickel cathode is 0.075 metric tons.
- Number of days per refinery campaign is 8 days.
- Number of campaigns per year is 46.
- Number of cells per campaign is 45.
- Number of cathodes per cell is 39.
- Number of anodes per cell is 40.

Figure 151.

MAJOR WORLD-WIDE SMELTERS
(NUMBER AND CAPACITY)[4,12,13,20]



SOURCE	MINIMUM ECONOMIC SIZE IN METRIC TONS
HAYS	90,700 CU-NI/YEAR 55,300 CU/YEAR
MRRC	150,000 CU-NI/YEAR 93,600 CU/YEAR
BISWAS	91,250 CU/YEAR
EPA	90,700 CU/YEAR

WORLD-WIDE AVERAGE

87,200 METRIC TONS PER YEAR

U.S.A. AVERAGE

93,100 METRIC TONS PER YEAR

nickel smelter is 100,000 metric tons of metal (copper and nickel) per year.

SIZE CONSIDERATIONS IN NONFERROUS REFINING

The copper (and nickel) throughput of an electrolytic refinery is usually equivalent to the anode output of a smelter.[4] In cases, however, where electrolytic refineries treat anodes or other materials from several smelters, the throughput of the refinery will necessarily be larger than the throughput of any one of the smelters feeding it.

ENERGY CONSIDERATIONS IN THE NONFERROUS METALS INDUSTRY (Smelting and Refining)

All the vertical stages which are necessary to transform copper and nickel ores into copper and nickel metals use or consume energy. In practice, the energy necessary to make mechanical and chemical transformations is normally two orders of magnitude greater than that which is theoretically calculated as being necessary.[28] Many physical realities account for this phenomenum. Among the more important are:

- ore grades (diminishing with time)(lower than expected)
- incomplete combustion of fuels
- heat losses due to inefficient equipment
- radiation losses
- energy losses in electrical current efficiency and resistance losses
- energy losses in fuel conversion

Table 30 summarizes the efforts made by various individuals and groups to determine the total energy necessary in the mining-refining of copper. Table 31 summarizes similar information pertaining to nickel. The information on mining, milling, and concentrating appears for convenience, and also to demonstrate that the smelting and refining categories, when treated as

Table 30.

TOTAL ENERGY REQUIREMENTS MINING-REFINING COPPER[1,4,6,11,12,18,19,20,24,28]

CATEGORY	SOURCE							
	Battelle (1974) ① (KCAL/MET. Ton Cu)	Battelle (1975) ② (KCAL/MET. Ton Cu)	MRRC (1975) ③ (KCAL/MET. Ton Cu-Ni) (Power Requirements)	Biswas (1976) (KCAL/MET. Ton Cu)	Chapman (1974) ④ (KCAL/MET. Ton Cu)	J.C. Agarwal, J.R. Sinek (1975) (KCAL/MET. Ton Cu)	H.H. Kellogg (1976) (KCAL/MET. Ton Cu)	USBM (1976) (KCAL/MET. Ton Cu)
Mining	1,667,000	6,004,000	800,000	4,000,000	2,905,000	2,994,000	4,657,000*	4,201,000
Milling	3,890,000	8,359,000	4,522,000	1,938,000* (2,769,000)	5,078,000* (7,254,000)	5,292,000* (7,560,000)	7,606,000* (15,523,000)	6,977,000* (9,967,000)
Flotation	1,945,000	3,401,000	1,141,000	831,000*	2,176,000*	2,268,000*	3,260,000*	2,990,000*
SMELTING	4,029,000- 6,529,000	8,207,000- 11,335,000	9,834,000	6,307,000	4,278,000	6,048,000	8,467,000	9,959,000
REFINING	2,223,000	2,815,000	----------	2,154,000	1,155,000	1,991,000	4,234,000	3,341,000

AVERAGE VALUES
(KCAL/MET. TON CU)

MINING	3,404,000
MILLING	5,458,000
FLOTATION	2,252,000
SMELTING	7,499,000
REFINING	2,559,000

* Values adjusted to fit category.

1. Average 0.6% copper ore, reverberatory smelting, no gas effluent treatment.
2. Energy estimates based on data from eight open pit mines, reverberatory smelting, sulfuric acid plant treatment.
3. Electric furnace smelting.
4. Open pit operations.

Table 31. TOTAL ENERGY REQUIREMENTS MINING-REFINING NICKEL[11,12]

CATEGORY	SOURCE
	BATTELLE (1975) (KCAL/MET. TON NI)
Mining	8,743,000
Milling	5,385,000
Flotation	1,129,000
SMELTING	16,748,000
REFINING	7,923,000

Data derived from literature.

individual stages, under certain situations and conditions, consume less energy than the mining and concentrating phases. This is evident in Figures 152, 153, 154, and 155, which illustrate the effects of grade change on the percentage of total energy needed in the various mineral phases (mining, concentrating, smelting, and refining). Figure 155 dramatically illustrates the sensitivity of energy distribution to the ore grade. Incidentally, the curves were generated at the discretion of this author to illustrate a trend or pattern, and may or may not be representative of the real curves. Figures 156, 157, 158, and 159 depict similar (but limited) information for nickel.

Returning to the categories of smelting and refining, the average values indicate that 7.5 X 10^6 KCAL/metric ton copper are required for smelting and 2.6 X 10^6 KCAL/metric ton copper are required for refining. Inherent savings in energy will become apparent later in the discussion when specific smelting furnaces are considered on an individual basis. Table 31 indicates that 16.7 X 10^6 KCAL/metric ton nickel are required for smelting and 7.9 X 10^6 KCAL/metric ton nickel are required for refining (these are also average values).

ENERGY REQUIREMENTS FOR VARIOUS TYPE SMELTING FURNACES
(Smelting and Refining)

New technology has made it possible to include more energy saving furnaces other than the conventional reverberatory furnace in the design criteria of contemporary smelting complexes. Table 32 compares and contrasts the inherent differences in energy consumption for six different type smelting furnaces. Each furnace has been described in earlier reports, so a discussion of each will not be included here.

Figure 152. DISTRIBUTION OF ENERGY CONSUMED IN THE PRODUCTION OF COPPER MINING AND CONCENTRATING[1,2,4,6,11,12,18,22,28]

(AS A PERCENT OF TOTAL ENERGY CONSUMPTION)

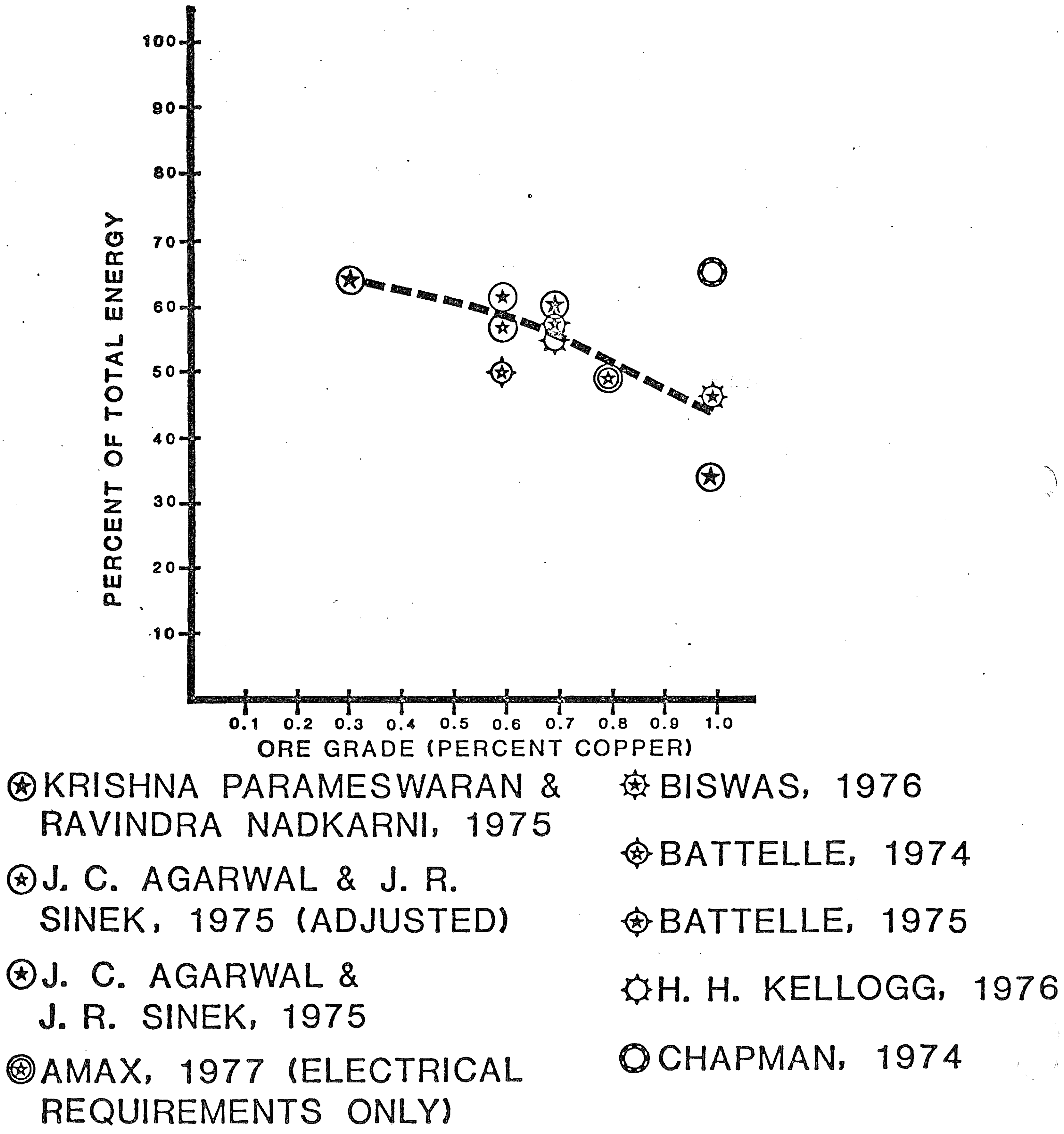


Figure 153. DISTRIBUTION OF ENERGY CONSUMED IN THE PRODUCTION OF COPPER SMELTING[1,2,4,6,11,12,18,22,28]

(AS A PERCENT OF TOTAL ENERGY CONSUMPTION)

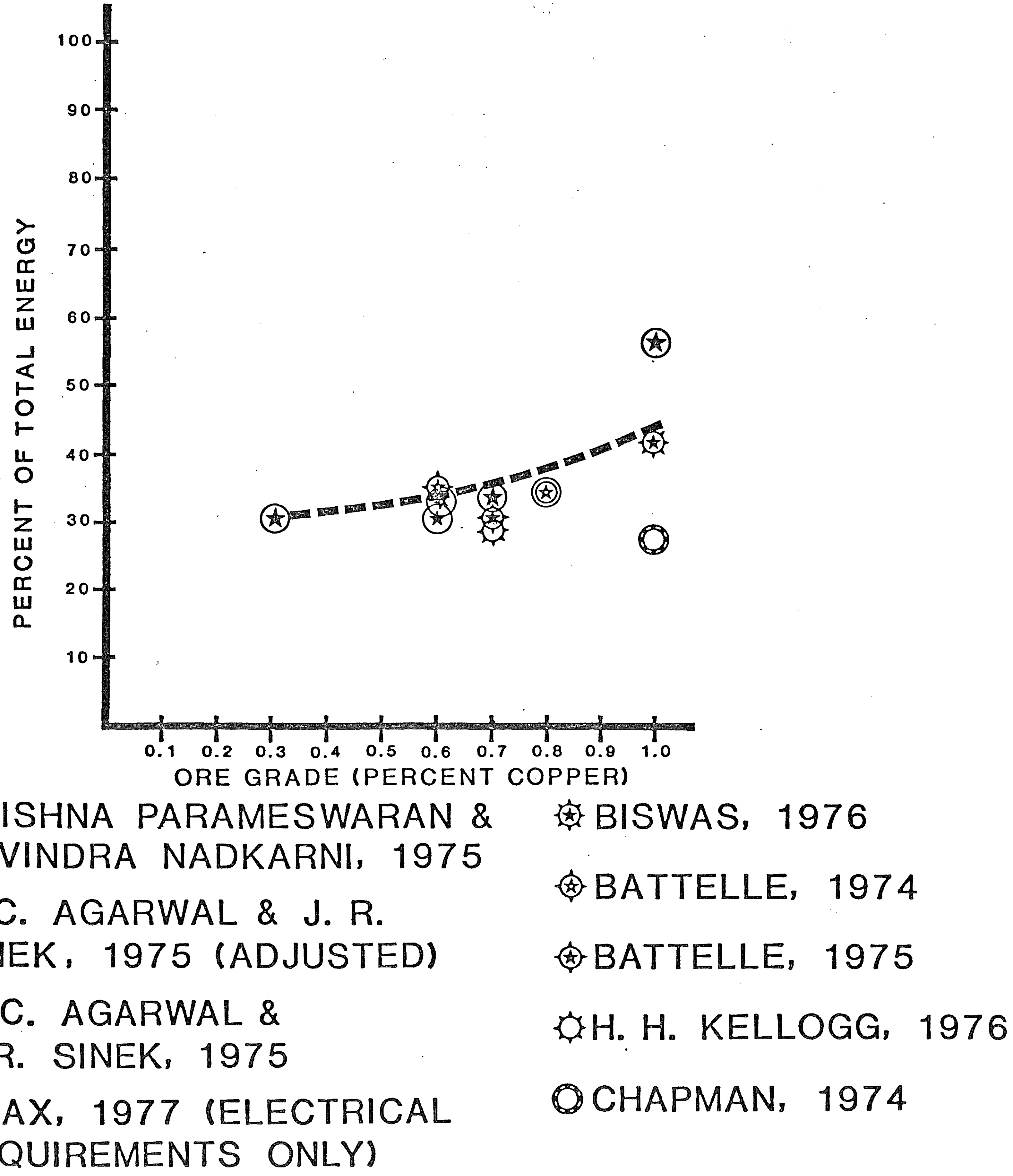


Figure 154. DISTRIBUTION OF ENERGY CONSUMED IN THE PRODUCTION OF COPPER REFINING[1,2,4,6,11,12,18,22,28]

(AS A PERCENT OF TOTAL ENERGY CONSUMPTION)

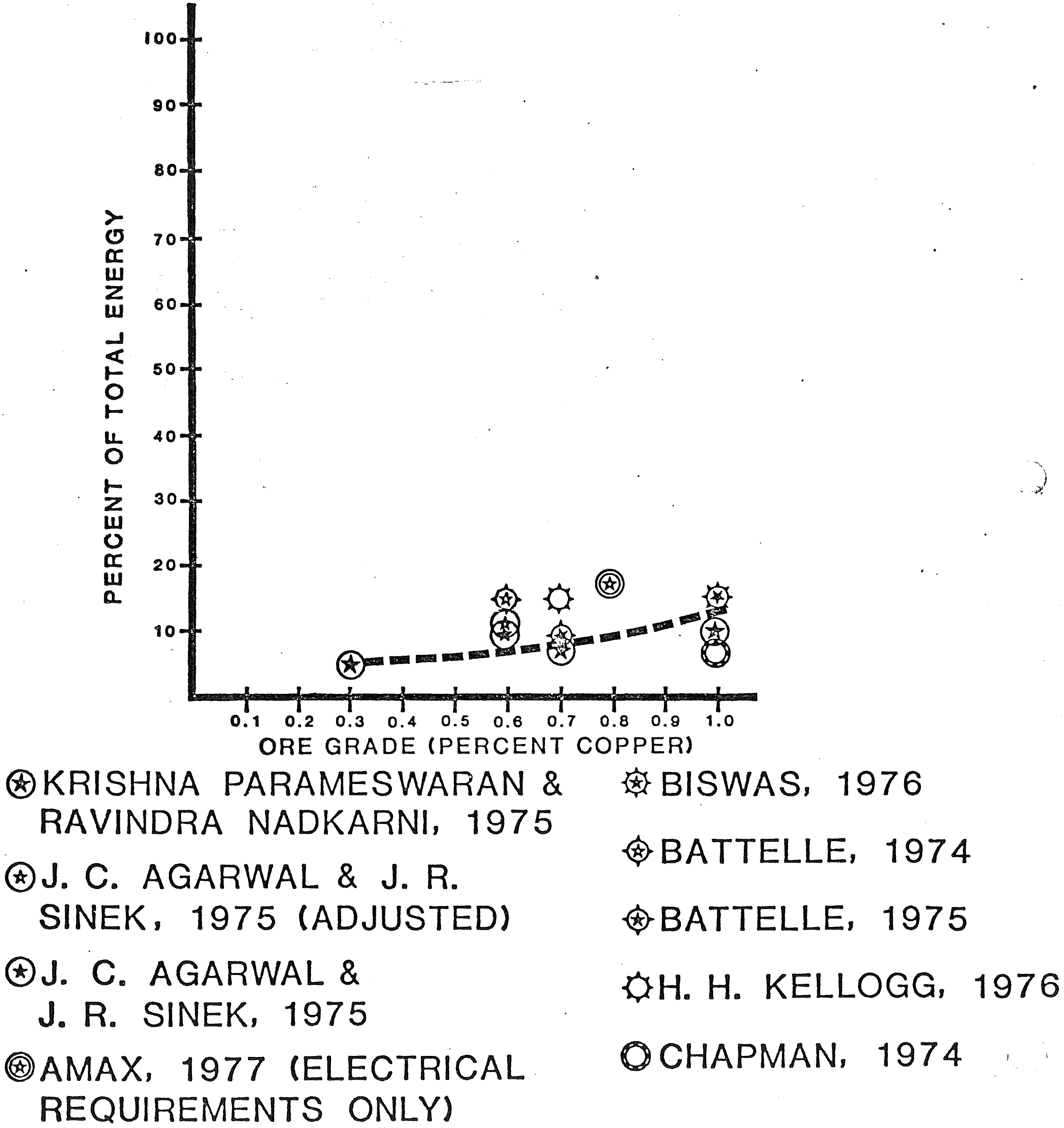


Figure 155. DISTRIBUTION OF ENERGY CONSUMED IN THE PRODUCTION OF COPPER

(AS A PERCENT OF TOTAL ENERGY CONSUMPTION)

