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Production of Blister Copper by Electric Furnace Smelting of Dead-Burned Copper Sulfide Concentrates

UNITED STATES DEPARTMENT OF THE INTERIOR

Report of Investigations 8131

Production of Blister Copper by Electric Furnace Smelting of Dead-Burned Copper Sulfide Concentrates

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PRODUCTION OF BLISTER COPPER BY ELECTRIC FURNACE SMELTING OF DEAD-BURNED COPPER SULFIDE CONCENTRATES

by

D. L. Paulson,[1] R. B. Worthington,[1] and W. L. Hunter[2]

ABSTRACT

The Bureau of Mines is investigating pyrometallurgical and chemical methods for extracting copper from sulfide ores under conditions that provide for control of gaseous and solid emissions. Under this program, a new electrothermic reduction process was developed to produce blister copper after collecting essentially all of the sulfur by a single-unit operation.

The process feasibility was indicated by a laboratory-scale investigation that demonstrated chalcopyrite concentrate can be roasted to less than 1 percent sulfur and that the resulting calcine can be reduced with carbon to blister copper by a cyclic process in a single arc-smelting furnace.

Pilot-scale tests in an 800-kV-A electric-arc furnace demonstrated that 98.4 pct of the contained copper can be continuously recovered as metal by operating at bath temperatures between 1,350° and 1,375° C. Smelting rates of approximately 225 lb of blend ft^2 of bath surface/hr resulted in power consumptions of approximately 520 kW-hr/ton of blend. The blister copper that was produced contained less than 0.1 pct iron while the precious metals reporting to the copper product and gangue material elimination were similar to those in a conventional smelting operation.

INTRODUCTION

Historically, copper has been recovered from its ores principally by pyrometallurgical methods. The copper smelting pit developed into the blast furnace process which, in turn, yielded to reverberatory furnace smelting. The products of the reverberatory furnace are slag, matte, and stack gases. The matte, a mixture of iron and copper sulfides, is oxidized by air or oxygen to blister copper which, in turn, is deoxidized in an anode furnace for subsequent electrolytic refining.

The relatively recent increase in awareness of the general public and public officials of the total effect of undesirable industrial byproducts on

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[1] Metallurgist.

[2] Supervisory metallurgist.

our environment has resulted in the establishment of air and water quality standards by various governmental agencies. Many of these standards appear to be impossible to conform to under present operating procedures. To promote industrial compliance with the environmental standards, the Bureau of Mines is continually working to develop new methods for extracting metals (4, 10)[3] to reduce and control emissions from existing facilities (7) and find uses for process byproducts (1-2).

Several approaches have been suggested for investigation to improve air quality control during copper extraction (3). These have included improved systems for conventional smelters, modification of smelting methods to produce higher SO_2 levels in offgases to allow easier recovery as sulfur or sulfuric acid, and the chemical extraction of copper by leaching or chlorination.

This report describes a process that was developed at the Bureau of Mines to recover blister copper by the electric furnace smelting of dead-roasted chalcopyrite concentrates. This pyrometallurgical approach aims toward solving the sulfur emission problem by removing the sulfur in a single-unit operation.

Figure 1 shows a flow diagram of the proposed process. The unique feature of the process is the cyclic selective carbon reduction in a submerged arc