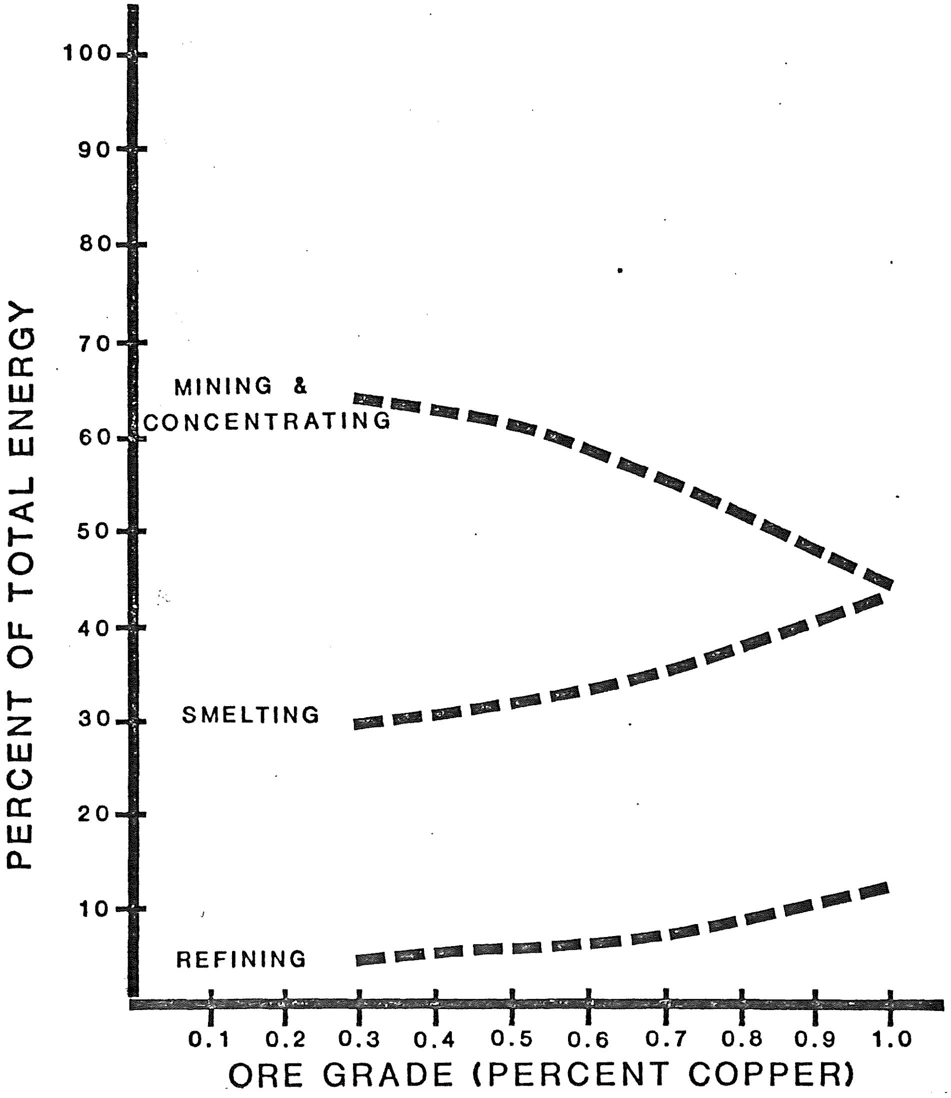


Figure 156. DISTRIBUTION OF ENERGY CONSUMED IN THE PRODUCTION OF NICKEL MINING AND CONCENTRATING[9,11,12]

(AS A PERCENT OF TOTAL ENERGY CONSUMPTION)

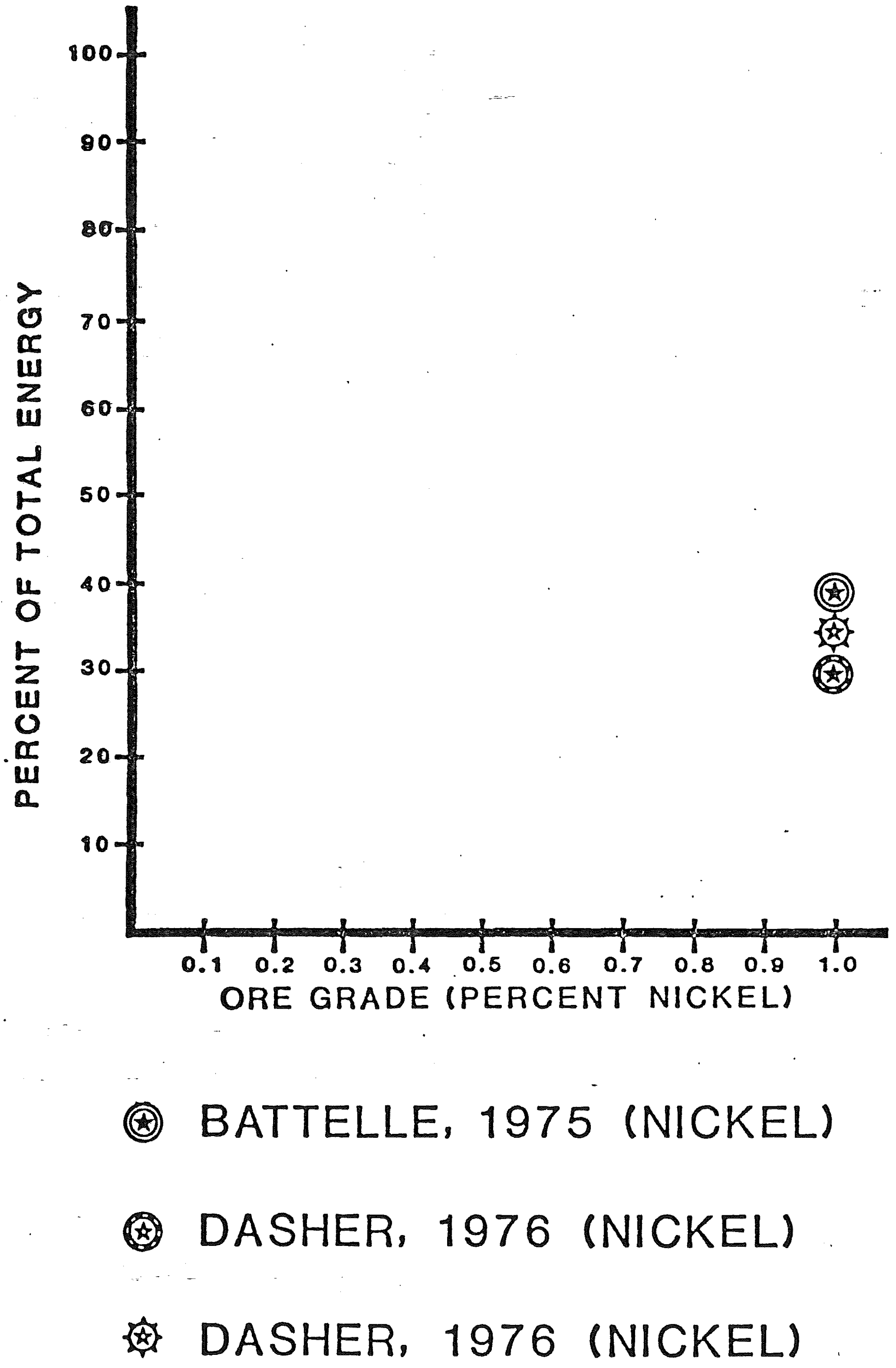


Figure 157. DISTRIBUTION OF ENERGY CONSUMED IN THE PRODUCTION OF NICKEL SMELTING[9,11,12]

(AS A PERCENT OF TOTAL ENERGY CONSUMPTION)

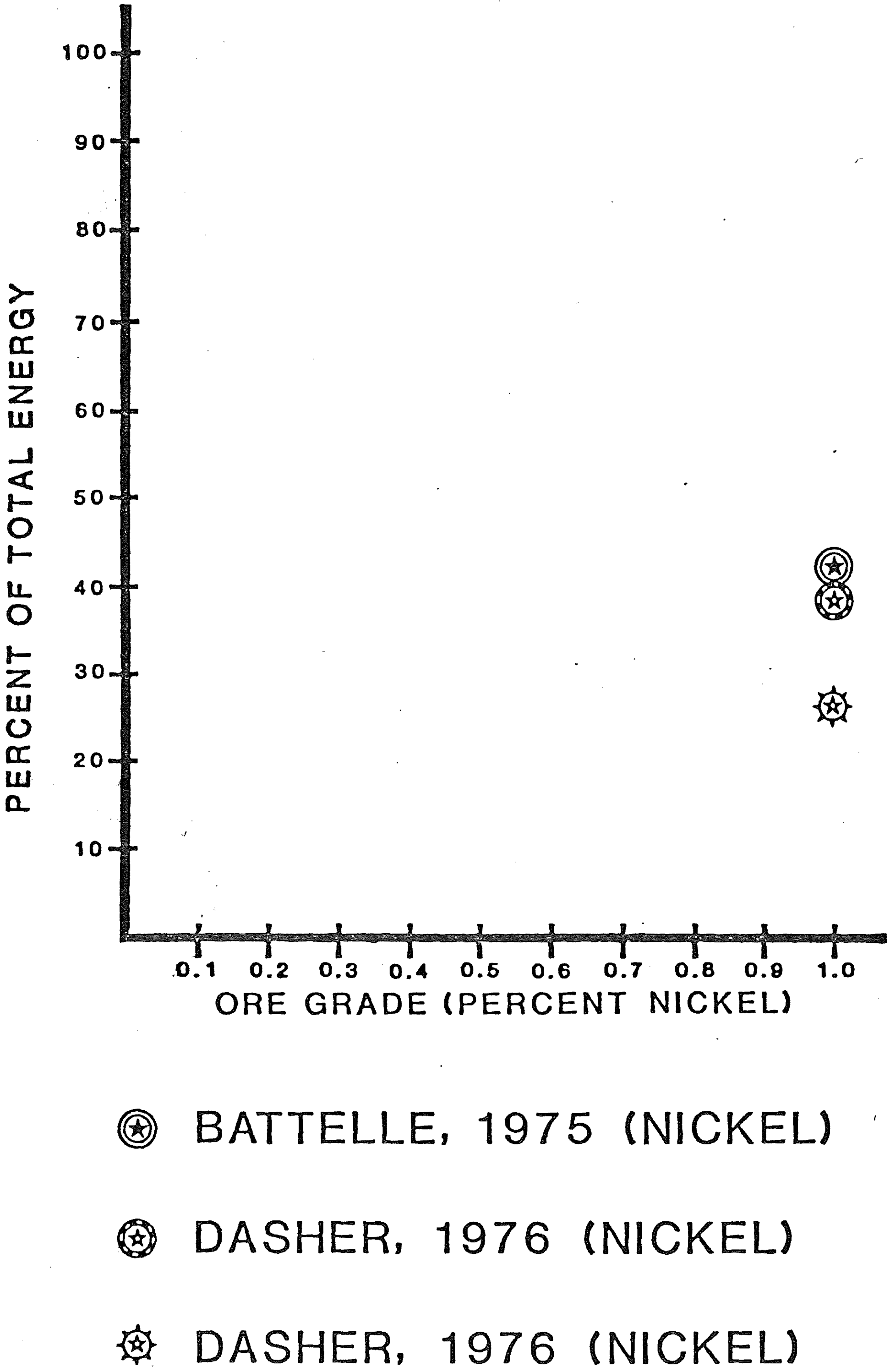


Figure 158. DISTRIBUTION OF ENERGY CONSUMED IN THE PRODUCTION OF NICKEL REFINING[9,11,12]

(AS A PERCENT OF TOTAL ENERGY CONSUMPTION)

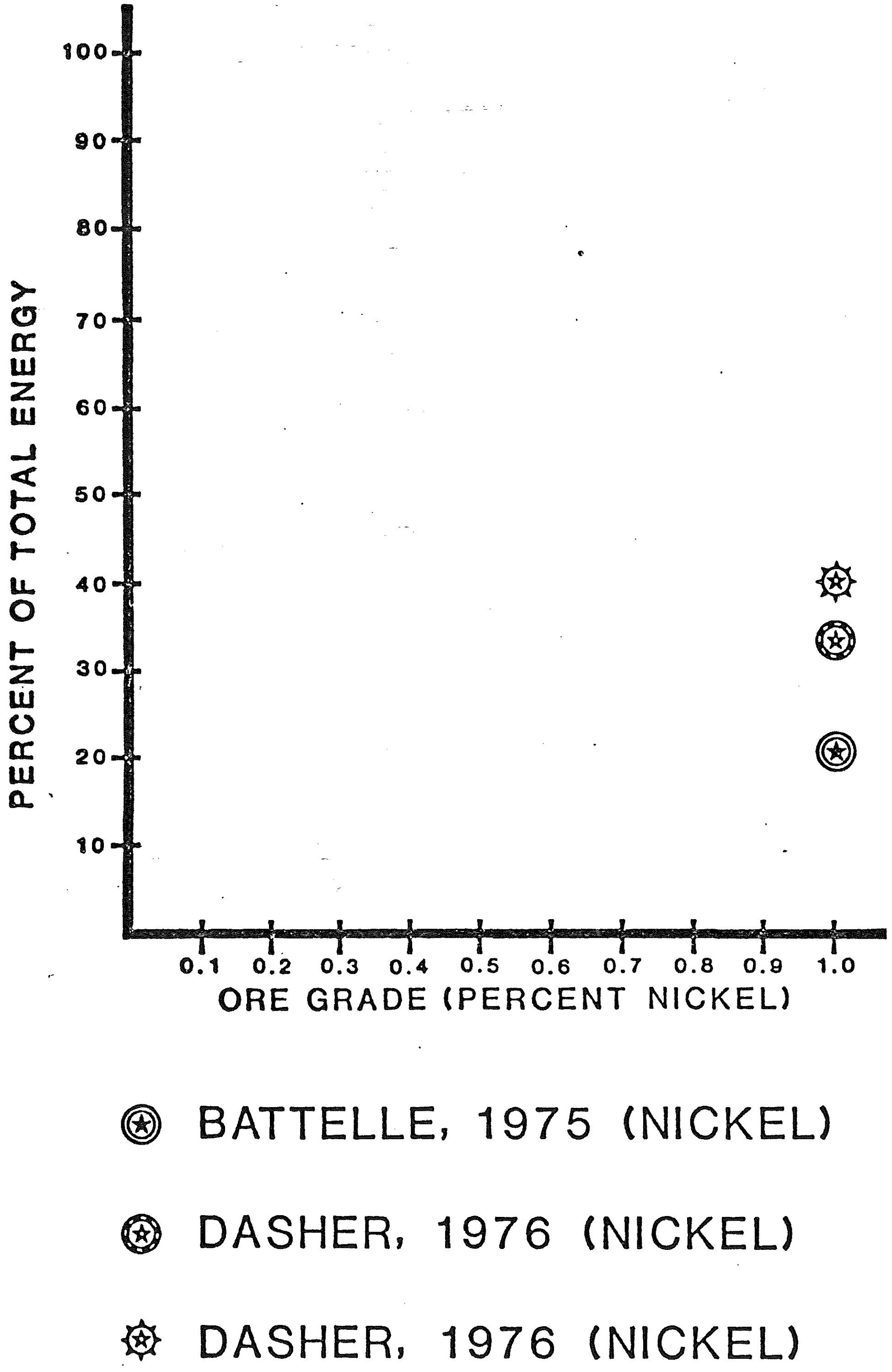


Figure 159. DISTRIBUTION OF ENERGY CONSUMED IN THE PRODUCTION OF NICKEL

(AS A PERCENT OF TOTAL ENERGY CONSUMPTION)

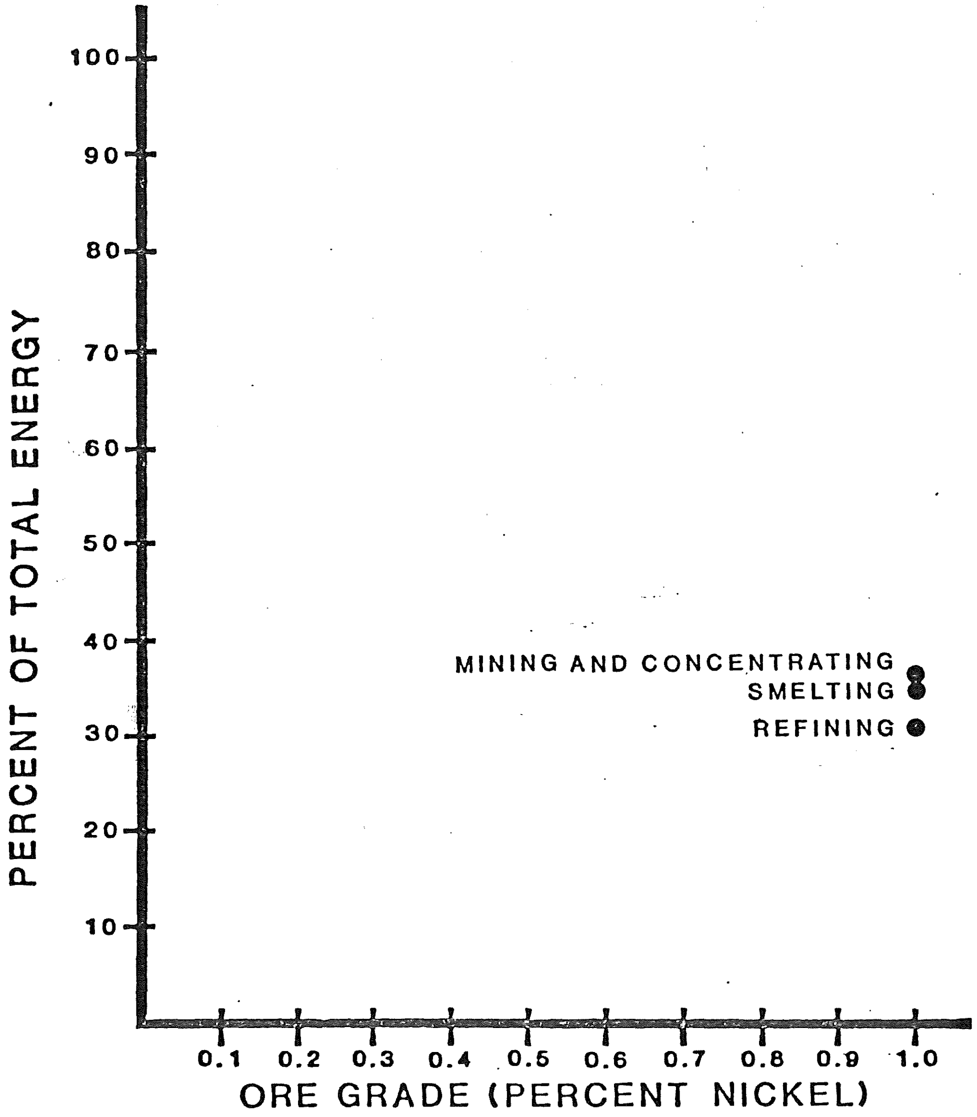


Table 32.

ENERGY REQUIREMENTS FOR VARIOUS TYPE SMELTERS[4,11,12,14,18,26,28,29,31,32]

(SMELTING, CONVERTING, SLAG TREATMENT, ANODE CASTING)

SOURCE	FURNACE TYPE (10^6 KCAL/MET. TON CU)					
	REVERBERATORY	ELECTRIC	OUTOKUMPU	INCO	NORANDA	MITSUBISHI
Biswas (1976)[1]	9.999-16.153	5.077	6.307	1.385	5.077	7.538
USBM (1976)	5.663-8.315	7.215	3.354	1.904	2.348	——
EPA (1976)	6.390-7.224	——	3.667-4.167	——	3.473	3.334
Battelle (1975)	7.567	——	——	——	——	——
Battelle (1974)	4.029-6.529	——	——	——	——	——
Themelis (1976)[3]	7.640	4.862	3.473	——	3.473	3.473
Themelis (1976)[3]	15.975-23.616[2]	——	——	——	10.974-19.448[2]	——
Sharma (1976)[3]	12.627	8.891	5.876	——	6.418	——
Sharma (1976)[3]	21.810[2]	11.905[2]	8.140[2]	——	8.168[2]	——
M. W. Kellogg (1976)[4]	4.498	5.849	3.441	2.057	4.647	3.073
M. W. Kellogg (1976)[4]	5.129[2]	6.747[2]	4.286[2]	2.762[2]	5.876[2]	3.876[2]
Henderson (1974)	5.909-7.186	9.098	5.188	——	——	——
AVERAGE VALUES						
SOURCES ACCOUNTING FOR ACID TREATMENT	15.447	7.910	6.249	2.762	9.909	5.707
SOURCES NOT ACCOUNTING FOR ACID TREATMENT	6.965	7.183	4.167	1.782	4.072	3.293
ALL SOURCES	9.792	7.456	4.790	2.027	6.990	4.259

---

1. Data adjusted slightly to include energy for anode casting, also includes energy requirements for acid plant treatment.
2. Data includes energy requirements for acid plant treatment.
3. Concentrate assumed to average 20% copper.
4. Concentrate averages 30% copper, no nickel, no impurities (numbers show only how process design influences energy use).

Referring to the average values of Table 32, it is possible to realize savings of energy of 25 to 80 percent depending, of course, upon the particular choice or selection of furnace. The more recent smelting furnace designs indicate a distinct advantage in the effective use of energy over the conventional smelting furnace designs.

WATER REQUIREMENTS OF THE NONFERROUS METALS INDUSTRY (Smelting and Refining)

Very limited data is available for making estimates on typical amounts of water needed in the smelting and refining of copper (and nickel). The EPA, however, has summarized data which suggests that an average of 7700 gallons of water per metric ton of copper are required for a smelter operation, and an average of 2100 gallons of water per metric ton of copper are required for a refining operation.[7] Battelle reinforces the figures relating to smelting. No information has been found relative to nickel smelting and/or refining.

Table 33 indicates the typical water discharges experienced in copper smelting and refining processes where discharges occur. Since no information has been found relative to the treatment of nickel, it will be assumed that discharges occur similar to that of copper.

CAPITAL AND OPERATING COSTS FOR SMELTING AND REFINING PROCESSES

Tables 34 and 35 outline the capital and operating costs of the major copper-extraction processes (smelting and refining). Emphasis is placed on the well-established processes of smelting and refining, as well as the newer processes.

Table 33. WATER DISCHARGES FROM COPPER SMELTING AND REFINING PROCESSES[10,27,33]

WATER POLLUTION SOURCE	DISCHARGE (GALLONS/METRIC TON CU)
Smelter Discharges	
Slag Granulation Water[1]	1323
Anode Contact Cooling Water[1]	198
Acid Plant Bleed[1] Demister Acid Drip Scrubber Water Bleed Electrostatic Precipitator	794
Sub Total[1]	2315
Refinery Discharges	
Cathode Contact Cooling Water[1]	198
Product Casting Cooling Water[2]	320
Other Electrolytic Bleed Scrap Anode Rinse Scrubber Water Bleed Barometric Condensor	276
Sub Total[3]	794
TOTAL	3109

[1] EPA (1976)

[2] EPA (1975)

[3] Williams (1975)

Table 34.

CAPITAL COSTS FOR VARIOUS TYPE SMELTERS INCLUDING SULFURIC ACID TREATMENT

	SOURCE (1977 $/ANNUAL METRIC TON Cu)						
FURNACE TYPE	Biswas(1976)[1]	USBM(1976)	MRRC(1975)	Hays(1974)	EPA(1976)[2]	Gray(1977)	AVERAGE VALUES
Reverberatory	1329.00	952.00-1532.00		1635.00	745.00-859.00		1175.00
Electric	1550.00	1242.00	1772.00				1521.00
Outokumpu	1108.00	1211.00			859.00	1166.00-1458.00	1160.00
INCO	1108.00	1159.00					1134.00
Noranda	1108.00	1335.00			859.00		1101.00
Mitsubishi	1218.00				859.00		1039.00
Electrolytic Refining	332.00				512.00	420.00-525.00	447.00

[1] Copper ore @ 1% Cu.

[2] Copper sulfide concentrate @ 28.6% Cu, 29.3% Fe, 33.41% S.

Table 35.

DIRECT OPERATING COSTS FOR VARIOUS TYPE SMELTERS INCLUDING SULFURIC ACID TREATMENT

	SOURCE (1977 $/METRIC TON Cu)					
FURNACE TYPE	Biswas(1976)[1]	USBM(1976)	MRRC(1975)	EPA(1976)[2]	Gray(1977)	AVERAGE VALUES
Reverberatory	288.00	157.00-188.00		204.00-208.00		209.00
Electric		195.00	177.00			186.00
Outokumpu		166.00		166.00	162.00-194.00	172.00
INCO		166.00				166.00
Noranda		173.00		148.00		160.00
Mitsubishi				163.00		163.00
Electrolytic Refining	111.00			84.00	53.00-64.00	78.00

[1] Copper ore @ 1% Cu.

[2] Copper sulfide concentrate @ 28.6% Cu, 29.3% Fe, 33.41% S.

Smelter and electrolytic refinery direct capital costs are almost independent of the type and grade of the ore. Refineries (copper in particular) are standard in nature, and their capital costs can be predicted accurately. Smelter direct capital costs vary slightly according to the type of smelting method (Reverberatory, Electric, Flash, INCO, Noranda, Mitsubishi). This effect, however, is not large due to the fact that the smelting facilities (flues, stack, dust collection, acid plant, converters, anode casting facilities) are common to all smelting methods.[4]

ENERGY, WATER, COST, MANPOWER, LAND, AND MAJOR EQUIPMENT NEEDS FOR THE SMELTER/REFINERY MODEL

Table 36 summarizes the energy, water, cost, manpower, land, and major equipment needs for the smelting model. Energy requirements are broken down by percentage and category as found in Table 37.

Direct (fixed) capital costs are broken down by percentage and category a found in Table 38. Indirect capital costs (i.e. engineering, project management, contingencies, fees, purchases, etc.) are assumed to equal 76 percent of direct capital costs. Direct operating costs (i.e. excluding depreciation, capital repayment and income taxes) are also broken down by percentage and category as found in Table 39.

Manpower requirements have been determined on the basis of a single smelter/refinery complex and is as shown in the smelter/refinery organization chart (Figure 160). It has been assumed that 70 percent of the total work force will be associated with the smelter, and that the remaining 30 percent, with the refinery.

Table 36. Energy, Water, Cost, Manpower, Land, and Major Equipment Needs for a Copper-Nickel Smelter (Values Exclude Contributions from Intermediate Treatment by Mineral Processing).

CATEGORY	UNIT OF MEASURE	AVE. VALUE	NUMBER OF UNITS
SMELTER			
Energy			
Copper	10^6 KCAL/Met. Ton Cu	6.249	
Nickel	10^6 KCAL/Met. Ton Ni	16.7	
Metal	10^6 KCAL/Met. Ton Metal ($\frac{Cu}{Ni}$=7.00)	7.56	
Water			
Intake	Gallons/Met. Ton Metal	7,700	
Recirculating		42,300	
Discharge		2,315	
Cost			
Direct Capital	1977$/Met. Ton Metal	1,160	
Indirect Capital		886	
Operating		172	
Manpower			
Administrative	Number of People	41	
Operating, Maintenance, & Other		394	
Land Requirement	Acreage	55	
Principal Equipment Needs			
Dryer			1
Flash Furnace			1
Electric Furnace			2
Converter			2
Anode Furnace			1
Boiler Systems			3
Gas Cleaning Systems			3
Electrostatic Units			9
Scrubbers			3
Double Contact Acid Plant (includes Spray Towers, Demisters, Scrubbers, Precipitators, etc.)			1
Support Equipment			-
Grinding, Processing, and Flotation Equipment			-

Table 37. Summary of Energy Requirements for Flash Smelting.[4]

CATEGORY	PERCENT OF TOTAL ENERGY USE
Receiving of Concentrate, Crushing and Storage	0.3
Furnace Smelting	51.4
Converter Operation	10.3
Anode Furnace, Anode Casting	8.9
Sulfuric Acid Plant	29.1
TOTAL	100%

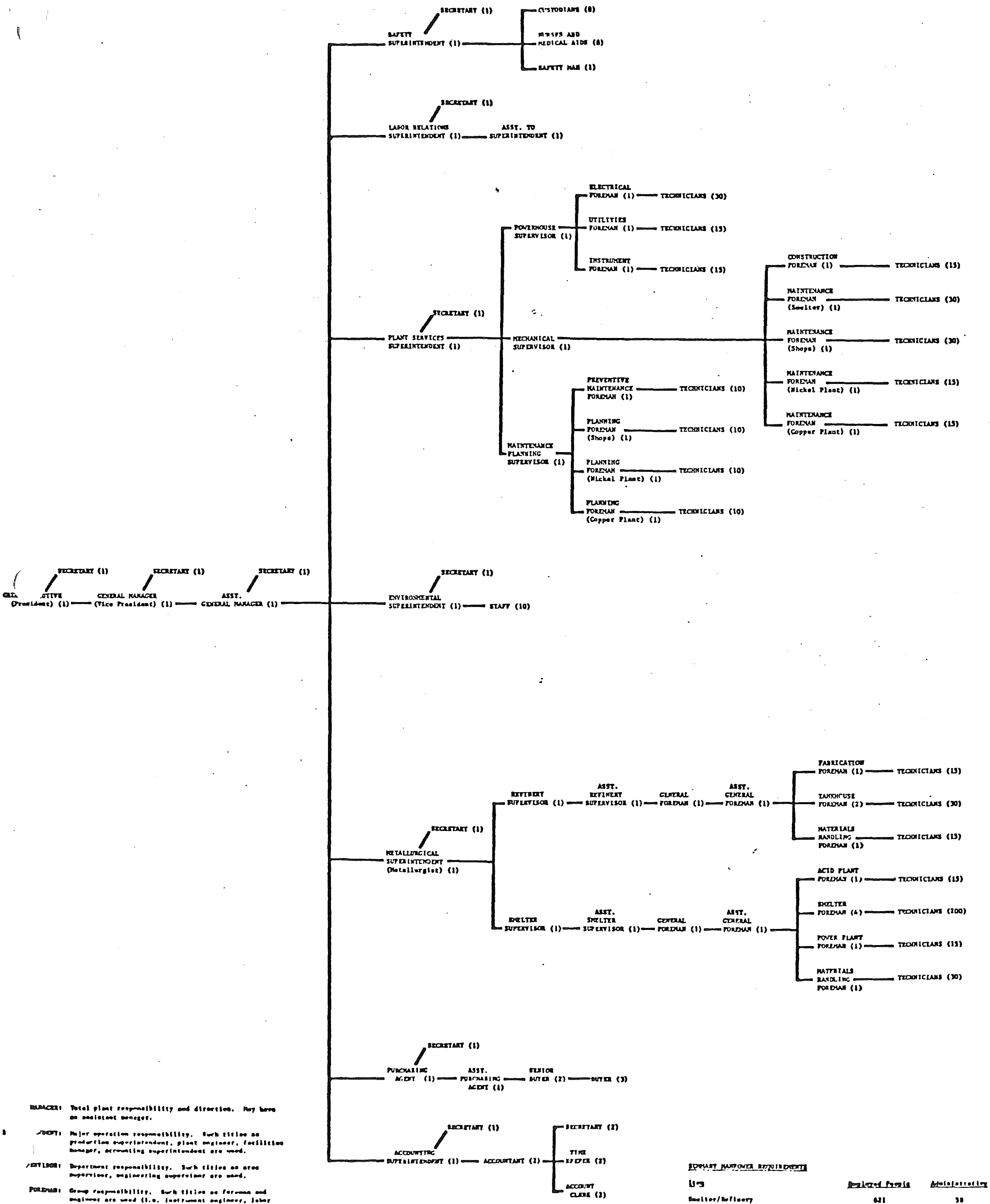
Table 38. Capital Costs of the Various Parts of a Smelter Expressed as a Percentage of the Total Direct (Fixed) Capital Cost. The Costs Include Installation and Housing of the Units.[4]

CATEGORY	PERCENT OF TOTAL DIRECT CAPITAL COST
Concentrate Handling Facilities, Conveying Equipment, Cranes	5
Smelting Furnace, Including Wasteheat Boilers, and Slag-Disposal Equipment	35
Converters, Including Running Gear and Blowers	15
Rotary Anode Furnace	5
Anode Casting Wheel	5
Gas Collection, Flues, Chimney Stack	10
Prec., Dust Removal and Collection	15
Sulfuric Acid Plant	10
TOTAL	100%

Table 39. Direct Operating Costs for Producing Anodes in a Smelter.[4]

CATEGORY	PERCENT OF TOTAL DIRECT OPERATING COST
Receiving of Concentrate, Storage, Distribution	3.9
Smelter Furnace Operation, Matte Delivered to Converters, Slag Disposal Complete, Includes Refractory Costs	33
Converter Operation, Includes Refractory Costs	19.2
Anode Furnace, Anode Casting, Handling, Loading for Transport	7.7
Prec. and Acid Plant Operation	17
Engineering (Maintenance Labor, Supplies and Equipment, Not Including Purchast Cost of Refractories, Replacement of Moving Equipment)	7.7
SUB-TOTAL	88.5
Local Administration, Supervision, Laboratory, General Local Overhead	11.5
TOTAL	100%

Figure 160. Smelter/Refinery Organization Chart.



MANAGER:	Total plant responsibility and direction. May have an assistant manager.
SUPERINTENDENT:	Major operation responsibility. Such titles as production superintendent, plant engineer, facilities manager, accounting superintendent are used.
SUPERVISOR:	Department responsibility. Such titles as area supervisor, engineering supervisor are used.
FOREMAN:	Group responsibility. Such titles as foreman and engineer are used (i.e. instrument engineer, labor foreman, pipefitter foreman, mechanical engineer, etc.)
TECHNICIANS:	Wage roll.

SUMMARY MANPOWER REQUIREMENTS

Site	Employed People	Administrative	Operating & Maintenance
Smelter/Refinery	621	58	563
Smelter (70% of total manpower needs)	435	41	394
Refinery (30% of total manpower needs)	186	17	169

Table 40 summarizes the energy, water, cost, manpower, land, and major equipment needs for the refinery model. Direct capital costs are broken down by percentage and category as found in Table 41. Indirect capital costs are assumed to equal 78 percent of direct capital costs. Direct operating costs are also broken down by percentage and category as found in Table 42.

Manpower requirements for the refinery have been explained earlier in the smelter model. Refer to the smelter/refinery organization chart (Figure 160) for more detail.

Table 40. Energy, Water, Cost, Manpower, Land, and Major Equipment Needs for a Copper-Nickel Refinery.

CATEGORY	UNIT OF MEASURE	AVE. VALUE	NUMBER OF UNITS
REFINERY			
Energy			
Copper	10^6 KCAL/Met. Ton Cu	2.6	
Nickel	10^6 KCAL/Met. Ton Ni	7.9	
Metal	10^6 KCAL/Met. Ton Metal$(\frac{Cu}{Ni}=7.00)$	3.26	
Water			
Intake	Gallons/Met. Ton Metal	2,100	
Recirculating		7,900	
Discharge		794	
Cost			
Direct Capital	1977$/Met. Ton Metal	447	
Indirect Capital		349	
Operating		78	
Manpower			
Administrative	Number of People	17	
Operating, Maintenance, & Other		169	
Land Requirement	Acreage	25	
Principal Equipment Needs			
Copper			
Anode Furnace			1
Tankhouse			331
Cells (Production)			19
Cells (Starter Sheet)			10
Cells (Electrolyte Purification)			1
Refining Furnace			1
Support Equipment			-
Nickel			
Grinding Mill			1
Filters			5
Leaching Tanks			5
Thickeners			2
Tankhouse			1
Cells (Production)			45
Agitation Tanks			5
Support Equipment			-

Table 41. Direct (Fixed) Capital Costs of Various Parts of an Electrolytic Refinery Expressed as a Percentage of the Total Direct (Fixed) Capital Cost.[4]

CATEGORY	PERCENT OF TOTAL DIRECT CAPITAL COST
Anode Receiving Facilities, Anode and Cathode Transportation Equipment	14.3
Anode Scrap Furnaces, Anode Casting Facilities	14.3
Starting Sheet Preparation Equipment, Including Starting Sheet Cells and Electrical Equipment	14.3
Production Electrolytic Cells, Including Transformers, Rectifiers, Electrical Distribution System	50.0
Electrolyte Purification (Liberators)	7.1
TOTAL	100%

Table 42. Direct Operating Costs for Producing Cathodes from Anodes in an Electrolytic Refinery.[4]

CATEGORY	PERCENT OF TOTAL DIRECT OPERATING COST
Receiving and Handling of Anodes, Including Remelting and Casting of Anode Scrap	9.9
Starting Sheet Preparation	10.0
Production Electrorefining	50.1
Purification of Electrolyte (not including byproduct recovery)	9.9
Engineering, Maintenance, Replacement of Moving Equipment	10.0
Local Administration, Supervision, Laboratory, General Overhead	10.1
TOTAL	100%

APPENDIX A

Production data of various industrial metallurgical processes.

(Source: Biswas, 1976)

PRODUCTION DATA

	Lubumbashi, Zaire, (Claus and Guebels, 1967)	Shisakajima, Japan (Momoda *et al.*, 1970)	Shisakajima, Japan (Momoda *et al.*, 1970)	Copper–Nickel, Falconbridge (Falconbridge, 1963)
Furnace size				
W × l × h (inside) (m)	1.5 × 6.1 × 6	1.3 × 7.3 × 5.6	1.2 × 5.5 × 4.3	1.5 × 7.6 × 5.2
Shaft volume (m^3)	55	60	28	60
Hearth area (m^2)	9.2	9.5	6.6	11.4
Production details				
Type of charge	Sinter + solid conv. slag	Wet conc. (31% Cu) + solid conv. slag	Wet conc. (31% Cu) + solid conv. slag	Sinter 8% Ni + 4% Cu (plus ore and solid conv. slag)
Solid charge rate (excluding coke (tonnes/day)	1190	320	250	1300
Average % Cu in charge	27	20	20	6Ni, 3Cu
Matte (tonnes/day)	490	140	110	650
Matte grade % Cu	63	46	46	12Ni, 6Cu
Slag (tonnes/day)	700	160	120	600
% Cu in slag	1.0	0.4	0.4	0.16Ni, 0.15Cu
Productivity (tonnes of charge per day/m^2)	130	35	35	115
Fuel				
Coke (tonnes/day)	131	15	11	126
Oil in tuyeres (tonnes/day)	none	8	6	none
Total energy in fuel (kcal/tonne of charge)	8×10^5	6×10^5	6×10^5	7×10^5

Table A-1. Production details of copper matte and copper-nickel matte blast furnaces.

	Mount Isa (Middlin *et al.*, 1970)	Luanshya (Verney, 1967)	Kennecott, Utah (Anderson, 1970)	Onahama (Niimura *et al.*, 1973)	Noranda wet charge (1974)	Noranda calcine charge (1974)
Furnace size						
W × l × h (m)	10 × 33 × 4	9½ × 34 × 4	9 × 35 × 4	10 × 34 × 4	10½ × 35 × 4	10½ × 35 × 4
Hearth area (m^2)	330	320	315	340	360	360
Fuel	Pulverized coal	Pulverized coal	Natural gas	Bunker C oil	Bunker C oil	Bunker C oil
Rate (tonnes/day)	115	70	1.3×10^5 Nm^3/day	115 The air is enriched to 23% O_2	76 Natural gas 2.8×10^4 Nm^3 (plus 1.2×10^4 Nm^3 of nat. gas per day for air preheating)	80 (plus 10^4 Nm^3 of natural gas per day for air preheating)
Production details						
Type of charge	Calcine (21% Cu)	Wet concentrate (30% Cu)	Wet concentrate (29% Cu)	Wet concentrate (30% Cu)	Wet concentrate (25% Cu)	Calcine (19% Cu)
Solid charge rate (tonnes/day)[a]	(Total) 1300 (Calcine) 1100	(Total) 400 (Conc.) 350	(Total) 900 (Conc.) 800	(Total) 1150 (Conc.) 950	(Total) 720 (Conc.) 625	(Total) 1100 (Calcine) 1000
Average % Cu in charge	19%	26%	30%	25%	22%	18%
Converter slag (tonnes/day)	400	260	420	450	480	470
Matte (tonnes/day)	650	250	700	800	650	600
Matte grade (% Cu)	36	47	41	34	34	40
Discard slag (tonnes/day)	750	260	480	840	450	900
% Cu in slag	0.38	0.6	0.41	0.47	0.43	0.57
Productivity						
(tonnes of charge per day/m^2)	3.6	1.2	2.9	3.4	2	3
Energy requirement						
kcal/tonne of charge	8×10^5 (plus an additional 2×10^5 kcal in the roaster)	15×10^5	13×10^5	With O_2 enrichment, 13×10^5 Without 16×10^5	15×10^5 plus 1×10^5 for air preheating	8×10^5 plus 1×10^5 for combustion air heating plus 1×10^5 in roasters

[a] Excluding converter slag.

Table A-2. Production details of industrial reverberatory furnaces. Charges and analyses are on a dry basis.