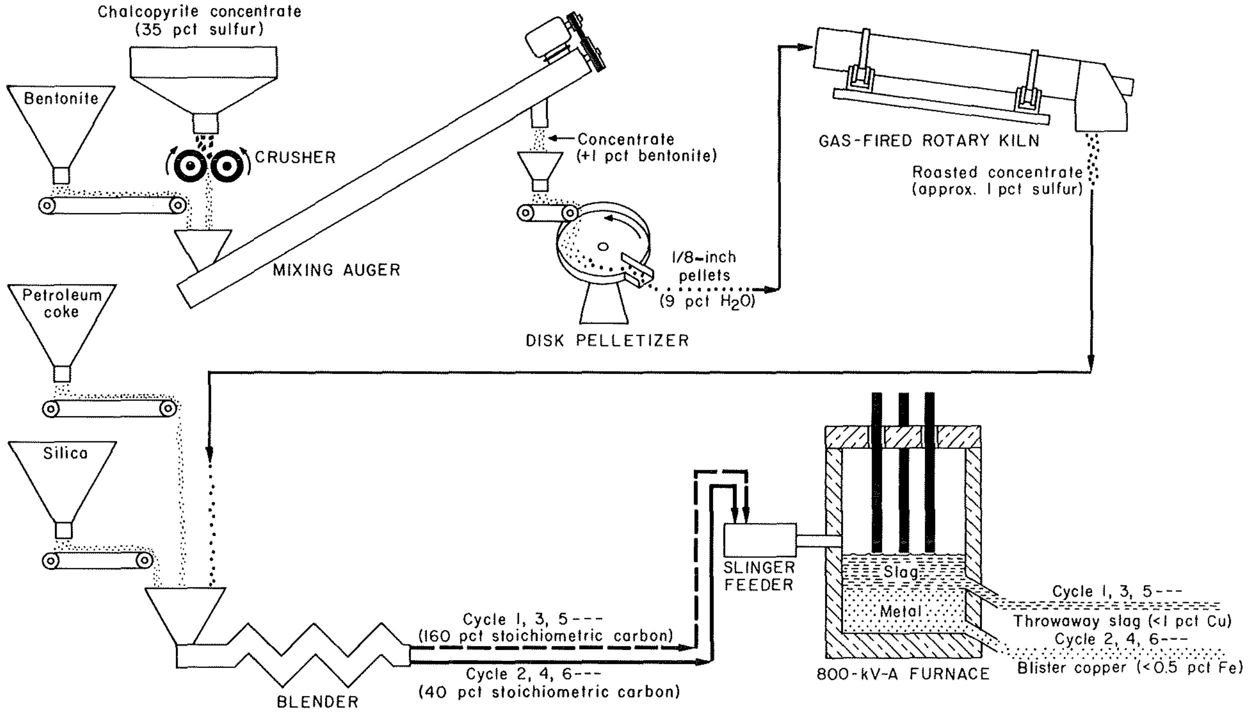

FIGURE 1. - Process flow diagram.

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[3] Underlined numbers in parentheses refer to items in the list of references preceding the appendix.

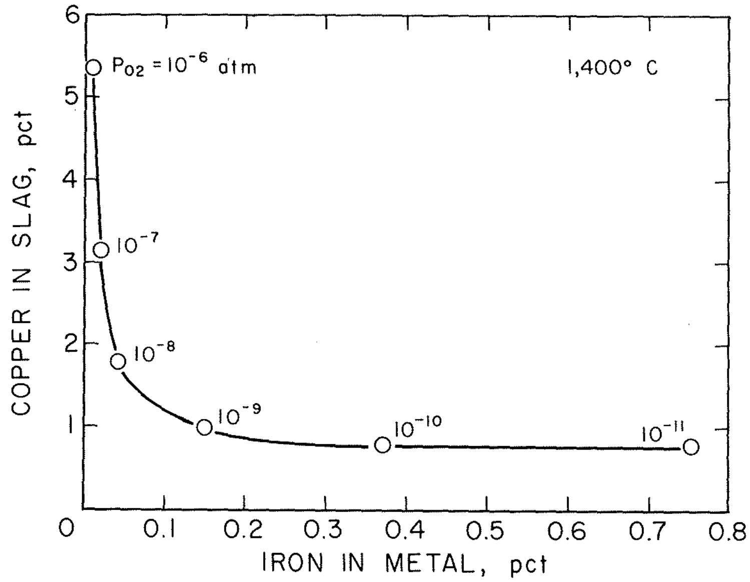

FIGURE 2. - The effect of oxygen partial pressure on copper and iron distribution.

furnace. One-half of the cyclic smelting process is based on the current practice of producing black copper by the carbothermic reduction of dead-roasted chalcopyrite concentrates (5). Due to the effect of oxygen partial pressures on the copper and iron distributions (8) shown in figure 2, the current practice was extended to a cyclic process. The copper content of the slag was minimized by adding excess carbon to the furnace charge prior to tapping slag and, alternately, the iron content of the copper product was lowered by establishing a carbon deficiency in the system prior to tapping copper.

Each step of the process was initially established on a laboratory scale in a 50-kV-A electric-arc furnace. After the laboratory-scale work was completed, the entire process was scaled up to ultimately produce tonnage lots of blister copper by smelting dead-roasted chalcopyrite in a 800-kV-A electric-arc furnace.

MATERIALS

Two chalcopyrite concentrates were used during this investigation. The early laboratory-scale work was done on concentrate from the Anaconda Company's operation in Yerrington, Nev., and the bulk of the work was done on a concentrate that was received from American Smelting and Refining Company's smelter in Tacoma, Wash. Moisture contents of the as-received materials were 5.2 and 7.8 pct, respectively. Chemical and particle analyses of the dried concentrates are shown in table 1. Green petroleum coke and foundry coke were used as reducing agents. The flux additions consisted of silica sand and, occasionally, pebble lime. Chemical analyses of the reductants and fluxes are shown in table 2.