				Cu–Ni matte smelters[a]		
	Boliden (Sweden) (Herneryd *et al.*, 1954)	Jinja (Uganda) (Treilhard, 1965)	Inspiration (Arizona) (Dayton, 1974) design data only	INCO (Thompson) (Boldt and Queneau, 1967) updated to 1975	Rustenburg (South Africa) (Mostert, 1973)	Copper smelting from dead roasted concentrates Brixlegg, Austria (Kettner *et al.* 1972)
Furnace size				2 Furnaces		
W × l × h (m)	6 × 24 × 3	6 × 14 × 3	10 × 35 × 5	8.5 × 30 × 5	7 × 26 × 4.5	5 m diameter
Hearth area (m^2)	144	84	350	260	181	20
kVA (kilovolt-amperes)	12,000	5500	50,000	30,000	19,500	2500
Power density (kW/m^2)	60	60	120 (max)	115	100	125 max
Electrodes						
Number	6	3	6	6	6	3
Diameter (m)	1.2	1.1	1.8	1.4	1.1	0.8
Current density (A/cm^2)	2–3	2–3		2.0	2.6 (max)	
Consumption (kg/tonne of charge)	1.6	2.5		2.1	2.7	2.5
Production details						
Type of charge	Hot calcine (700°C) and dry concentrate	Concentrate (7% H_2O)	Dry concentrate	Calcine	Dry concentrate pellets (2% H_2O)	Dead roasted calcine
Solid charge rate (tonnes/day)	500 (total)	250	1650 (total)	1500 per furnace (including 300 tonnes of flux)	565 (total)	75 (total) plus 6 (coke)
	450 (calcine plus concentrates)		1500 (concentrates plus precipitates)		435 (concentrates)	55 (calcine)
Average % Cu in charge	15	20		0.2 (5% Ni)	2 (4% Ni)	25
Converter slag (tonnes/day)	130	150		360	110	nil
Matte (tonnes/day)	200	150	1100	330	110	20 (metal)
Matte grade (% Cu)	36	30–40	40	22.5 (Cu + Ni)	27 (Cu + Ni)	92.5
Discard slag (tonnes/day)	400	160		1360	500	35
% Cu in slag	0.4	0.63		0.18 (Cu + Ni)	0.16 (Cu + Ni)	0.7
Productivity (tonnes of charge per day/m^2)	3.5	3.0	3.7	5.8	3.1	3.8
Energy requirement						
kWh/tonne of charge	400	420		430	690	520
or						
kcal/tonne of charge	3.5×10^5	3.6×10^5		3.7×10^5	5.9×10^5[b]	4.5×10^5 plus coke 5.5×10^5 oil 0.9×10^5 Total 10.9×10^5

[a] INCO also have three 18,000-kVA furnaces. Falconbridge are replacing their blast furnaces with electric furnaces.

[b] This anomalously high-energy requirement is probably due to (a) the moisture content and (b) a high temperature (1350°C) of furnace operation.

Table A-3. Product details of industrial matte smelting electric furnaces.

PRODUCTION DATA - (contd.)

	INCO, Canada (Merla *et al.*, 1972, updated to 1975)	Tamano, Japan (Kitamura *et al.*, 1973)	Hitachi, Japan (Yasuda, 1974)	Toyo, Japan (Ogura, 1973, 1974)
Furnace size				
W × l × h (m)	6 × 23 × 5	7 × 20 × 3	7 × 19 × 3	7 × 20 × 3
Hearth area (m^2)	138	140	133	140
Burner tower	none			
Diameter × height from roof (m)		6 × 6	5.7 × 6	6 × 6
Concentrate burners	4	4	3	3
Inlet air temperature	20°C	450°C	930°C	450°C
Gas offtake				
Length × width (m)	6 × 3	2.5 diameter	7 × 3	7 × 3
Height (m)	6	7	6	6
Production details				
Type of charge	Dry concentrate (30% Cu)	Dry concentrate (30% Cu)	Dry concentrate (26% Cu)	Dry concentrate (28% Cu)
Solid charge rate (tonnes/day)	(Total) 1600	(Total) 1200	(Total) 1200	(Total) 1100
Average % Cu in charge	25	24	20	25
Converter slag (tonnes/day)	None	None	None	None
Matte (tonnes/day)	850	600	550	550
Matte grade (% Cu)	47	47	49	49
Flash furnace slag (tonnes/day)	550	(500)	450	300
% Cu in slag	0.7	0.55	≈ 1	≈ 1
% SiO_2 in slag	36	36	34	40
Auxiliary Cu recovery system	None	Electric furnace	Electric furnace	Electric furnace
% Cu in final discard slag	0.7	within flash furnace, 0.55% Cu	0.5	0.45
Productivity (tonnes of charge per day/m^2)	11.6	8.6	9.0	7.9
Fuel (Bunker C oil, tonnes/day)				
Flash furnace combustion tower		37	11	32
Flash furnace hearth		7 plus 5.2×10^4 kWh/day	17	16
Air preheater		none	31	15 (est.)
Charge dryer	≈ 7	7	9	None (uses waste gases)
Total	≈ 7	51 $+0.5 \times 10^5$ kWh	68	63
Oxygen (tonnes/day)	300 ($95–97\%\ O_2$)	None	40 ($99.5\%\ O_2$)	None
Energy requirement, including O_2 manufacture (kcal/tonne of charge)	1.1×10^5	4.5×10^5	5.8×10^5	5.8×10^5
Energy in auxiliary Cu recovery system (kcal/tonne of flash-furnace charge)			$\approx 0.3 \times 10^5$	0.3×10^5
Total energy requirement (kcal/tonne of charge)	1.1×10^5	4.5×10^5	6.1×10^5	6.1×10^5

Table A-4. Production details of industrial flash smelting furnaces.

PRODUCTION DATA - (contd.)

	INCO flash furnace (Merla, 1972 updated)	Outokumpu type flash furnace (Table 8.1)
Productivity (tonnes of charge per day/m^2)	11.6	8.7
Dust loss in effluent gas (% of charge)	2	5–10
SO_2 concentration in gas (vol.%)	80	10–15
Volume of effluent gas (Nm^3 per tonne of charge)	175	1100
Copper concentration in slag (%)	0.7	≈1
Auxiliary facilities	Oxygen plant	Air preheater or heat exchanger Small electric furnace for recovering copper from slag
Fuel required per tonne of charge (kg)	8 (Oil in concentrate dryer)	55 (oil)
Oxygen required per tonne of charge (kg)	200 (95% O_2)	
Electrical energy in slag retreatment furnace per tonne of charge (kWh)		40 (Section 10.3)

Table A-5. Comparative operating data for INCO and Outokumpu flash smelting furnaces.

Size of reactor Total plan area	21 m long, 5 m diameter ≈100 m^2	
Operation (tuyere blast)	Air	Air enriched to 30.5% O_2
Charges	Tonnes per day (dry basis)	
Wet concentrate: 10% H_2O 25% Cu (dry basis)	730	1200
Slag concentrate from slag flotation plant, 55% Cu (dry basis)	165	250
Silica flux	150	250
Revert dusts (10% Cu)	30	60
Pure oxygen		200
Products	Tonnes per day	
Blister copper (98% Cu, 0.1% Fe, 1.5% S)	180	300
Slag (12% Cu, 36% Fe, 22% SiO_2) to slag flotation circuit	700	1150
Revert dusts	30	60
Productivity, tonnes of new charge smelted per day per m^2 of plan area	9	15
Fuel requirement (total) kcal per tonne of new charge (dry basis)	11×10^5	3×10^5*

*Excluding energy for oxygen manufacture (≈400 kWh/tonne of oxygen).

Table A-6. Production data for the industrial Noranda process reactor.

PRODUCTION DATA - (contd.)

Size of reactor	Shown in Fig. 11.3
Total plan area	50 m^2
Charges	Rates (dry tonnes/day)
Dry concentrates: 25% Cu	85
Silica flux	25
Revert dusts (10% Cu)	5
Products	
Blister copper (0.9% S)	20
Slag (0.5% Cu, 35% SiO_2, 40% Fe)	65
Revert dusts	5
Productivity: tonnes of new charge per day per m^2 of plan area	2–3
Gases	
Oxidation branch	8–12% SO_2
Slag-settling branch	1–2% SO_2
Fuel requirement	"same order as in large scale wet charge reverberatory smelting"

Table A-7. Production data for the WORCRA pilot reactor.

PRODUCTION DATA - (contd.)

Production details	Semi-commercial unit[a]	Commercial unit
Smelting furnace	Rates	(Dry tonnes per day)
Charges:		
Wet concentrate	120 (25% Cu)	500 (27% Cu)
Silica and lime fluxes	25	200
Revert dusts	5	15
Solid recycle slag from converting furnace (10–15% Cu)	50	55
Pure oxygen (30% O_2 in blast)	7	55
Products:		
Matte and slag to electric settling furnace		
Electric settling furnace		
Charge: pyrite reductant	NR	3 (or less)
Products:		
Matte to converting furnace	60 (60% Cu)	225 (65% Cu)
Discard slag (0.5% Cu, 30–40% Fe, 30–35% SiO_2 5–10% CaO, 2-6% Al_2O_3)	90	440
Converting furnace		
Charge: limestone ($CaCO_3$) flux	15	15
Pure oxygen (25% O_2 in blast)	1	0
Products:		
Blister copper (0.1–0.8% S, 99–99.5% Cu)	30	135
Recycle converting furnace slag (10–15% Cu, 65–70% Fe_3O_4, 10–20% CaO, little or no SiO_2) Recycle to smelting furnace	50	55
Productivity, tonnes of new charge smelted per day per m^2 of plan area (3 furnaces)	2–3	furnace dimensions not given
Fuel requirement	kcal/tonne of charge	kcal/tonne of charge
Smelting furnace	6×10^5	6×10^5
Electric settling furnace	0.4×10^5	0.3×10^5
Converting furnace	0.6×10^5	0
Total	7×10^5	6×10^5

[a] Smelting furnace: 7 m diameter, 2–4 lances. Electric settling furnace: 3½ m dia, 250 kVA. Converting furnace: 3 m diameter, 2–3 lances.

Table A-8. Production data from semi-commercial and commercial Mitsubishi continuous smelting systems.

APPENDIX B

Pictorial summary of particulate removal pollution equipment.

(Source: Ross, 1972)

MECHANICAL COLLECTORS

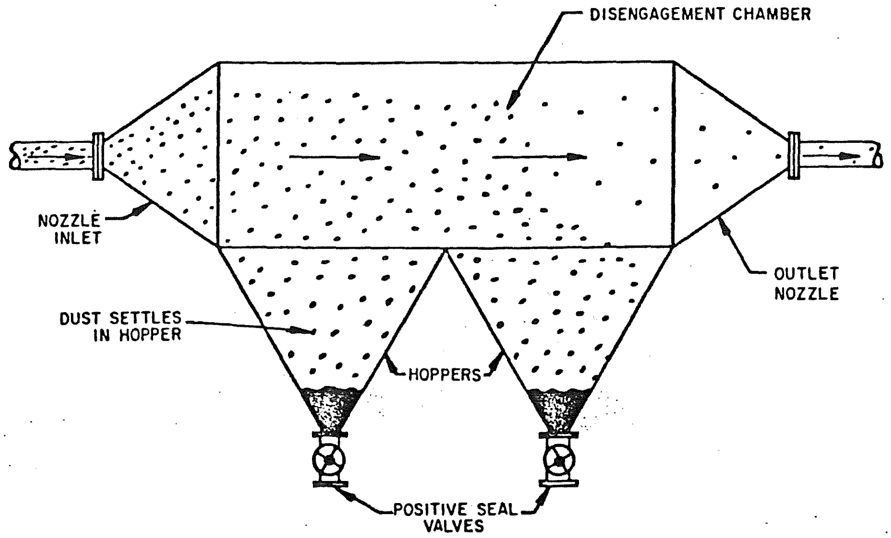


Figure B-1. Gravity settling chamber.

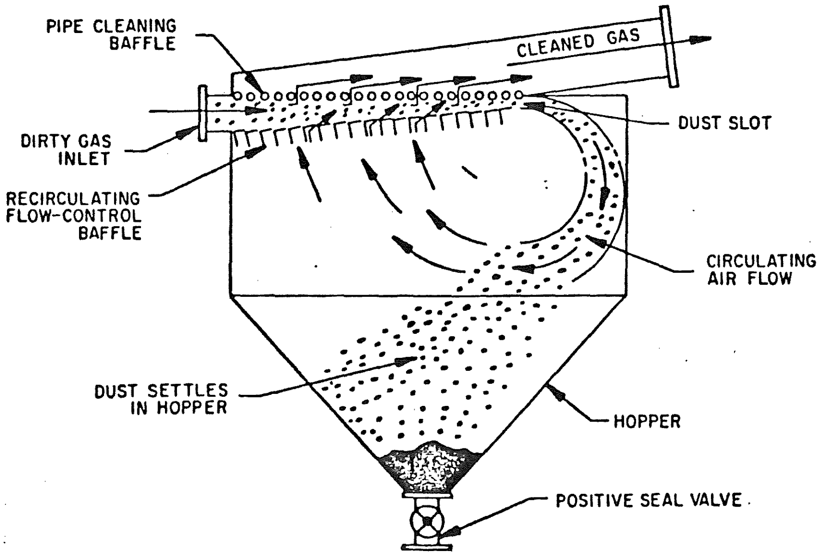


Figure B-2. Recirculating baffle collector.

MECHANICAL COLLECTORS - (contd.)

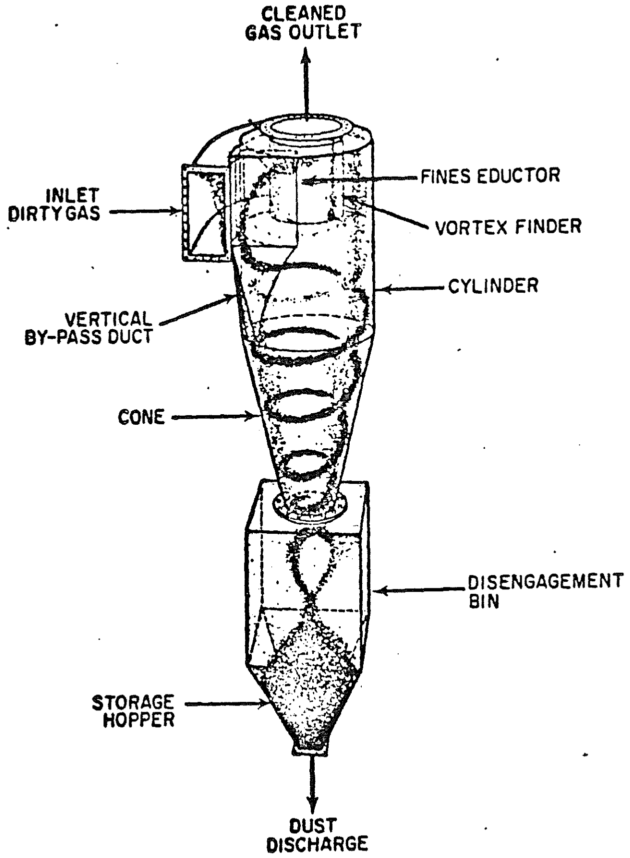


Figure B-3. Flow pattern in mechanical cyclone with vertical fines eductor.

MECHANICAL COLLECTORS - (contd.)

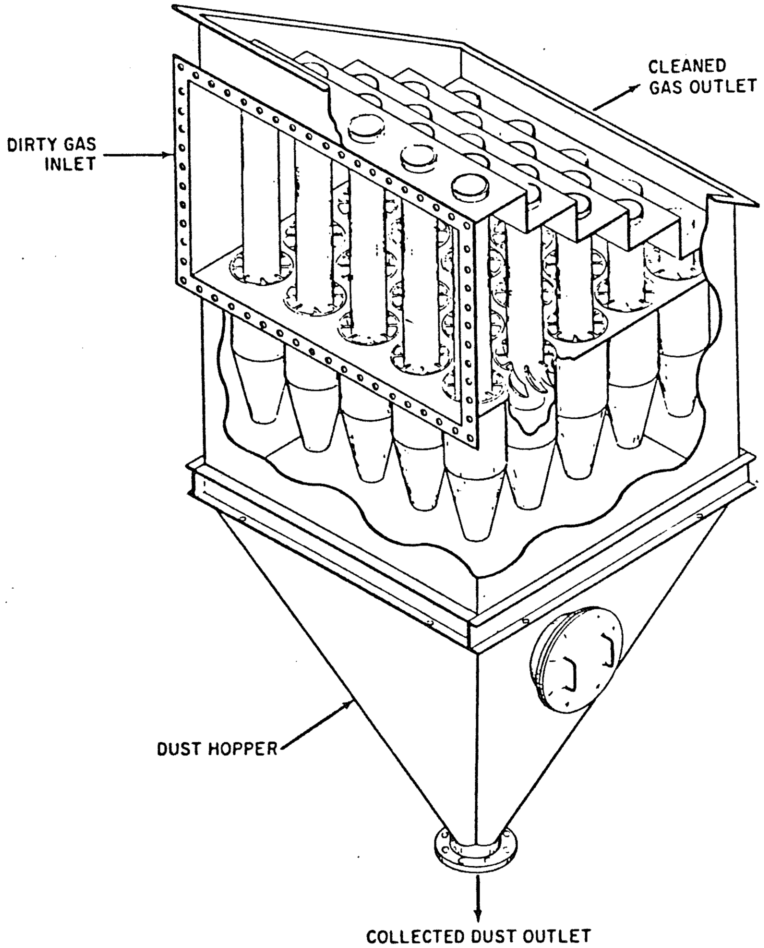


Figure B-4. Vane axial cyclone collector.

MECHANICAL COLLECTORS - (contd.)

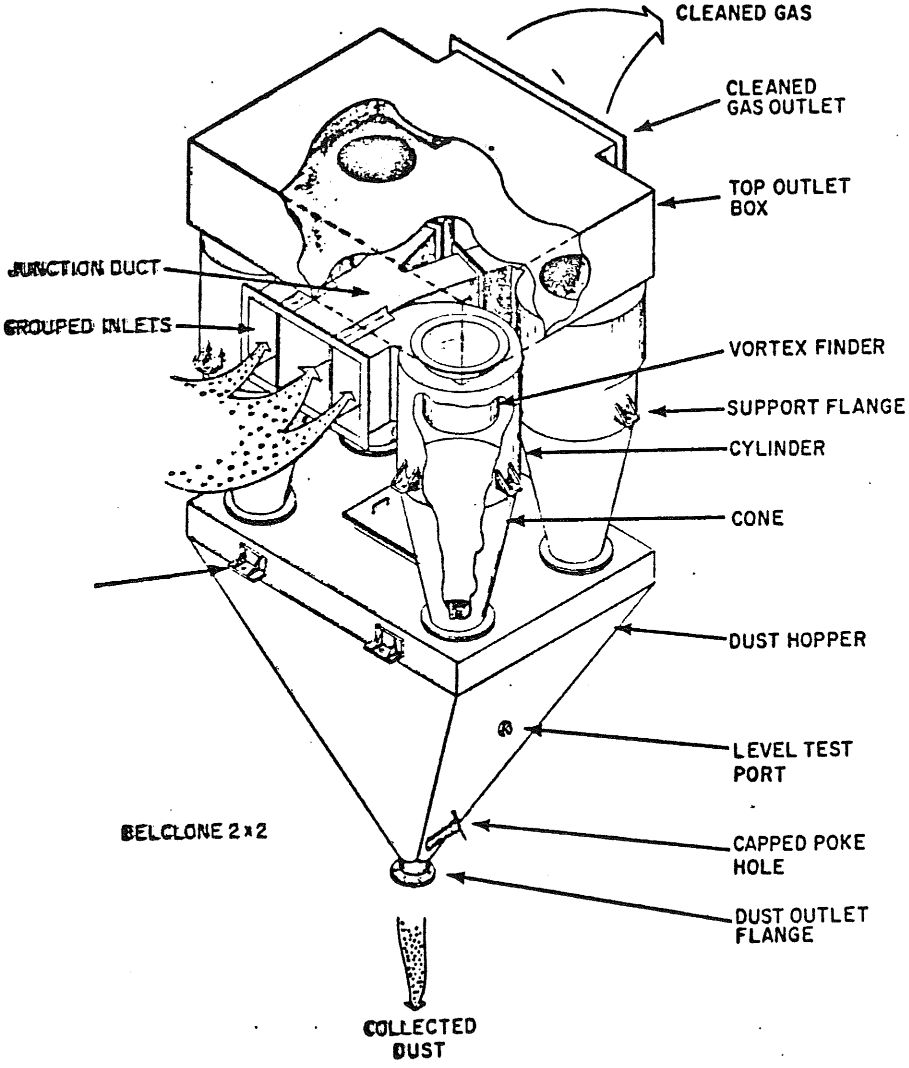


Figure B-5. Group of four involute cyclones.