TABLE 1. - Chemical analyses of chalcopyrite concentrates

Component	Concentration	
	ASARCO concentrate	Anaconda concentrate
Cu..................pct..	30.5	29.2
Fe..................pct..	27.1	27.3
S...................pct..	35.1	31.4
CaO.................pct..	0.17	-
SiO_2................pct..	2.07	4.47
Al_2O_3...............pct..	0.58	1.32
MgO.................pct..	-	<0.1
Bi..................pct..	0.67	-
Pb..................pct..	0.17	-
Zn..................pct..	2.48	-
Ag...............oz/ton..	27.1	-
Au...............oz/ton..	.06	-
Particle size, Tyler mesh	Distribution, pct	
	ASARCO concentrate	Anaconda concentrate
Plus 35..................	0	20.6
Minus 35 plus 100.........	7.6	17.8
Minus 100 plus 325........	52.8	20.5
Minus 325	39.6	41.1
	100	100

TABLE 2. - Chemical analyses of reductants and fluxes

Component	Concentration, pct	
REDUCTANTS		
	Green petroleum coke	Foundry coke
Carbon (fixed on dry basis)..............	90	90.8
Sulfur.................................	1.35	.4
Volatiles..............................	9	1.24
Ash....................................	.25	7.46
Moisture...............................	6	.51
FLUXES		
	Pebble lime	Silica sand
CaO....................................	97.8	-
Fe.....................................	.13	0.01
MgO....................................	.32	0.05
SiO_2..................................	.34	99.80
Al_2O_3.................................	.08	0.11
S......................................	.03	-
LOI[1]	1.30	-

[1] Loss on ignition.

ROASTING CHALCOPYRITE CONCENTRATES

Dead roasting of copper sulfide concentrate can eliminate essentially all of the contained sulfur in one-process step and greatly simplify sulfur dioxide recovery and air pollution control. Because dead roasting is not being practiced commercially with the exception of a small tonnage processed by the Montanwerke Brixlegg GmbH in the Inn Valley of Tyrol, Austria (5), it was not possible to purchase tonnage quantities of dead-roasted calcine for the electric-smelting tests. It was decided to expedite the smelting tests by calcining the concentrates in the roasting equipment that was available at the Bureau of Mines.

Equipment and Procedure for Preparing Concentrates

The raw concentrates were put through an 8-inch-diameter roll mill that was set at less than one-fourth of an inch to break up the wet agglomerates. The rolled concentrates were blended with 1 pct bentonite binder additions in a 6-inch-inside-diameter by 22-foot long grain auger. The blends were pelletized on a 39-inch disk pelletizer.

Equipment and Procedure for Dead-Roasting Concentrates

The pelletized concentrates were roasted in a propane gas-fired rotary kiln that was 15 feet long and 14.5 inches inside diameter. Three passes through the kiln were required to remove essentially all of the sulfur. The kiln temperatures were increased for consecutive passes through the kiln as shown in figure 3.

Results of Pelletizing and Roasting Concentrates

Pelletizing the concentrates at about 800 lb/hr produced pellets that were less than one-quarter of an inch diameter and contained from 9 to 10 pct moisture. After three passes through the kiln at consecutively higher temperatures, the calcine contained from 1.0 to 1.7 pct sulfur. Copper ferrites were undoubtedly formed during the final roasting stages (6, 9). However, even high levels of ferrites are not detrimental to electrothermic reduction processes.

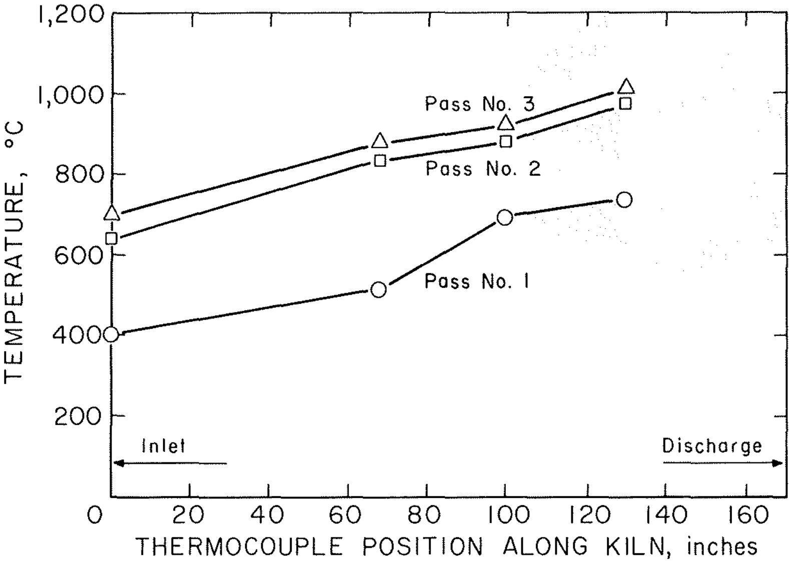

FIGURE 3. - Temperature profiles for pass No. 1 to pass No. 3 in the 14-inch rotary kiln.

CARBOTHERMIC REDUCTION OF DEAD-ROASTED CONCENTRATES

Smelting Tests in a 50-kV-A Electric-Arc Furnace

Preliminary smelting tests were conducted in a 50-kV-A electric-arc furnace. Single-charge smelting tests were designed to establish the process chemistry. Subsequently, six-step (three-cycle) tests were made to determine the feasibility of a cyclic smelting operation.

Materials Used for Smelting Tests in the 50-kV-A Electric-Arc Furnace

Dead-roasted Anaconda concentrate (calcine) was used exclusively for the laboratory smelting tests. The calcine contained the following principal components, in percent:

Cu	35.9	CaO	0.08
Fe	32.1	Al_2O_3	1.47
SiO_2	6.08	S	1.03

Petroleum coke was used as a reductant in all of the smelting tests in the 50-kV-A electric-arc furnace along with silica as a flux. In one test, pebble lime was added to the furnace feed.

Equipment and Operating Procedures for Smelting Tests in the 50-kV-A Electric-Arc Furnace

All of the laboratory-scale smelting tests were conducted in a 50-kV-A electric-arc furnace. The single-phase current for the furnace was supplied by a 50-kV-A welding transformer which was operated from a portable control panel near the furnace. Manually operated, rack- and pinion-mounted electrode clamps controlled the position of the 2-inch graphite electrodes. The furnace shell was approximately 17 inches in outside diameter and 17 inches high and was lined with high-purity, castable alumina refractory to obtain a working crucible about 12 inches in inside diameter. The crucible was covered with a rammed, high-alumina roof supported by a water-cooled ring. A taphole was provided at hearth level to remove metallic copper, and the slag taphole was located about 2 inches above the metal taphole. The furnace is shown in figure 4.

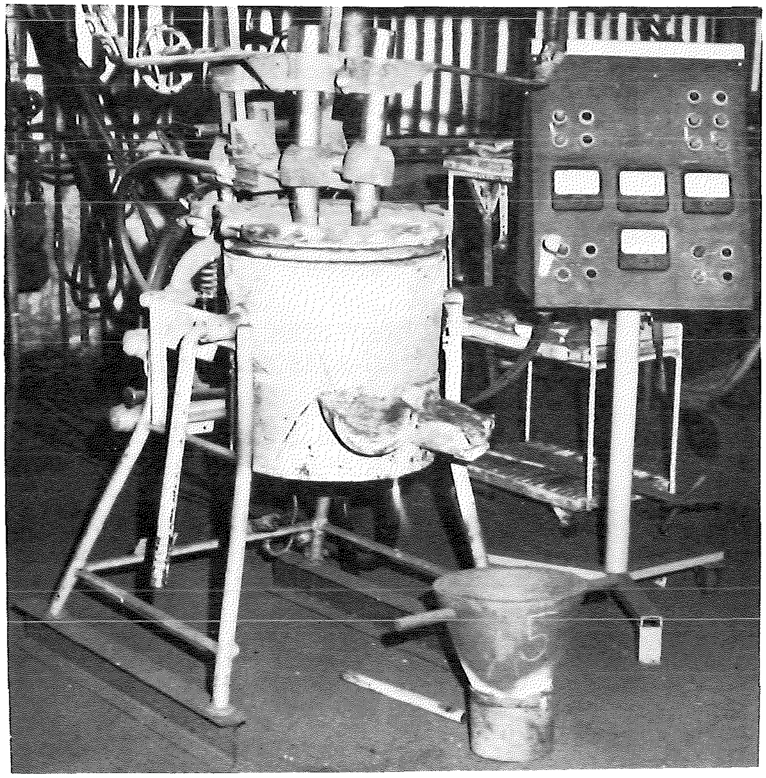

FIGURE 4. - 50-kV-A electric-arc furnace.

During a test, the charge was fed to the furnace by means of a 4-inch-wide belt feeder. The variable speed belt provided a constant feed rate to the furnace which could be correlated with energy input to obtain the desired smelting conditions in the furnace. Bath temperatures were measured with an immersion thermocouple just before tapping the furnace. The large surface-area-to-bath-volume ratio made it particularly difficult to control the bath temperatures in the 50-kV-A furnace and, in most cases, the tapping temperatures were higher than necessary.