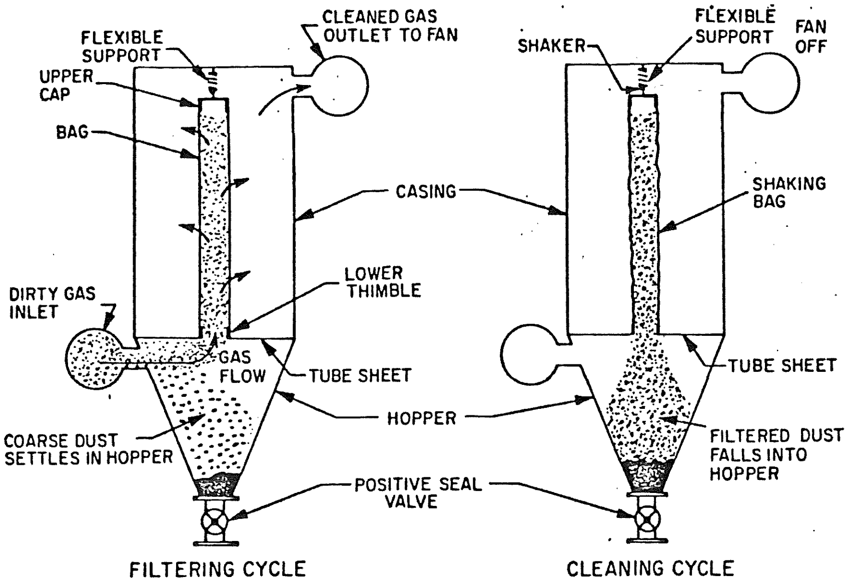


Figure B-6. Intermittent baghouse with manual or powered shaking.

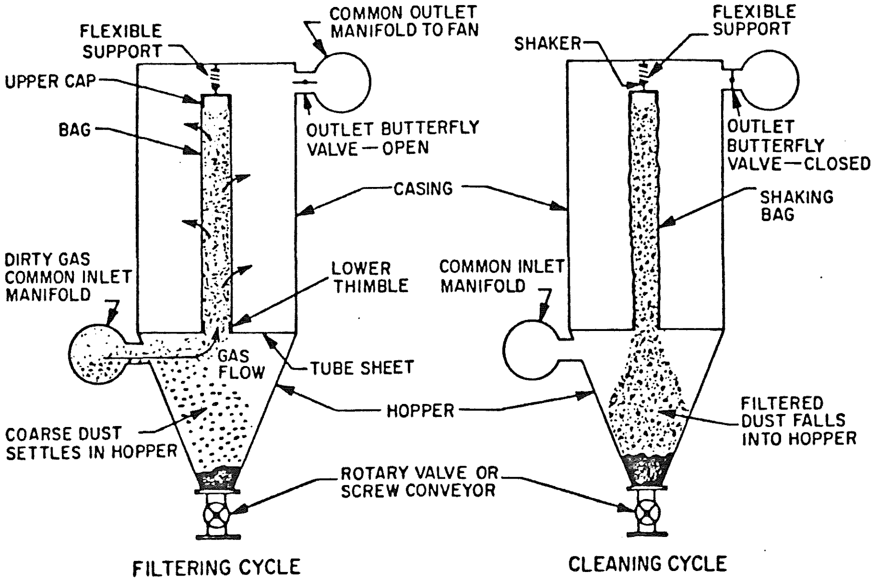


Figure B-7. Automatic conventional baghouse with mechanical shaking.

FABRIC FILTERS - (contd.)

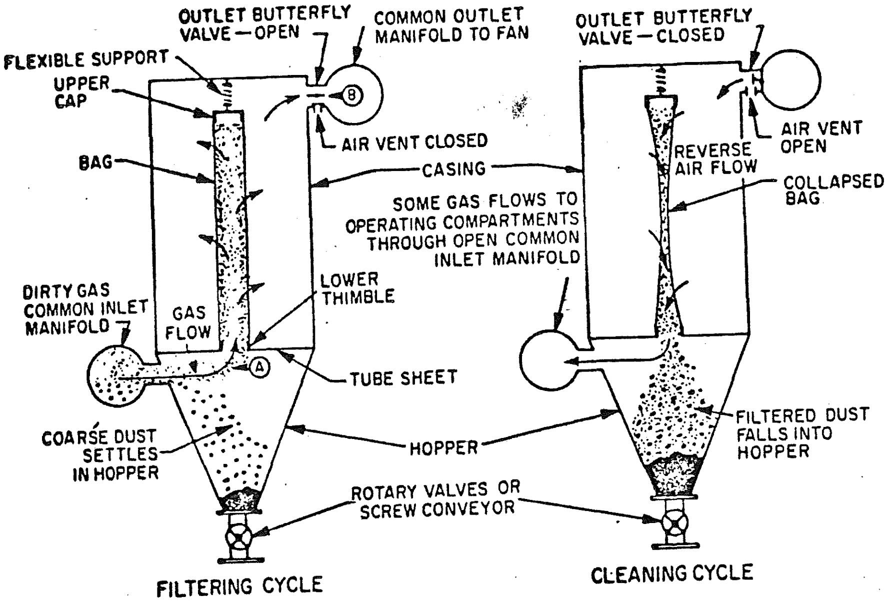


Figure B-8. Automatic baghouse with reverse airflow cleaning.

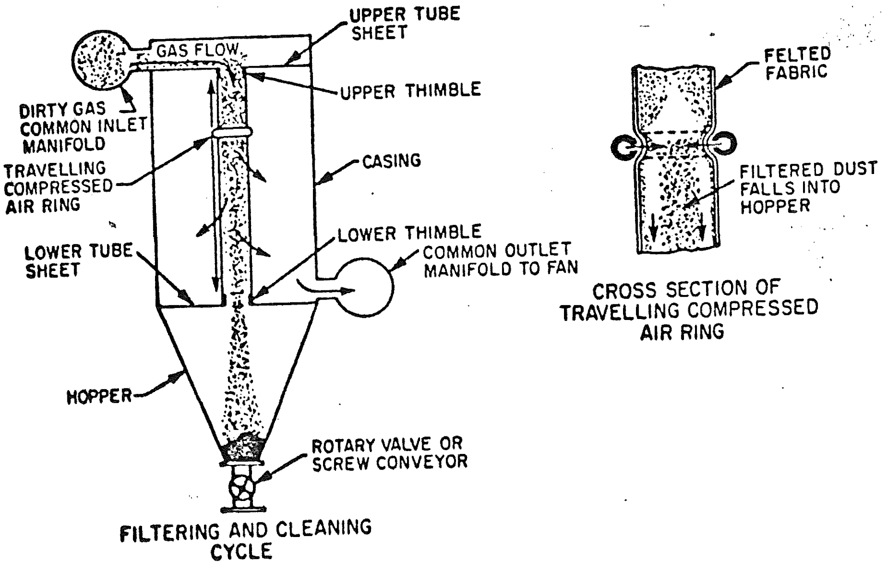


Figure B-9. Automatic baghouse with reverse ring jet cleaning.

FABRIC FILTERS - (contd.)

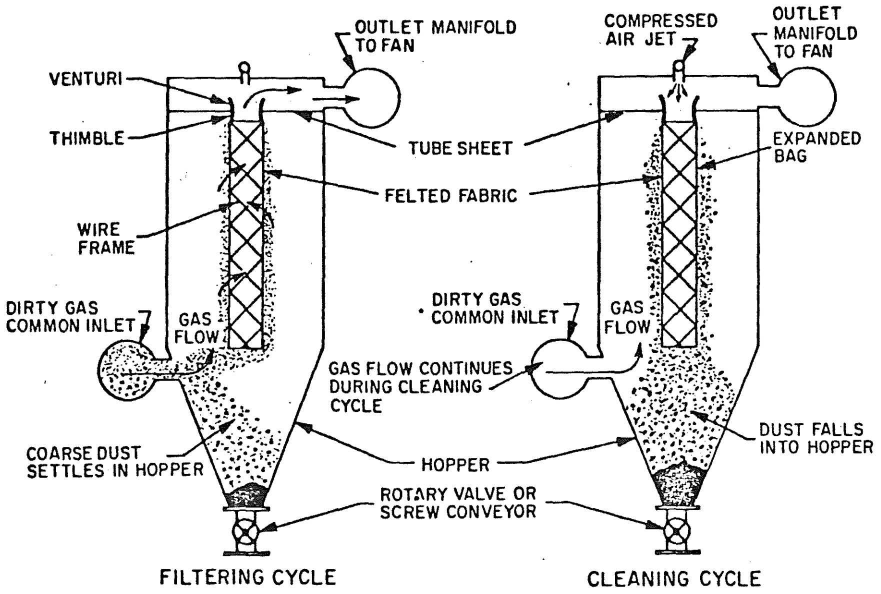


Figure B-10. Automatic baghouse with pulse jet cleaning.

WET SCRUBBERS

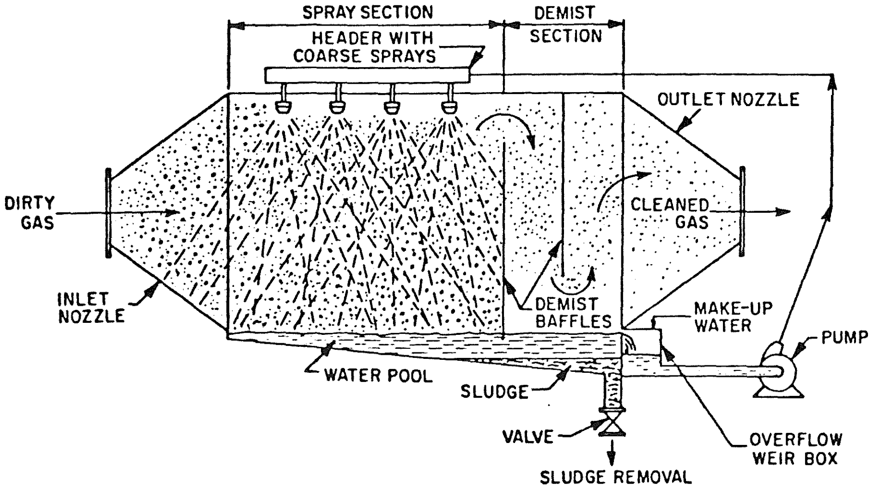


Figure B-11. Gravity settling chamber scrubber.

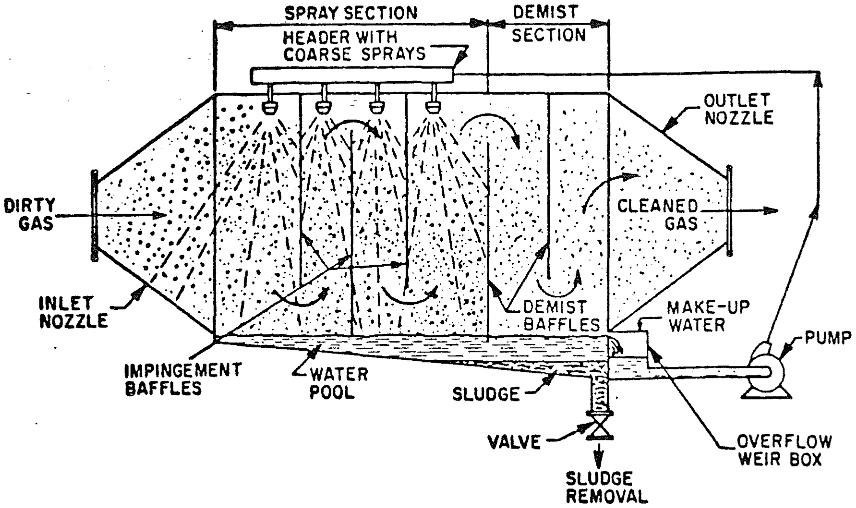


Figure B-12. Wetted impingement baffle scrubber.

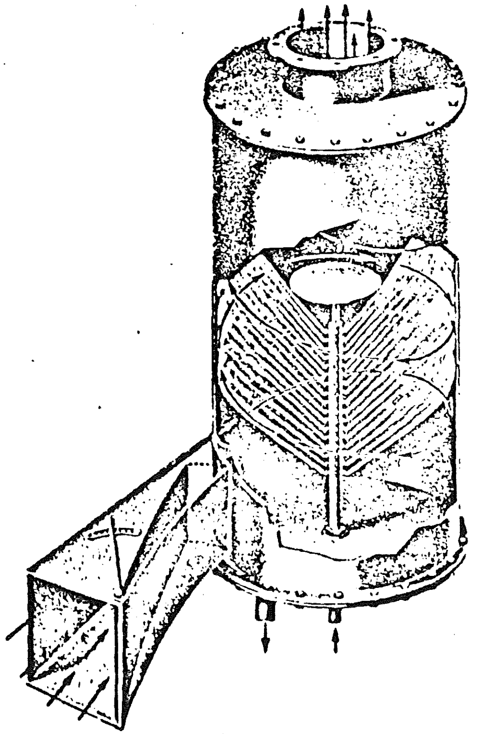


Figure B-13. Cyclone scrubber.

WET SCRUBBERS - (contd.)

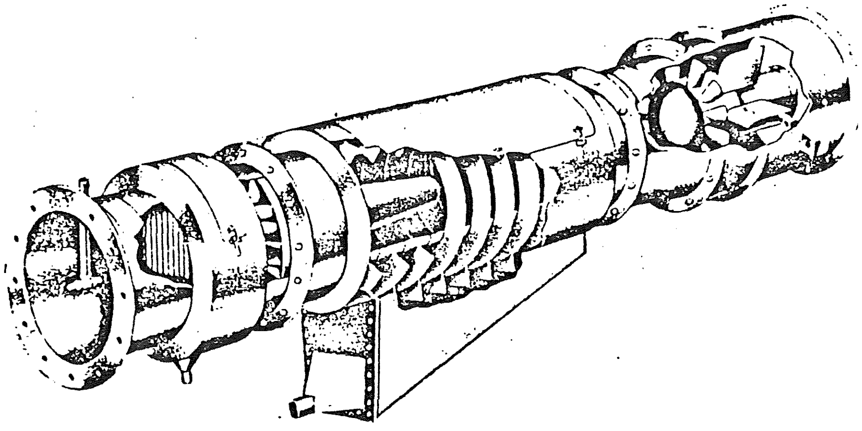


Figure B-14. Multiple-action scrubber.

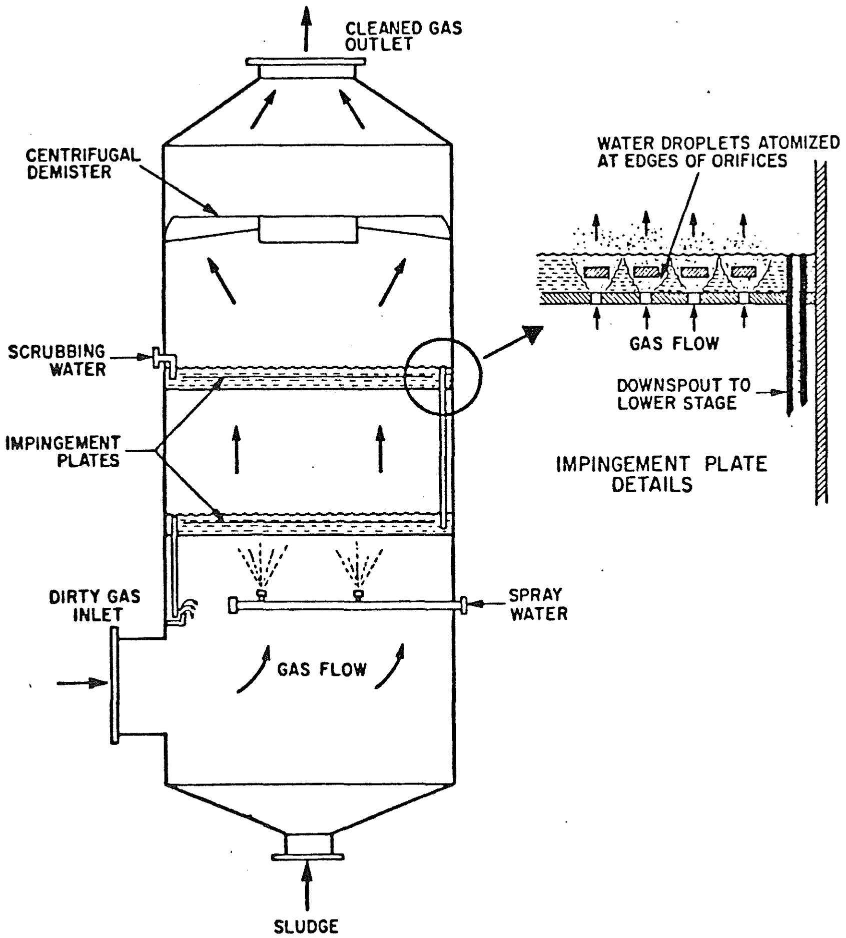


Figure B-15. Impingement plate scrubber.

WET SCRUBBERS - (contd.)

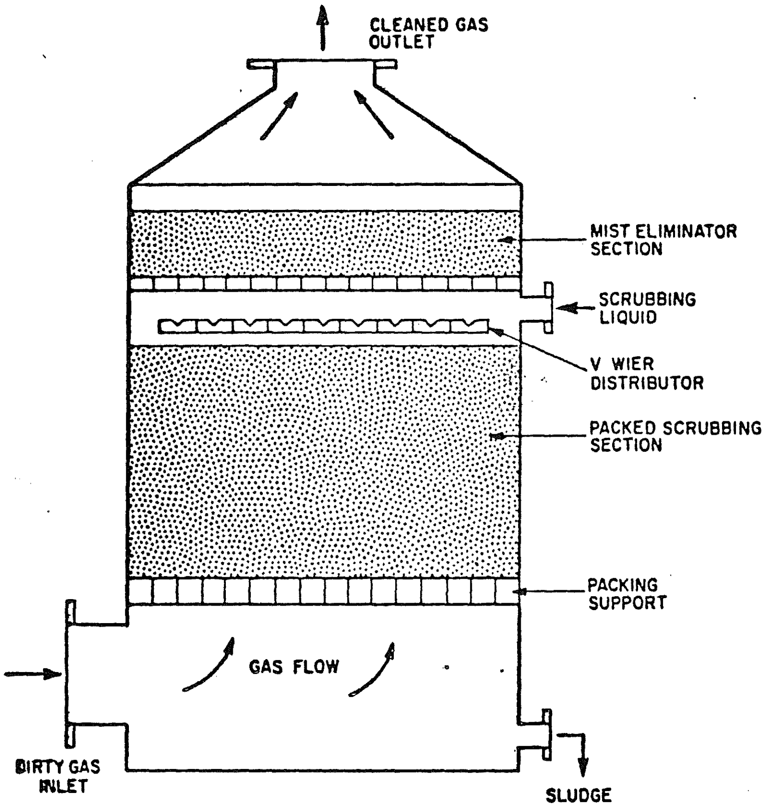


Figure B-16. Countercurrent flow scrubber.