The slag that was recovered from each smelting test was pulverized and a representative sample was submitted for analyses. Several random drill samples were taken from each metallic copper ingot and all of the shavings were submitted for analyses.

Single-Charge Smelting Tests in the 50-kV-A Electric-Arc Furnace

The initial smelting heats were single-charge tests made to determine the relationship of copper in the slag to iron in the metallic copper product when adding varying amounts of carbon in the furnace charge. The furnace charges and product analyses that were used to establish the basic relations are shown in table 3.

TABLE 3. - Test data used to establish iron in the metal versus copper in the slag as a function of the carbon additions

Test	Charge, lb			Pct stoichiometric C[2]	Fe in metal, pct	Cu in slag, pct
	Calcine[1]	SiO_2	C			
W-376....	33.6	2.16	1.0	45.0	0.034	21.9
W-377....	33.0	2.1	2.7	123.9	.030	13.3
W-378....	35.0	2.25	1.75	75.7	.05	11.7
W-379....	35.0	2.3	2.25	97.4	.084	2.9
W-380....	35.0	2.3	3.5	151.5	.38	1.99
W-383....	35.0	2.3	3.5	151.5	1.4	.78
W-388-A..	50.0	4.0	4	121.2	2.8	.84
W-390[3]...	50.0	-	4	121.2	2.9	.70
W-393....	50.0	-	4	121.2	.9	1.7
W-418....	35.0	3.0	2.8	121.2	.11	3.2
W-454....	35.0	2.86	2.88	121.2	12.1	1.28
W-459A...	30.0	2.43	1.20	60	.006	17.9
W-459B...	30.0	2.43	2.30	115	.15	3.79
W-459C...	30.0	2.43	2.00	100	.075	3.86
W-459D...	30.0	2.43	2.60	130	.065	4.00

[1] Roasted Anaconda concentrate.

[2] The stoichiometric carbon requirement was calculated on the assumption that all of the copper present in the calcine had been oxidized to CuO and must be reduced to metallic copper.

[3] Five pounds of lime were added to the charge.

A single batch of calcine and silica was blended for tests W-459A through D and the indicated amounts of carbon (contained in petroleum coke) were added to equal portions of the blend. The four tests indicated how changing the carbon additions affected the copper and iron distributions when other test parameters were held as constant as possible.

Results of the Single-Charge Smelting Tests in the 50-kV-A Electric-Arc Furnace

The relationship between copper in the slag and iron in the copper metal is plotted in figure 5. With appreciable concentrations of copper in the slag, iron contents of the metal products were low, usually something less than 0.05 pct. At about 4 pct copper in the slag and below, the amounts of iron contained in the metal rapidly increased. The curves shown in figure 6 represent the same data as functions of the amounts of carbon included in the furnace charge. The scattered data describing the iron contents of the metal bath as a result of the higher carbon additions are probably due to subtle changes in the bath conditions affecting carbon utilization. Five tests were run using petroleum coke additions equivalent to 121.2 pct of the stoichiometric-carbon requirement. The wide range of iron contents in the metal bath was due to the changes in SiO_2 and CaO additions that were made to determine the effect of flux additions on the copper product. Better partition of the iron and copper in the oxide smelting system appeared to be attainable with an acid slag. Therefore, only SiO_2 was added for subsequent tests.

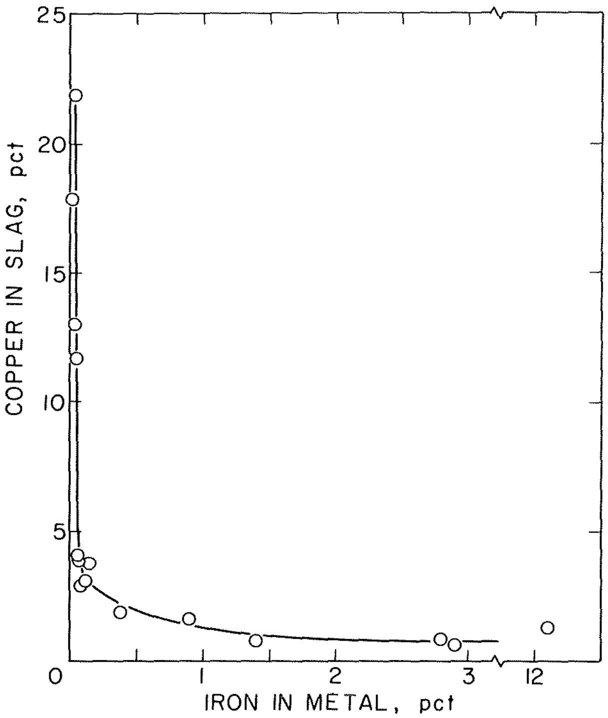

FIGURE 5. - Copper in slag versus iron in metal.