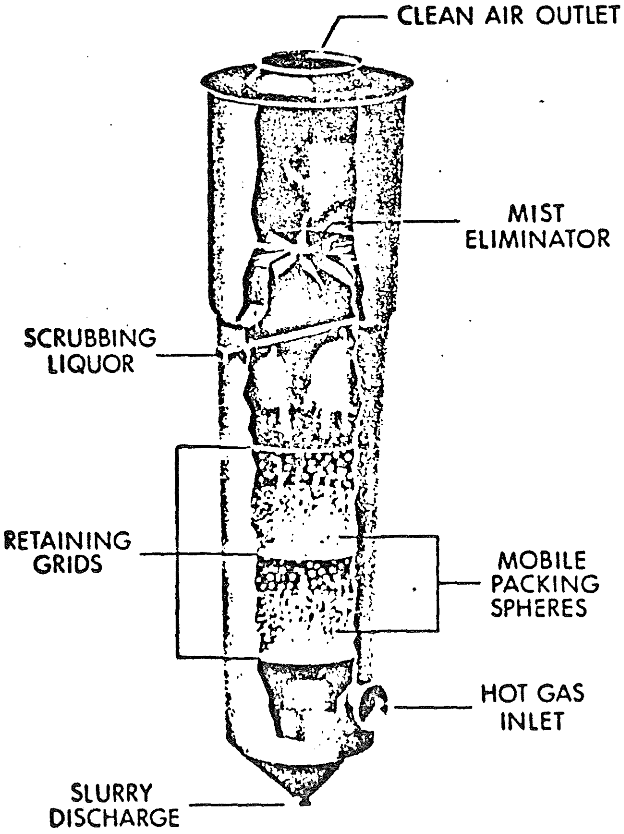


Figure B-17. Fluidized bed scrubber.

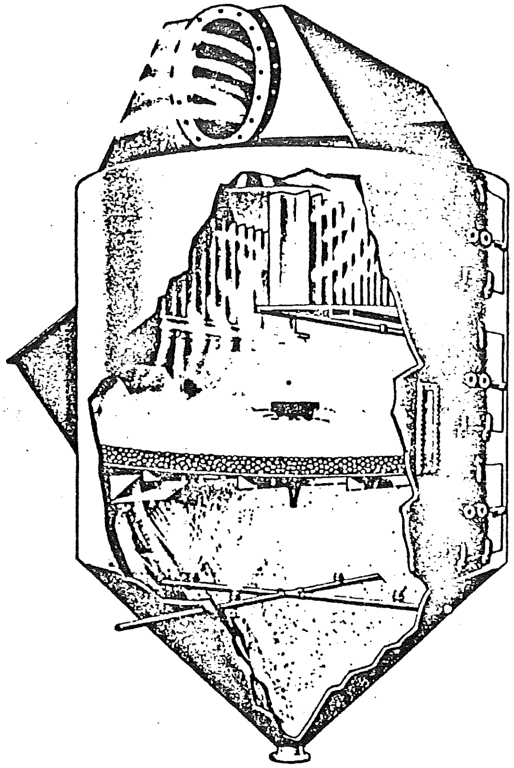


Figure B-18. Flooded bed scrubber.

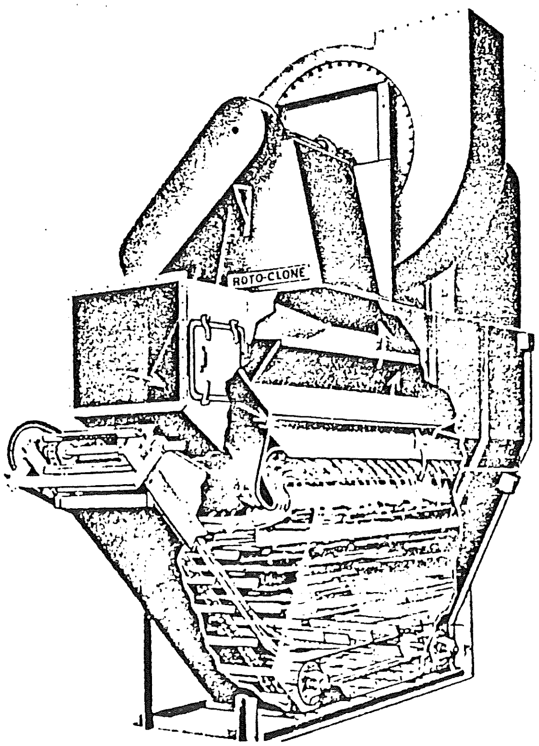


Figure B-19. Submerged-orifice scrubber.

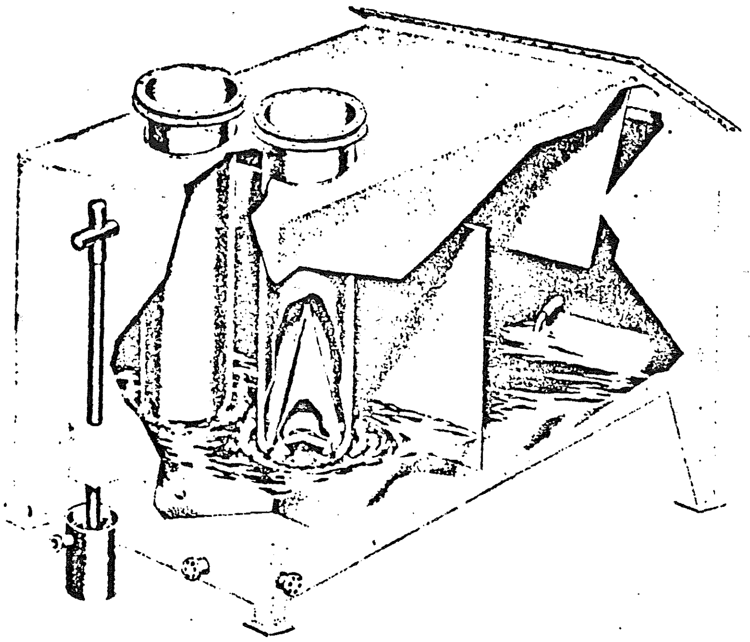


Figure B-20. Multiple submerged-orifice scrubber.

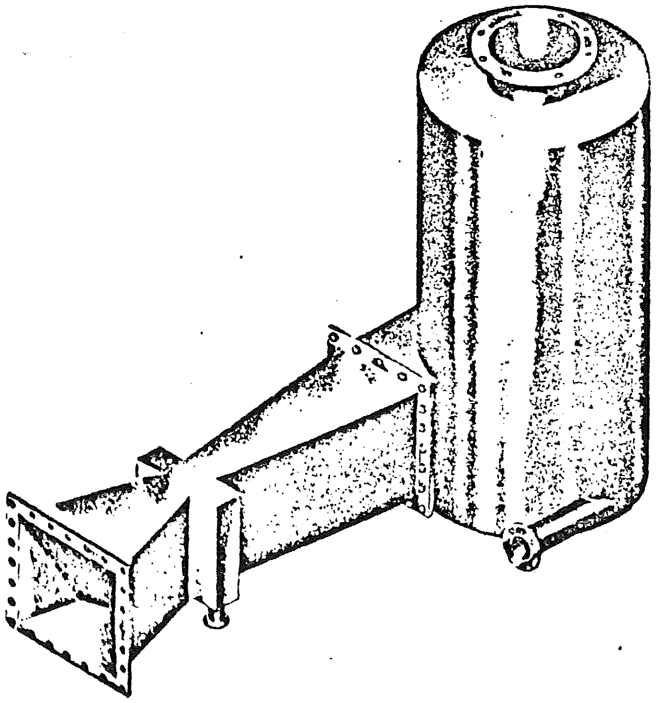


Figure B-21. Rectangular, horizontal gas flow venturi scrubber.

WET SCRUBBERS - (contd.)

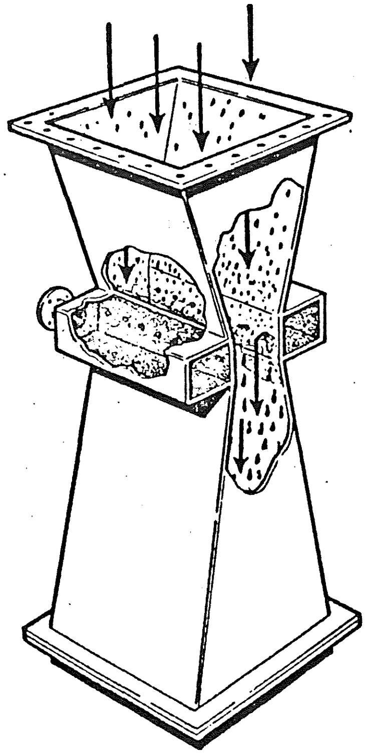


Figure B-22. Vertical downward gas flow venturi scrubber.

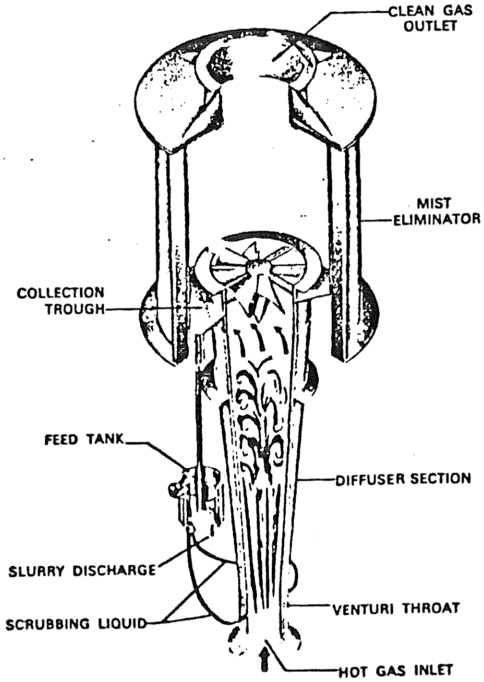


Figure B-23. Vertical upward gas flow venturi scrubber.

WET SCRUBBERS - (contd.)

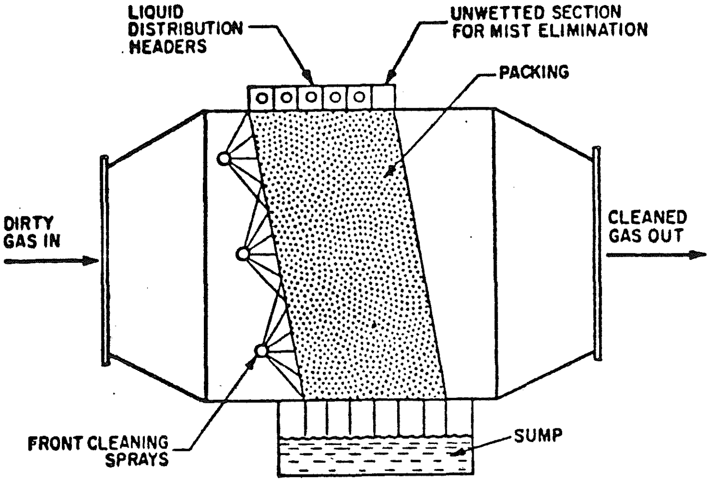


Figure B-24. Cross-flow packed scrubber.

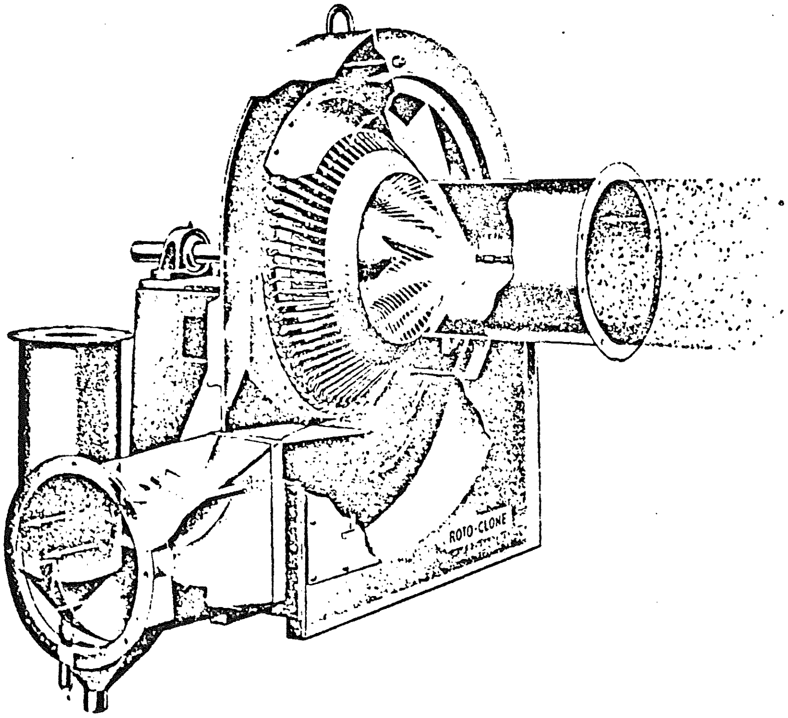


Figure B-25. Centrifugal fan-type wet scrubber.

WET SCRUBBERS - (contd.)

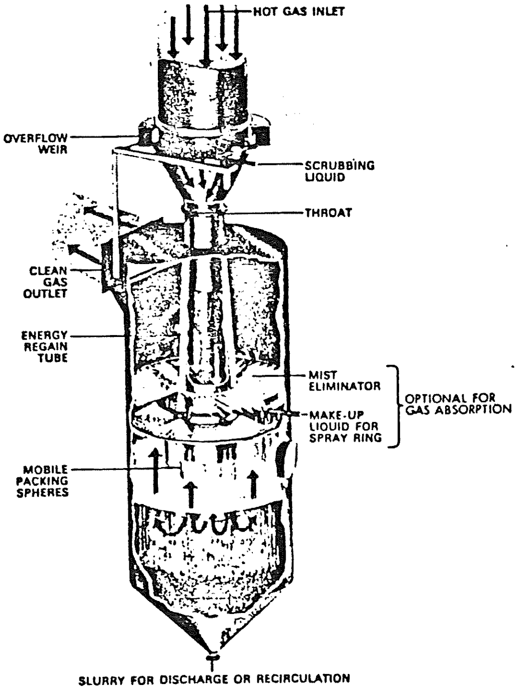
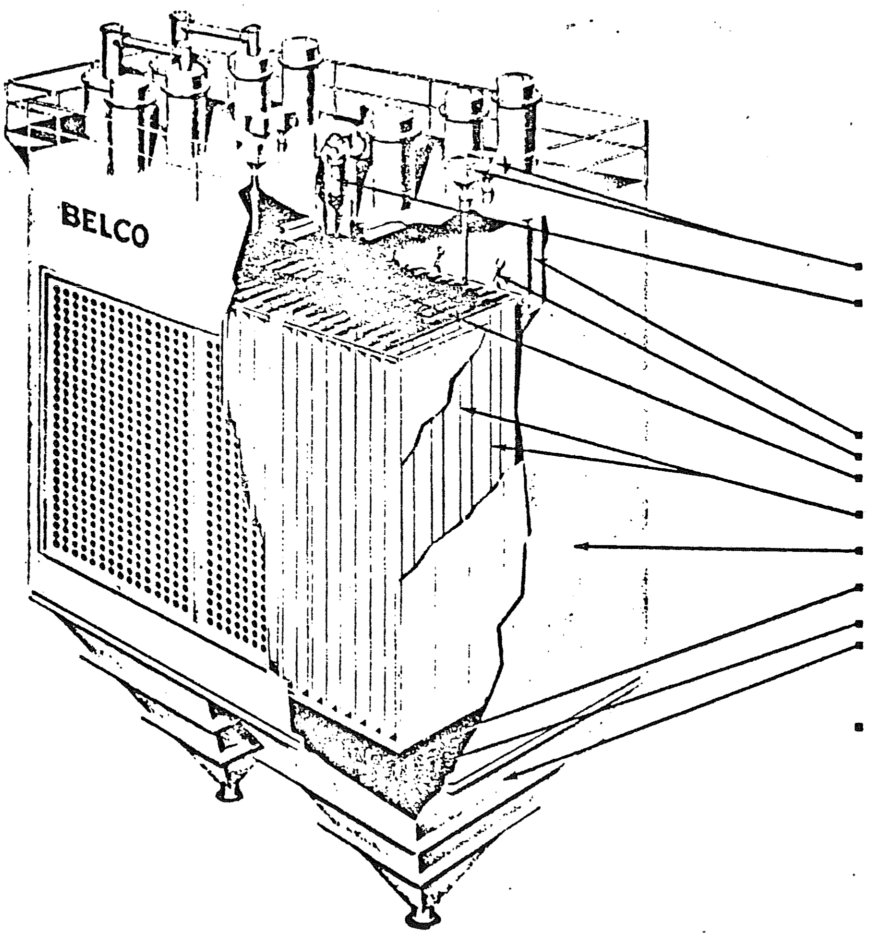


Figure B-26. Combination wet scrubber.



- All Vibrators and Rapping Devices are Roof Mounted for Easy Checking and Adjustment.
- Each High Voltage Support Insulator is Mounted in an Individual, Roof-Mounted, Insulator Housing providing many operational advantages:
  a — Insulators Far Removed from Gas Stream.
  b — Insulators can be Inspected, Cleaned, and Even Replaced from Roof without Entering Casing.
  c — Insulators Protected from Dust by Grounded Shroud.
  d — For Unusual Services Insulators can be Protected by Dried, Heated Air.
- Casing Roof of Shop-Fabricated, Integral-Beam Steel Construction.
- Collectrode Modules Suspended from Shop-Fabricated, Shock-Isolated Grids.
- Discharge Electrode Fields Suspended from Shop-Fabricated Grids, Are Held in Precise Position by Four High Voltage Support Insulators.
- Discharge Electrode Wires are Specially Fitted on Both Ends to Insure Good Electrical Contact, to Eliminate Burning, and to Extend Life.
- Casing Roof, Walls, and Hoppers Have Clean Exterior Design to Facilitate Application of Thermal Insulation.
- A Broken Wire can be Located, Removed, and Replaced Within Minutes of Entering Casing.
- Weights Cannot Fall Into Hoppers to Block or Break Hopper Valves.
- Pyramidal Hoppers are Equipped with Flush Access Doors, Poke Holes, Steep Valley Angles, and Large, Flanged Outlets. Where Service Requirements Warrant, Hoppers can be Equipped with Heating Coils, Vibrators, Hammers, and Level Indicators. Other Hopper Construction: Scraper Bottom, Trough Type, and Wet Bottom.
- Safety-Key-Interlocked, Quick-Opening, Doors Provide Easy Access Through Roof and Sides of Casing.

Figure B-27. Modern dry-plate-type electric precipitator.