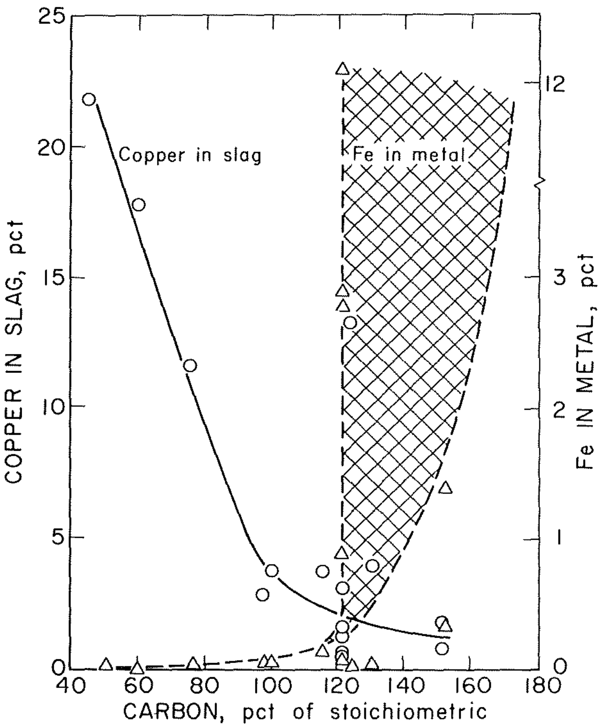

FIGURE 6. - The effect of carbon on the Cu and Fe distributions in the furnace bath.

Cyclic Smelting Tests in the 50-kV-A Electric-Arc Furnace

Several problems were anticipated before the series of cyclic smelting tests was started. These were as follows:

1. Residual carbon would carry over from a carbon-rich step to a carbon-lean step and increase the iron level in the blister copper product.

2. Changes in bath composition would require changes in the carbon additions from cycle to cycle.

3. Physical entrapment would cause high copper levels in the slags that were removed from the furnace while the metal portion of the bath was retained.

It was decided to establish a standard SiO_2 flux addition that would be adhered to when preparing most of the cyclic smelting test charges. The discussion of slag viscosities and softening temperatures by A. A. Tseidler (11) indicates that a slag containing a 75 pct FeO to 25 pct SiO_2 phase would allow the most latitude in iron distributions without realizing extensive increase in the melting point. Almost one-half of the contained iron could report to the metal bath and the low melting point range of the FeO-SiO_2 slag component could still be retained.

The cyclic charge compositions and iron and copper distributions in the smelting products are shown in table 4. The percent of stoichiometric values for SiO_2 that are included in table 4 fluctuate between carbon-rich and carbon-lean charges because the SiO_2 additions were held constant and it was assumed that more iron reports to the slag as Fe_2O_3 with carbon deficiencies than as FeO under reducing conditions. The carbon-rich charges (steps A and C) were varied from 121.2 to 212.1 pct of the stoichiometric requirements and SiO_2 additions were varied from 95.5 to 102 pct to determine the best approach toward minimizing the copper contents of the slags.

One smelting test was made in the 50-kV-A furnace after the cyclic-smelting test series to determine if the iron content of a metal bath could be lowered by adding copper calcine to the system. A blend of 30 lb of calcine, some SiO_2 flux, and 150 pct of the stoichiometric-carbon requirement was melted in the furnace and the metal bath was sampled. Enough calcine was added to lower the contained carbon to 130 pct of the stoichiometric requirement before a second metal sample was taken. The process was repeated at carbon levels equivalent to 115 and 50 pct of the stoichiometric requirement. The iron content of the metal samples decreased from 0.88 to 0.058 pct and a drill sample of the copper that was recovered when the furnace was tapped contained 0.01 pct iron.

Results of Cyclic Smelting Tests in the 50-kV-A Electric-Arc Furnace

Test results from the five three-step smelting tests included in table 4 indicated that the iron content of the metal from step B in each test was always under the 0.5 pct limit specified for blister copper. Therefore, carbon carryover from the carbon-rich steps was not a problem.

TABLE 4. - Cyclic smelting tests in the 50-kV-A electric-arc furnace

Test	Additions, pct stoichiometric		Metal, pct		Slag, pct	
	C	SiO_2	Cu	Fe	Cu	Fe
W-388:[1]						
A	121.2	95.5	97.4	2.8	0.84	48.5
B	0	81.2	-	-	29.7	32.2
C	121.2	95.5	99.2	.04	12.0	43.2
W-389:						
A	121.2	95.5	-	-	.95	45.6
B	0	81.2	98.2	.04	32.9	29.7
C	272.7	95.5	99.4	.06	7.7	41.6
W-392:						
A	151.5	102	-	-	2.1	46.0
B	64.84	87	98.7	<.01	9	43.0
C	151.5	102	98.8	.1	3.3	46.4
W-396:						
A	181.8	102	-	-	.8	48.4
B	60.6	87	98.8	.3	3.3	48.1
C	212.1	102	93.4	5.8	.8	46.9
W-401:						
A	136.3	102	-	-	3.1	48.7
B	60.6	87	99.2	.06	5.4	47.7
C	136.3	102	98.8	.2	1.9	50.5
W-403:						
A	138	102	-	-	3.6	49.8
B	60.6	87	98.9	.07	12.1	43.5
C	138	102	-	-	5.5	47.6
D	60.6	87	98.8	.05	10.7	31.6
E	138	102	-	-	5.7	46.7
F	60.6	87	98.2	.13	5.7	46.7
W-405:						
A	173.1	102	-	-	.7	50.7
B	86.5	87	95.4	2.8	1.0	49.5
C	173.1	102	-	-	.8	44.7
D	86.5	87	91.2	7.9	1.1	52.5
E	173.1	102	72.4	26.4	1.0	49.5
F	86.5	87	86.5	5.3	3.4	47.7
W-407:						
A	151.5	102	-	-	.9	44.4
B	64.8	87	97.8	1.1	2.3	42.8
C	151.5	102	-	-	1.7	27.9
D	64.8	87	96.9	.6	2.7	48.3
E	151.5	102	-	-	1.1	49.9
F	64.8	87	98.0	.6	3.2	50.6
W-409:						
A	138.5	102	-	-	1.2	47.6
B	60.6	87	99.4	-	8.3	46.8
C	138.5	102	-	-	2.4	50.2
D	60.6	87	99.1	-	6.4	47.7
E	138.5	102	-	-	1.7	50.7
F	60.6	87	99.0	.3	2.3	49.3
W-413:						
A	121.2	102	-	-	1.4	43.6
B	43.2	87	99.7	.02	14.0	-
C	121.2	102	-	-	6.0	43.2
D	43.2	87	98.8	.03	14.3	40.4
E	121.2	102	98.8	-	1.3	46.6
F	43.2	87	98.8	.3	8.5	45.5
W-416:						
A	121.2	102	-	-	5.2	45.2
B	43.2	87	99.8	.01	15.5	39.9
C	121.2	102	-	-	11.3	41.7
D	43.2	87	99.7	.03	-	-
E	121.2	102	-	-	7.7	45.0
F	43.2	87	99.2	.02	23.0	40.3
W-418:						
A	121.2	102	98.6	0.11	3.2	47.7
B	43.2	87	99.0	.02	4.8	38.3
C	121.2	102	98.2	.07	6.2	47.1
D	43.2	87	98.6	.03	-	-
E	121.2	102	-	-	7.6	46.9
F	43.2	87	98.4	.02	-	-
W-420:						
A	121.2	102	-	-	3.7	49.3
B	43.2	87	98.7	2.2	12.1	49.0
C	121.2	102	98.5	.2	4.0	49.1
D	43.2	87	97.8	.05	12.1	49.0
E	121.2	102	98.8	.3	6.3	46.3
F	43.2	87	99.0	.15	11.6	46.0
W-421:						
A	121.2	102	-	-	4.7	48.0
B	43.2	87	98.8	.04	17.3	39.9
C	121.2	102	-	-	5.2	46.7
D	43.2	87	98.2	.04	15.0	40.0
E	121.2	102	99.2	.05	4.6	48.2
F	43.2	87	-	-	7.7	45.9
W-424:						
A	138.5	102	-	-	1.3	47.5
B	60.6	87	95.8	3.7	.8	47.1
C	138.5	102	-	-	1.1	44.3
D	60.6	87	89.4	5.1	1.9	44.4
E	138.5	102	-	-	1.4	44.5
F	60.6	87	90.5	9.5	1.8	46.6
W-429:						
A	138.5	153	-	-	3.6	39.3
B	60.6	130.5	98.5	.2	3.4	41.9
C	138.5	153	-	-	2.2	39.7
D	60.6	130.5	96.1	3.0	1.7	41.1
E	138.5	153	-	-	1.3	41.0
F	60.6	130.5	92.6	7.1	2.5	41.4
W-432:						
A	138.5	153	-	-	3.6	45.0
B	60.6	130.5	98.9	.1	3.9	45.7
C	138.5	153	-	-	1.6	46.6
D	60.6	130.5	98.7	.3	2.4	46.4
E	138.5	153	-	-	2.1	45.1
F	60.6	130.5	98.2	.8	4.1	44.6
W-434:						
A	138.5	153	-	-	1.4	43.2
B	60.6	130.5	97.5	.3	1.8	46.7
C	138.5	153	-	-	1.6	46.8
D	60.6	130.5	96.9	1.2	1.8	48.0
E	138.5	153	-	-	1.7	48.2
F	60.6	130.5	96.2	.4	2.7	47.9
W-436:						
A	138.5	204	-	-	3.0	41.0
B	60.6	174	98.5	1.9	4.6	41.4
C	138.5	204	-	-	2.1	43.2
D	60.6	174	96.8	2.8	-	43.0
E	138.5	204	-	-	1.6	43.5
F	60.6	174	96.0	5.3	4.5	41.7
W-458:						
A	121.2	102	-	-	2.9	46.6
B	43.2	87	-	.006	8.2	44.1
C	163.7	102	-	-	1.2	47.0
D	33.2	87	-	.005	8.3	46.5
E	121.2	102	-	-	2.5	50.4
F	43.2	87	-	.006	6.4	46.6