APPENDIX C

Computer program for smelter
distribution plots.

```
        SMELTER DISTRIBUTION PLOT DATA - 24 CARDS FOR CU

 NUMBER   COMPANY  FURNACE  PMATTE   PSLAG   POTHER    PCUX
    5        4        2      97.86    1.44      .70    30.00
   20       15        3      90.00    2.02     7.98    30.00
   10        7       11      97.00     .86     2.14    30.20
    9        7        5      98.25    1.41      .34    33.00
   19       14        3      90.67    1.65     7.68    34.00
   17       12        2      88.64    1.08    10.28    36.00
   14        9        2      99.14     .67      .19    41.00
    6        6        2      99.23     .77        0    43.00
    8        7        4      89.96     .85     9.19    44.00
   11        7       12      96.34    1.57     2.09    44.20
   16       11       12      99.09     .91        0    46.00
   18       13        2      97.92    1.30      .78    47.00
   21       16        4      99.15     .85        0    47.00
   24       19        1      97.92     .92     1.16    47.00
   22       17        1      93.58     .78     5.64    49.00
   23       18        1      98.00     .49     1.51    49.00
    7        6        1      98.23    1.35      .42    50.50
   13        8        1      89.70    1.10     9.20    56.60
   15       10       12      96.08    2.31     1.61    63.00
    4        2        1      87.88    1.24    10.88    71.60
    3        1        0      81.16   15.04     3.80    75.80
    2        1        0      71.78   24.77     3.45    96.40
    1        1        0      69.98   26.16     3.86    97.00
   12        7       13      77.16    8.43    14.41    97.30

THE COEFFICIENTS FOR CURVE A -   .91632346E+01  .37434238E+00 -.81434005E-01

THE COEFFICIENTS FOR CURVE B -   .36151939E+00 -.26687145E+00  .62406159E-01

THE COEFFICIENTS FOR CURVE C -   .47524597E+00 -.10747093E+00  .19027846E-01

THE TEMP COEFFS A -   .91632346E+01  .37434238E+00 -.81434005E-01

THE TEMP COEFFS B -   .36151939E+00 -.26687145E+00  .62406159E-01

THE TEMP COEFFS C -   .47524597E+00 -.10747093E+00  .19027846E-01

THE FINAL COEFFS FOR CURVE A -   .91632346E+01  .37434238E+00 -.81434005E-01

THE FINAL COEFFS FOR CURVE B -   .36151939E+00 -.26687145E+00  .62406159E-01

THE FINAL COEFFS FOR CURVE C -   .47524597E+00 -.10747093E+00  .19027846E-01

THE ERRORS FOR CURVE A  -  R2 =   .80848719E+00  SE =   .40068040E+00

THE ERRORS FOR CURVE B  -  R2 =   .76978437E+00  SE =   .36677105E+00

THE ERRORS FOR CURVE C  -  R2 =   .11887739E+00  SE =   .41906678E+00
***PLOTPAC  ACCOUNTING RECORD FRAME:     1= 000001     10 WORDS  (ACCOUNTING)
***PLOTPAC           1.3FT    FRAME:     2= 000002   1324 WORDS  (CU        )    1 TIME(S)  [NORMAL ]    FILE: CALCOM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 NUMBER   COMPANY  FURNACE  PMATTE   PSLAG   POTHER    PCUX
    5        4        2      48.77   50.88      .35    30.00
   17       12        2      42.84   57.16        0    36.00
    6        6        2      45.31   54.69        0    43.00
   14        9        2      55.62   44.26      .12    41.00
   18       13        2      49.02   50.98        0    47.00
   21       16        4      43.16   39.75    17.09    47.00
   24       19        1      44.78   55.22        0    47.00
   22       17        1      38.50   51.00    10.50    49.00
   23       18        1      50.00   39.13    10.87    49.00
   15       10       12      13.73   81.70     4.57    63.00
    7        6        1      35.60   62.68     1.72    50.50
    4        2        1       3.00   94.30     2.70    71.60
    3        1        0       1.75   96.99     1.26    75.80
    2        1        0        .17   98.70     1.13    96.40
    1        1        0        .17   98.82     1.01    97.00

THE COEFFICIENTS FOR CURVE A -   .73965176E+01 -.14164702E+01  .51931538E-01

THE COEFFICIENTS FOR CURVE B -   .29134630E+01  .10175044E+01 -.53179875E-02

THE COEFFICIENTS FOR CURVE C -  -.30998062E+00  .39896584E+00 -.46613551E-01

THE TEMP COEFFS A -   .73965176E+01 -.14164702E+01  .51931538E-01

THE TEMP COEFFS B -   .29134630E+01  .10175044E+01 -.53179875E-02

THE TEMP COEFFS C -  -.30998062E+00  .39896584E+00 -.46613551E-01

THE FINAL COEFFS FOR CURVE A -   .73965176E+01 -.14164702E+01  .51931538E-01

THE FINAL COEFFS FOR CURVE B -   .29134630E+01  .10175044E+01 -.53179875E-02

THE FINAL COEFFS FOR CURVE C -  -.30998062E+00  .39896584E+00 -.46613551E-01

THE ERRORS FOR CURVE A  -  R2 =   .84679390E+00  SE =   .88990637E+00

THE ERRORS FOR CURVE B  -  R2 =   .77910345E+00  SE =   .11382106E+01

THE ERRORS FOR CURVE C  -  R2 =   .12618891E+00  SE =   .52606183E+00
***PLOTPAC           1.3FT    FRAME:     3= 000003   1175 WORDS  (FE        )    1 TIME(S)  [NORMAL ]    FILE: CALCOM
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```
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INUMBER  COMPANY  FURNACE  PMATTE  PSLAG   POTHER   PCUX
[illegible]   4        2      70.90    3.91   25.19   30.00
            6        2      78.43    2.65   18.92   43.00
            7        4      69.51    1.87   28.62   44.00
            6        1      38.38    2.16   59.46   50.50
   4        2        1       5.20     .08   94.72   71.60
   3        1        0      16.99    2.90   80.11   75.80
   2        1        0       2.12    7.04   90.84   96.40
   1        1        0       1.08    7.25   91.07   97.00

THE COEFFICIENTS FOR CURVE A -   .10563750E+02 -.22634433E+01  .11509001E+00

THE COEFFICIENTS FOR CURVE B -   .72421782E+00 -.31970914E+00  .41304861E-01

THE COEFFICIENTS FOR CURVE C -  -.12879676E+01  .25831525E+01 -.15639487E+00

THE TEMP COEFFS A -   .10563750E+02 -.22634433E+01  .11509001E+00

THE TEMP COEFFS B -   .72421782E+00 -.31970914E+00  .41304861E-01

THE TEMP COEFFS C -  -.12879676E+01  .25831525E+01 -.15639487E+00

THE FINAL COEFFS FOR CURVE A -   .10563750E+02 -.22634433E+01  .11509001E+00

THE FINAL COEFFS FOR CURVE B -   .72421782E+00 -.31970914E+00  .41304861E-01

THE FINAL COEFFS FOR CURVE C -  -.12879676E+01  .25831525E+01 -.15639487E+00

THE ERRORS FOR CURVE A  -  R2 =   .87685882E+00  SE =   .13842037E+01

THE ERRORS FOR CURVE B  -  R2 =   .89716845E+00  SE =   .95117592E-01

THE ERRORS FOR CURVE C  -  R2 =   .86041563E+00  SE =   .14373713E+01
***PLOTPAC          1.3FT    FRAME:     4= 000004    1220 WORDS  (S          )    1 TIME(S)  [NORMAL ]    FILE: CALCOM
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  10        7       11      97.13    2.87       0   30.20
   9        7        5      97.05    2.95       0   33.00
   6        6        2      41.49    1.00   57.51   43.00
   8        7        4      81.78    3.58   14.64   44.00
  11        7       12      20.22    8.07   71.71   44.20
   7        6        1      89.20   10.80       0   50.50
  12        7       13      20.57   79.43       0   97.30

THE COEFFICIENTS FOR CURVE A -   .11837776E+02 -.27205243E+01  .19041780E+00

THE COEFFICIENTS FOR CURVE B -   .46105679E+00 -.39729869E+00  .17674589E+00

THE COEFFICIENTS FOR CURVE C -  -.22988323E+01  .31178230E+01 -.36716369E+00

THE TEMP COEFFS A -   .11837776E+02 -.27205243E+01  .19041780E+00

THE TEMP COEFFS B -   .46105679E+00 -.39729869E+00  .17674589E+00

THE TEMP COEFFS C -  -.22988323E+01  .31178230E+01 -.36716369E+00

THE FINAL COEFFS FOR CURVE A -   .11837776E+02 -.27205243E+01  .19041780E+00

THE FINAL COEFFS FOR CURVE B -   .46105679E+00 -.39729869E+00  .17674589E+00

THE FINAL COEFFS FOR CURVE C -  -.22988323E+01  .31178230E+01 -.36716369E+00

THE ERRORS FOR CURVE A  -  R2 =   .43088657E+00  SE =   .32558498E+01

THE ERRORS FOR CURVE B  -  R2 =   .99403911E+00  SE =   .26838913E+00

THE ERRORS FOR CURVE C  -  R2 =   .24492607E+00  SE =   .32823785E+01
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NUMBER   COMPANY   FURNACE   PMATTE   PSLAG    POTHER    PCUX
  5         4         2       73.03   24.64     2.33     30.00
 13         8         2       38.30   11.00    50.50     43.00
  4         2         1        9.49    1.79    88.72     71.60
  3         1         0        9.36   54.23    36.21     75.80
  2         1         0        3.78   60.02    34.20     96.40
  1         1         0        3.84   75.50    20.66     97.00
THE COEFFICIENTS FOR CURVE A -   .94810011E+01 -.26702361E+01  .19473457E+00

THE COEFFICIENTS FOR CURVE B -   .36697177E+01 -.16604554E+01  .27232094E+00

THE COEFFICIENTS FOR CURVE C -  -.31507188E+01  .43306915E+01 -.46705551E+00

THE TEMP COEFFS A -   .94810011E+01 -.26702361E+01  .19473457E+00

THE TEMP COEFFS B -   .36697177E+01 -.16604554E+01  .27232094E+00

THE TEMP COEFFS C -  -.31507188E+01  .43306915E+01 -.46705551E+00

THE FINAL COEFFS FOR CURVE A -   .94810011E+01 -.26702361E+01  .19473457E+00

THE FINAL COEFFS FOR CURVE B -   .36697177E+01 -.16604554E+01  .27232094E+00

THE FINAL COEFFS FOR CURVE C -  -.31507188E+01  .43306915E+01 -.46705551E+00

THE ERRORS FOR CURVE A  -  R2 =   .98774781E+00  SE =   .39616537E+00

THE ERRORS FOR CURVE B  -  R2 =   .71913084E+00  SE =   .20234168E+01

THE ERRORS FOR CURVE C  -  R2 =   .66491553E+00  SE =   .21917845E+01
***PLOTPAC          D-3FT    FRAME:    6= 000006    1227 WORDS  (PB         )     1 TIME(S)  [NORMAL ]    FILE: CALCOM
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NUMBER   COMPANY   FURNACE   PMATTE   PSLAG    POTHER    PCUX
  5         4         2       41.61   52.64     5.75     30.00
 13         8         1       27.40   52.20    20.40     50.60
  4         2         1        3.25   10.13    86.62     71.60
  3         1         0        2.88   82.36    14.76     75.80
  2         1         0           0   77.39    22.61     96.40
  1         1         0           0   85.29    14.71     97.00
THE COEFFICIENTS FOR CURVE A -   .57056492E+01 -.14248721E+01  .87336066E-01

THE COEFFICIENTS FOR CURVE B -   .69018197E+01 -.15967153E+01  .22991650E+00

THE COEFFICIENTS FOR CURVE C -  -.26074689E+01  .30215874E+01 -.31725256E+00

THE TEMP COEFFS A -   .57056492E+01 -.14248721E+01  .87336066E-01

THE TEMP COEFFS B -   .69018197E+01 -.15967153E+01  .22991650E+00

THE TEMP COEFFS C -  -.26074689E+01  .30215874E+01 -.31725256E+00

THE FINAL COEFFS FOR CURVE A -   .57056492E+01 -.14248721E+01  .87336066E-01

THE FINAL COEFFS FOR CURVE B -   .69018197E+01 -.15967153E+01  .22991650E+00

THE FINAL COEFFS FOR CURVE C -  -.26074689E+01  .30215874E+01 -.31725256E+00

THE ERRORS FOR CURVE A  -  R2 =   .96478090E+00  SE =   .42807008E+00

THE ERRORS FOR CURVE B  -  R2 =   .37804658E+00  SE =   .28954993E+01

THE ERRORS FOR CURVE C  -  R2 =   .34084480E+00  SE =   .30839721E+01
***PLOTPAC          D-3FT    FRAME:    7= 000007    1183 WORDS  (ZN         )     1 TIME(S)  [NORMAL ]    FILE: CALCOM
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```
        SMELTER DISTRIBUTION PLOT DATA -  7 CARDS FOR SIO2
NUMBER   COMPANY  FURNACE  PMATTE  PSLAG   POTHER    PCUX
  5         4        2          0  100.00       0    30.00
 14         9        2       5.56   94.21     .23    41.00
  6         6        2          0  100.00       0    43.00
  7         6        1          0   82.23   17.77    50.51
  3         1        0          0   99.02     .98    75.80
  2         1        0          0   99.01     .99    96.40
  1         1        0          0   98.98    1.02    97.00
THE COEFFICIENTS FOR CURVE A -   .15355949E+00 -.44772069E-02 -.22951677E-02

THE COEFFICIENTS FOR CURVE B -   .10453107E+02 -.61808078E+00  .73023648E-01

THE COEFFICIENTS FOR CURVE C -  -.60666668E+00  .62255799E+00 -.70728480E-01

THE TEMP COEFFS A -   .15355949E+00 -.44772069E-02 -.22951677E-02

THE TEMP COEFFS B -   .10453107E+02 -.61808078E+00  .73023648E-01

THE TEMP COEFFS C -  -.60666668E+00  .62255799E+00 -.70728480E-01

THE FINAL COEFFS FOR CURVE A -   .15355949E+00 -.44772069E-02 -.22951677E-02

THE FINAL COEFFS FOR CURVE B -   .10453107E+02 -.61808078E+00  .73023648E-01

THE FINAL COEFFS FOR CURVE C -  -.60666668E+00  .62255799E+00 -.70728480E-01

THE ERRORS FOR CURVE A  -  R2 =   .11458323E+00  SE =   .24218390E+00

THE ERRORS FOR CURVE B  -  R2 =   .27918210E+00  SE =   .67331567E+00

THE ERRORS FOR CURVE C  -  R2 =   .22398454E+00  SE =   .70455040E+00
***PLOTPAC           1.3FT    FRAME:     8= 000010    1129 WORDS  (SIO2        )     1 TIME(S)  [NORMAL ]    FILE: CALCOM
```

```
        SMELTER DISTRIBUTION PLOT DATA -  4 CARDS FOR AL203
NUMBER   COMPANY  FURNACE  PMATTE  PSLAG   POTHER    PCUX
  5         4        2       3.95   96.05       0    30.00
 14         9        2       8.66   91.33     .01    41.00
  6         6        2          0  100.00       0    43.00
  7         6        1          0  100.00       0    50.50
THE COEFFICIENTS FOR CURVE A -  -.26237602E+00  .98577597E+00 -.30092753E+00

THE COEFFICIENTS FOR CURVE B -   .10263930E+02 -.98789898E+00  .30146360E+00

THE COEFFICIENTS FOR CURVE C -  -.15541450E-02  .21230106E-02 -.53607597E-03

THE TEMP COEFFS A -  -.26237602E+00  .98577597E+00 -.30092753E+00

THE TEMP COEFFS B -   .10263930E+02 -.98789898E+00  .30146360E+00

THE TEMP COEFFS C -  -.15541450E-02  .21230106E-02 -.53607597E-03

THE FINAL COEFFS FOR CURVE A -  -.26237602E+00  .98577597E+00 -.30092753E+00

THE FINAL COEFFS FOR CURVE B -   .10263930E+02 -.98789898E+00  .30146360E+00

THE FINAL COEFFS FOR CURVE C -  -.15541450E-02  .21230106E-02 -.53607597E-03

THE ERRORS FOR CURVE A  -  R2 =   .36503166E+00  SE =   .56819902E+00

THE ERRORS FOR CURVE B  -  R2 =   .36489185E+00  SE =   .56887720E+00

THE ERRORS FOR CURVE C  -  R2 =   .38677631E+00  SE =   .67817237E-03
***PLOTPAC           1.3FT    FRAME:     9= 000011     568 WORDS  (AL203       )     1 TIME(S)  [NORMAL ]    FILE: CALCOM
```

```
        SMELTER DISTRIBUTION PLOT DATA -  3 CARDS FOR MGO

NUMBER   COMPANY   FURNACE   PMATTE  PSLAG    POTHER    PCUX
  5         4         2          0   99.78      .22    30.00
  6         6         2          0  100.00       0     43.00
  7         6         1          0   91.53     8.47    50.50

THE COEFFICIENTS FOR CURVE A -             0              0              0

THE COEFFICIENTS FOR CURVE B -   .86750331E+01  .18621163E+01 -.55914947E+00

THE COEFFICIENTS FOR CURVE C -   .13249669E+01 -.18621163E+01  .55914947E+00

THE TEMP COEFFS A -   .28421709E-12 -.39790393E-12  .05923269E-13

THE TEMP COEFFS B -   .86750331E+01  .18621163E+01 -.55914947E+00

THE TEMP COEFFS C -   .13249669E+01 -.18621163E+01  .55914947E+00

THE FINAL COEFFS FOR CURVE A -   .94739031E-13 -.13263464E-12  .31974423E-13

THE FINAL COEFFS FOR CURVE B -   .86750331E+01  .18621163E+01 -.55914947E+00

THE FINAL COEFFS FOR CURVE C -   .13249669E+01 -.18621163E+01  .55914947E+00

THE ERRORS FOR CURVE A  -  R2 =                 0  SE =            0

THE ERRORS FOR CURVE B  -  R2 =   .10000000E+01  SE =            0

THE ERRORS FOR CURVE C  -  R2 =   .10000000E+01  SE =            0
***PLOTPAC           1.3FT    FRAME:    10= 000012     876 WORDS  (MGO        )    1 TIME(S)  [NORMAL ]    FILE: CALCOM
```

```
        SMELTER DISTRIBUTION PLOT DATA -  4 CARDS FOR CAO

NUMBER   COMPANY   FURNACE   PMATTE  PSLAG    POTHER    PCUX
  5         4         2       1.86   97.70      .44    30.00
 14         9         2       8.86   90.58      .56    41.00
  6         6         2          0  100.00       0     43.00
  7         6         1          0  100.00       0     50.50

THE COEFFICIENTS FOR CURVE A -  -.86676102E+00  .14794387E+01 -.39806089E+00

THE COEFFICIENTS FOR CURVE B -   .10833098E+02 -.15033373E+01  .40988918E+00

THE COEFFICIENTS FOR CURVE C -   .33663060E-01  .23898586E-01 -.11828286E-01

THE TEMP COEFFS A -  -.86676102E+00  .14794387E+01 -.39806089E+00

THE TEMP COEFFS B -   .10833098E+02 -.15033373E+01  .40988918E+00

THE TEMP COEFFS C -   .33663060E-01  .23898586E-01 -.11828286E-01

THE FINAL COEFFS FOR CURVE A -  -.86676102E+00  .14794387E+01 -.39806089E+00

THE FINAL COEFFS FOR CURVE B -   .10833098E+02 -.15033373E+01  .40988918E+00

THE FINAL COEFFS FOR CURVE C -   .33663060E-01  .23898586E-01 -.11828286E-01

THE ERRORS FOR CURVE A  -  R2 =   .34189383E+00  SE =   .59186762E+00

THE ERRORS FOR CURVE B  -  R2 =   .33983774E+00  SE =   .62771787E+00

THE ERRORS FOR CURVE C  -  R2 =   .50029525E+00  SE =   .35850253E-01
***PLOTPAC           1.3FT    FRAME:    11= 000013     574 WORDS  (CAO        )    1 TIME(S)  [NORMAL ]    FILE: CALCOM
```

```
0100 PROGRAM SMELTER(INPUT,OUTPUT)
0110 DIMENSION INUM(25),ICOM(25),IFUR(25),XMAT(25),XSLG(25),XOTH(25),
0120+XCUX(25),WEIGHT(25),COEF(3),XNV(150),NAME(3),ICAP(9)
0130 DO 500 JJ = 1,10
0140C
0150C READ THE HEADER CARD
0160C
0170 READ 305,IELM,ICARDS
0180 305 FORMAT(30X,A5,18X,I3)
0190C
0200C READ THE NUMBER OF CARDS TO FOLLOW
0210C
0220 READ 306,(INUM(I),ICOM(I),IFUR(I),XMAT(I),XSLG(I),XOTH(I),XCUX(I),
0230+I=1,ICARDS)
0240 306 FORMAT(3I3,3F7.2,F6.1)
0250C
0260C PRINT OUT THE DATA READ IN TO CHECK IT
0270C
0280 PRINT 307,ICARDS,IELM
0290 307 FORMAT(/,9X,≠SMELTER DISTRIBUTION PLOT DATA - ≠I2≠ CARDS FOR ≠,
0300+A5,/)
0310 PRINT 308
0320 308 FORMAT(≠ INUMBER  COMPANY  FURNACE  PMATTE  PSLAG   POTHER   PCUX≠)
0330 PRINT 309,(INUM(I),ICOM(I),IFUR(I),XMAT(I),XSLG(I),XOTH(I),
0340+XCUX(I),I=1,ICARDS)
0350 309 FORMAT(I5,2I9,F11.2,3F8.2)
0360C
0370C SET UP THE WEIGHT ARRAYS, SCALE THE X AND Y VALUES FOR PLOTTING
0380C
0390 DO 10 I=1,ICARDS
0400 WEIGHT(I) = 1.0
0410 XMAT(I) = XMAT(I)*0.1
0420 XSLG(I) = XSLG(I)*0.1
0430 XOTH(I) = XOTH(I)*0.1
0440 XCUX(I) = XCUX(I)*0.1 - 2.0
0450 10 CONTINUE
0460C
0470C COMPUTE THE LEAST SQUARES CURVE FOR A
0480C
0490 CALL LSQORPY(XCUX,XMAT,WEIGHT,ICARDS,-2,NDEGF,COEF,XNV)
0500 IFIT = 1HA
0510 A1=COEF(1)
0520 B1=COEF(2)
0530 C1=COEF(3)
0540 PRINT 310,IFIT,A1,B1,C1
0550C
0560C COMPUTE THE LEAST SQUARES CURVE FOR B
```

```
0570C
0580 CALL LSQORPY(XCUX,XSLG,WEIGHT,ICARDS,-2,NDEGF,COEF,XNV)
0590 IFIT = 1HB
0600 A2 = COEF(1)
0610 B2 = COEF(2)
0620 C2 = COEF(3)
0630 PRINT 310, IFIT,A2,B2,C2
0640C
0650C COMPUTE THE LEAST SQUARES CURVE FOR C
0660C
0670 CALL LSQORPY(XCUX,XOTH,WEIGHT,ICARDS,-2,NDEGF,COEF,XNV)
0680 IFIT = 1HC
0690 A3 = COEF(1)
0700 B3 = COEF(2)
0710 C3 = COEF(3)
0720 PRINT 310,IFIT,A3,B3,C3
0730 310 FORMAT(/,≠ THE COEFFICIENTS FOR CURVE ≠A1≠ - ≠3E15.8)
0740C
0750C COMPUTE THE TEMPERARY COEFFS FOR CURVE A
0760C
0770 IFIT = 1HA
0780 TA1 = 10.0-(A2 + A3)
0790 TB1 = -(B2 + B3)
0800 TC1 = -(C2 + C3)
0810 PRINT 312,IFIT,TA1,TB1,TC1
0820C
0830C COMPUTE THE TEMPERARY COEFFS FOR CURVE B
0840C
0850 IFIT = 1HB
0860 TA2 = 10.0 - (A1 + A3)
0870 TB2 = -(B1 + B3)
0880 TC2 = -(C1 + C3)
0890 PRINT 312,IFIT,TA2,TB2,TC2
0900C
0910C COMPUTE THE TEMPERARY COEFFS FOR CURVE C
0920C
0930 IFIT = 1HC
0940 TA3 = 10.0 - (A1 + A2)
0950 TB3 = -(B1 + B2)
0960 TC3 = -(C1 + C2)
0970 PRINT 312, IFIT,TA3,TB3,TC3
0980 312 FORMAT(/,≠ THE TEMP COEFFS ≠A1≠ - ≠3E15.8)
0990C
1000C COMPUTE THE FINAL COEFFS FOR CURVE A
1010C
1020 IFIT = 1HA
1030 FA1 = (2.0*A1+TA1)/3.0
```

```
1040 FB1 = (2.0*B1+TB1)/3.0
1050 FC1 = (2.0*C1+TC1)/3.0
1060 PRINT 314,IFIT,FA1,FB1,FC1
1070C
1080C COMPUTE THE FINAL COEFFS FOR CURVE B
1090C
1100 IFIT = 1HB
1110 FA2 = (2.0*A2 + TA2)/3.0
1120 FB2 = (2.0*B2 + TB2)/3.0
1130 FC2 = (2.0*C2 + TC2)/3.0
1140 PRINT 314, IFIT,FA2,FB2,FC2
1150C
1160C COMPUTE THE FINAL COEFFS FOR CURVE C
1170C
1180 IFIT = 1HC
1190 FA3 = (2.0*A3 + TA3)/3.0
1200 FB3 = (2.0*B3 + TB3)/3.0
1210 FC3 = (2.0*C3 + TC3)/3.0
1220 PRINT 314,IFIT,FA3,FB3,FC3
1230 314 FORMAT(/,≠ THE FINAL COEFFS FOR CURVE ≠A1≠ - ≠3E15.8)
1240C
1250C COMPUTE THE ERRORS FOR CURVES A,B,C
1260C
1262 IFIT = 1HA
1270 CALL ERRORS(XCUX,XMAT,ICARDS,2,FA1,FB1,FC1,SE1,R1)
1272 PRINT 316,IFIT,R1,SE1
1274 IFIT = 1HB
1280 CALL ERRORS(XCUX,XSLG,ICARDS,2,FA2,FB2,FC2,SE2,R2)
1282 PRINT 316,IFIT,R2,SE2
1284 IFIT = 1HC
1290 CALL ERRORS(XCUX,XOTH,ICARDS,2,FA3,FB3,FC3,SE3,R3)
1292 PRINT 316,IFIT,R3,SE3
1294 316 FORMAT(/,≠ THE ERRORS FOR CURVE ≠A1≠  -  R2 = ≠E15.8≠  SE = ≠
1295+E15.8)
1300C
1310C IT IS TIME TO DRAW THE PLOT
1320C
1330 CALL PLOTS(6HEQUIPT,0.01,14.0)
1340 CALL PLOTS(1,1,20HCUNISTU SMELTER PLOT,20.0,IELM)
 (1340) - WARNING  ARGUMENT 2 IS DIFFERENT TYPE IN CALL PLOTS
 (1340) - WARNING  SUBPROGRAM PLOTS APPEARED WITH FEWER ARGUMENTS
                   ------USED IN LINE!  1330.
1350 CALL PLOT(2.0,2.5,-3)
1360C
1370C GO UP THE LEFTHAND SIDE WITH THE AXIS
1380C
1390 Y = 0.0
1400 ENCODE(29,355,NAME)IELM
1410 355 FORMAT(24HPERCENT DISTRIBUTION OF ,A5)
```

```
1420 DO 25 I1 = 0,90,10
1430 IF(I1.EQ.0) GO TO 26
1440 CALL PLOT(-0.1,Y,1)
1450 CALL NUMBER(-.40,Y-.07,.14,I1,0.0,2HI2)
1460 IF(I1.EQ.40) CALL SYMBOL(-.5,Y-.74,.14,NAME,90.,29)
1470 CALL PLOT(-0.1,Y,1)
1480 26 CALL PLOT(0.0,Y,2)
1490 Y = Y + 1.0
1500 CALL PLOT(0.0,Y,1)
1510 25 CONTINUE
1520C
1530C GO ACROSS THE TOP AND WRITE A HEADER
1540C
1550 X = 0.0
1560 CALL SYMBOL(X+2.50,Y+.30,.14,25HSMELTER DISTRIBUTION PLOT,0.0,25)
1570 CALL PLOT(0.0,Y,1)
1580 DO 30 I1=1,8
1590 IF(I1.EQ.1) GO TO 32
1600 CALL PLOT(X,Y+.10,1)
1610 32 CALL PLOT(X,Y,2)
1620 X = X + 1.0
1630 CALL PLOT(X,Y,1)
1640 30 CONTINUE
1650C
1660C GO BACK DOWN THE RIGHTHAND SIDE
1670C
1680 DO 35 I1=1,10
1690 IF(I1.EQ.1) GO TO 37
1700 CALL PLOT(X+.10,Y,1)
1710 37 CALL PLOT(X,Y,2)
1720 Y = Y - 1.0
1730 CALL PLOT(X,Y,1)
1740 35 CONTINUE
1750C
1760C GO BACK ALONG THE BOTTOM BACKWARD
1770C
1780 I2 = 100
1790 DO 40 I1 = 1,8
1800 IF(I1.EQ.1) GO TO 42
1810 I2 = I2 - 10
1820 CALL PLOT(X,-0.10,1)
1830 CALL NUMBER(X-.12,-0.30,.14,I2,0.0,2HI2)
1840 IF(I1.EQ.7) CALL SYMBOL(X+.62,-.55,.14,23HPERCENT CU IN THE MATTE,0.,23)
1850 CALL PLOT(X,-0.10,1)
1860 42 CALL PLOT(X,0.0,2)
1870 X = X - 1.0
1880 CALL PLOT(X,0.0,1)
1890 40 CONTINUE
```

```
1900C
1910C PUT ON THE SYMBOLED DATA POINTS FOR CURVES A,B,C
1920C
1930 DO 50 I=1,ICARDS
1940 X = XCUX(I)
1950 Y1 = XMAT(I)
1960 Y2 = XSLG(I)
1970 Y3 = XOTH(I)
1980 IS = IFUR(I)
1990 CALL SYMBOL(X,Y1,.12,IS,0.0,-1)
2000 CALL SYMBOL(X,Y2,.12,IS,0.0,-1)
2010 CALL SYMBOL(X,Y3,.12,IS,0.0,-1)
2020 50 CONTINUE
2030C
2040C DRAW THE LINES FOR CURVES A,B,C
2050C
2060 CALL DRAWL(1,FA1,FB1,FC1,SE1)
2070 CALL DRAWL(2,FA2,FB2,FC2,SE2)
2080 CALL DRAWL(3,FA3,FB3,FC3,SE3)
2090C
2100C PUT THE LEGOND ON THE SIDE
2110C
2111 CALL SYMBOL(10.25,9.5,.14,6HLEGEND,0.0,6)
2112 CALL SYMBOL(9.00,9.00,.14,12HFURNACE TYPE,0.0,12)
2120 X1 = 9.54 $ X2 = 10.0 $ Y1 = 8.5 $ Y2 = Y1 + 0.07
2130 CALL SYMBOL(X1,Y1+.07,.12,0,0.0,-1)
2140 CALL SYMBOL(X2,Y1,.14,12H= CONTINUOUS,0.0,12)
2150 Y1 = Y1 - .5
2160 CALL SYMBOL(X1,Y1+.07,.12,1,0.0,-1)
2170 CALL SYMBOL(X2,Y1,.14,7H= FLASH,0.0,7)
2180 Y1 = Y1 - .5
2190 CALL SYMBOL(X1,Y1+.07,.12,2,0.0,-1)
2200 CALL SYMBOL(X2,Y1,.14,13H= REVERBATORY,0.0,13)
2210 Y1 = Y1 - .5
2220 CALL SYMBOL(X1,Y1+.07,.12,3,0.0,-1)
2230 CALL SYMBOL(X2,Y1,.14,10H= ELECTRIC,0.0,10)
2240 Y1 = Y1 - .5
2250 CALL SYMBOL(X1,Y1+.07,.12,4,0.0,-1)
2260 CALL SYMBOL(X2,Y1,.14,14H= OXYGEN FLASH,0.0,14)
2270 Y1 = Y1 - .5
2280 CALL SYMBOL(X1,Y1+.07,.12,5,0.0,-1)
2290 CALL SYMBOL(X2,Y1,.14,19H= HOT CHARGE REVERB,0.0,19)
2300 Y1 = Y1 - .5
2310 CALL SYMBOL(X1,Y1+.07,.12,11,0.0,-1)
2320 CALL SYMBOL(X2,Y1,.14,19H= WET CHARGE REVERB,0.0,19)
2330 Y1 = Y1 - .5
2340 CALL SYMBOL(X1,Y1+.07,.12,12,0.0,-1)
2350 CALL SYMBOL(X2,Y1,.14,15H= BLAST FURNACE,0.0,15)
```

```
2360 Y1 = Y1 - .5
2370 CALL SYMBOL(X1,Y1+.07,.12,13,0.0,-1)
2380 CALL SYMBOL(X2,Y1,.14,6H= TBRC,0.0,6)
2390 Y1 = Y1 - .5
2391 CALL SYMBOL(9.00,Y1,.14,10HCURVE TYPE,0.0,10)
2392 Y1 = Y1 - .5
2393 CALL PLOT(X1-.75,Y1+.07,3) $ CALL PLOT(X1,Y1+.07,2)
2394 CALL SYMBOL(X2,Y1,.14,7H= MATTE,0.0,7)
2395 Y1 = Y1 -.5
2396 CALL PLOT(X1-.75,Y1+.07,3) $ CALL PLOT(X1-.50,Y1+.07,2)
2397 CALL PLOT(X1-.40,Y1+.07,3) $ CALL PLOT(X1-.35,Y1+.07,2)
2398 CALL PLOT(X1-.25,Y1+.07,3) $ CALL PLOT(X1,Y1+.07,2)
2399 CALL SYMBOL(X2,Y1,.14,6H= SLAG,0.0,6)
2400 Y1 = Y1-.5
2401 CALL PLOT(X1-.75,Y1+.07,3) $ CALL PLOT(X1-.70,Y1+.07,2)
2402 CALL PLOT(X1-.60,Y1+.07,3) $ CALL PLOT(X1-.55,Y1+.07,2)
2403 CALL PLOT(X1-.45,Y1+.07,3) $ CALL PLOT(X1-.40,Y1+.07,2)
2404 CALL PLOT(X1-.30,Y1+.07,3) $ CALL PLOT(X1-.25,Y1+.07,2)
2405 CALL PLOT(X1-.15,Y1+.07,3) $ CALL PLOT(X1-.10,Y1+.07,2)
2406 CALL PLOT(X1,Y1+.07,3)
2407 CALL PLOT(X1+.05,Y1+.07,2)
2408 CALL SYMBOL(X2,Y1,.14,7H= OTHER,0.0,7)
2409 Y1 = Y1 - .5
2410 CALL NUMBER(X1-.18,Y1,.14,ICARDS,0.0,2HI2)
2411 CALL SYMBOL(X2,Y1,.14,18H= SMELTERS PLOTTED,0.0,18)
2412 Y1 = Y1 - .5
2413 CALL SYMBOL(X2-.48,Y1,.14,25H2   = DEGREE OF THE THREE,0.0,25)
2414 CALL SYMBOL(X2,Y1-.2,.14,23H  POLYNOMIAL FITS WHERE,0.0,23)
2415 CALL SYMBOL(X2,Y1-.4,.14,22H  Y = A + B*X + C*X**2,0.0,22)
2416 Y1 = Y1 -.9
2417 CALL SYMBOL(X2-.60,Y1,.14,31HR2   = REGRESSION COEFF SQUARED,0.0,31)
2418 Y1 = Y1 - .5
2419 CALL SYMBOL(X2-.60,Y1,.14,21HSE   = STANDARD ERROR,0.0,21)
2450C
2460C PRINT COEFFS AND ERRORS AT THE BOTTOM OF THE PLOT
2470C
2480 L=6HMATTE
2490 ENCODE(86,410,ICAP)L,FA1,FB1,FC1,R1,SE1
2500 410 FORMAT(A6≠CURVE, COEFFS  A =≠F9.4≠  B =≠F9.4≠  C =≠F9.4
2510+≠  R2 =≠F6.4≠  SE =≠F7.4)
2520 CALL SYMBOL(0.0,-1.0,.14,ICAP,0.0,86)
2530 L = 6HSLAG
2540 ENCODE(86,410,ICAP)L,FA2,FB2,FC2,R2,SE2
2550 CALL SYMBOL(0.0,-1.5,.14,ICAP,0.0,86)
2560 L = 6HOTHER
2570 ENCODE(86,410,ICAP)L,FA3,FB3,FC3,R3,SE3
2580 CALL SYMBOL(0.0,-2.0,.14,ICAP,0.0,86)
2590C
```

```
2600C THE SMELTER PLOT IS ENDED
2610C
2620 CALL PLOTS(-1)
 (2620) - WARNING  SUBPROGRAM PLOTS APPEARED WITH MORE ARGUMENTS
                   ------USED IN LINE:  1340.  1330.
2630 500 CONTINUE
2640 STOP 100
2650 END

2660 SUBROUTINE ERRORS(X,Y,NE,NF,A,B,C,SE,R2)
2670 DIMENSION X(NE),Y(NE)
2680C
2690C CALCULATE THE AVERAGE Y
2700C
2710 AVY = 0.0
2720 DO 10 I=1,NE
2730 AVY = AVY+Y(I)
2740 10 CONTINUE
2750 AVY = AVY/NE
2760C
2770C CALCULATE R2 AND THE STANDARD ERROR
2780C
2790 S1=S2=S3=0.0
2800 DO 20 I = 1,NE
2810 YF = A+B*X(I)+C*X(I)**2
2820 S1 = S1+(YF-AVY)**2
2830 S2 = S2+(Y(I)-AVY)**2
2840 S3 = S3+(Y(I)-YF)**2
2850 20 CONTINUE
2860 R2 = S1/S2
2870 SE = 0.0
2880 S4 = NE - (NF + 1)
2890 IF(S4.GT.0.0) SE = SQRT(S3/S4)
2900 RETURN
2910 END

2920 SUBROUTINE DRAWL(NPLOT,A,B,C,SE)
2930 I1 = 180 + (NPLOT-1)*10
2949 I2 = 380 + (NPLOT-1)*10
2950 I3 = 580 + (NPLOT-1)*10
2960 L = 1HA
2970 IF(NPLOT.EQ.2) L = 1HB
2980 IF(NPLOT.EQ.3) L = 1HC
3010 X = .0  $  ITAG = 0 $ II = 0
3020 DO 60 I=1,800
3030 X = X + 0.01 $ II =  II + 1
3040 Y = (C*X+B)*X+A
3050 IF(Y.LT.0.0.OR.Y.GT.10.0) GO TO 60
3051 IF(ITAG.EQ.0) CALL PLOT(X,Y,3)
3052 ITAG = 1
```

```
3053 GO TO ( 280,285,290) NPLOT
3054 280 CALL PLOT(X,Y,2) $ GO TO 295
3055 285 IF(II.LE.25) THEN
3056       CALL PLOT(X,Y,2)
3057       ELSE
3058          IF(II.LE.35) THEN
3059            CALL PLOT(X,Y,3)
3060            ELSE
3061               IF(II.LE.40)THEN
3062                 CALL PLOT(X,Y,2)
3063                 ELSE
3064                    IF(II.LT.50) THEN
3065                      CALL PLOT(X,Y,3)
3066                      ELSE
3067                         CALL PLOT(X,Y,3)
3068                         II=0
3069                    ENDIF
3070               ENDIF
3071          ENDIF
3072     ENDIF
3073     GO TO 295
3074 290 IF(II.LE.5) THEN
3075       CALL PLOT(X,Y,2)
3076       ELSE
3077          IF(II.LT.15) THEN
3078            CALL PLOT(X,Y,3)
3079            ELSE
3080            CALL PLOT(X,Y,3)
3081            II=0
3082          ENDIF
3083     ENDIF
3084 295 CONTINUE
3060 CALL PLOT(X,Y,1)
3090 IF(I.EQ.I1.OR.I.EQ.I2.OR.I.EQ.I3) THEN
3092 YY=Y+SE $ IF(YY.GT.10.0) YY=10.0
3100 CALL PLOT(X,YY,2)
3110 CALL PLOT(X-.05,YY,1)
3120 CALL PLOT(X+.05,YY,1)
3130 CALL PLOT(X,Y,3)
3135  YY=Y-SE $ IF(YY.LT.0.0) YY = 0.0
3140 CALL PLOT(X,YY,2)
3150 CALL PLOT(X-.05,YY,1)
3160 CALL PLOT(X+.05,YY,1)
3170 CALL PLOT(X,Y,3)
3190 ENDIF
3200 60 CONTINUE
3202 CALL WHERE(XP,YP,DUMMY)  $  CALL PLOT(XP,YP,3)
3210 RETURN
3220 END
```

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