[1] Slag is removed from the furnace after steps A, C, E, and F and metal is removed from the furnace after steps B, D, and F.

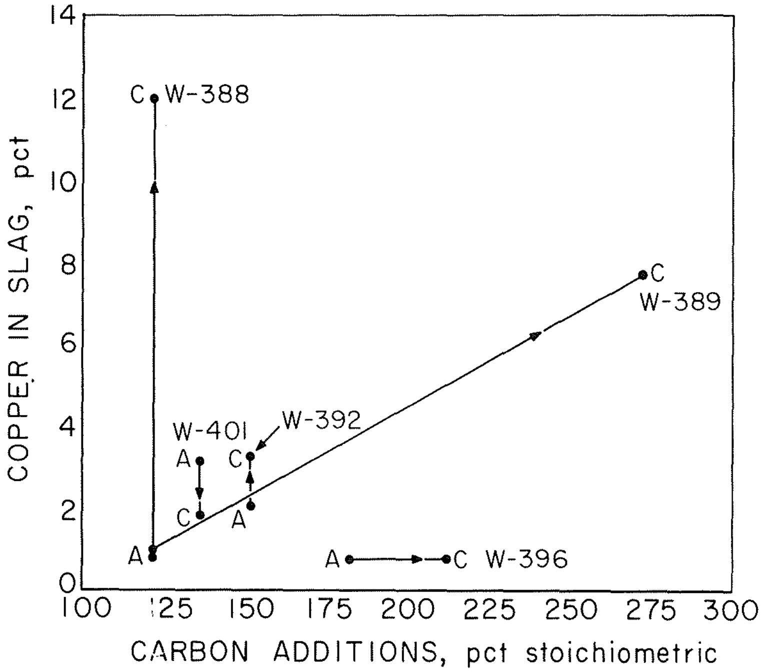

FIGURE 7. - Copper in slag versus carbon additions for three-step smelting tests.

Figure 7 compares the copper contained in the slags from steps A and C for each of the three-step tests. The data indicate that increasing the carbon additions as cyclic smelting progresses does not reduce the copper in the slag. However, extremely high carbon additions during the initial cycle improve the copper recovery. Although the SiO_2 additions were not included in figure 7, they were slightly lower in tests W-388 and W-389. The lower SiO_2 contents might have contributed to the variations in copper contents between slags A and C.

The six-step (three-cycle) tests shown in table 4 were primarily designed to minimize copper contents in the slags by using uniform carbon additions while reducing the carbon consumption as much as possible. When the charge compositions were held constant, the data were evaluated by averaging the copper contents of all the slags that were recovered in each test. This was done to minimize the apparent effects of the random process variables such as physical entrapment of metallic prills, bath temperature fluctuations, and variations in retention time. The copper contents of the slags and the iron contents of the copper buttons that were recovered are shown in figures 8 and 9, respectively. From these relationships, it was decided that by adding at least 160 pct of the stoichiometric carbon to the furnace charge, a low copper slag suitable for discard could be produced. An alternate charge containing 40 pct of the stoichiometric carbon requirement was chosen to produce a low-iron blister copper and make a stoichiometric carbon balance for each full smelting cycle.

The temperature readings that were taken during smelting tests in the 50-kV-A electric-arc furnace ranged from 1,225° to 1,625° C. The bath temperatures were plotted versus corresponding iron and copper distributions, copper recoveries, and oxygen levels in the copper products to determine the temperature effects. There were not any conclusive correlations between the bath temperatures and the values mentioned.

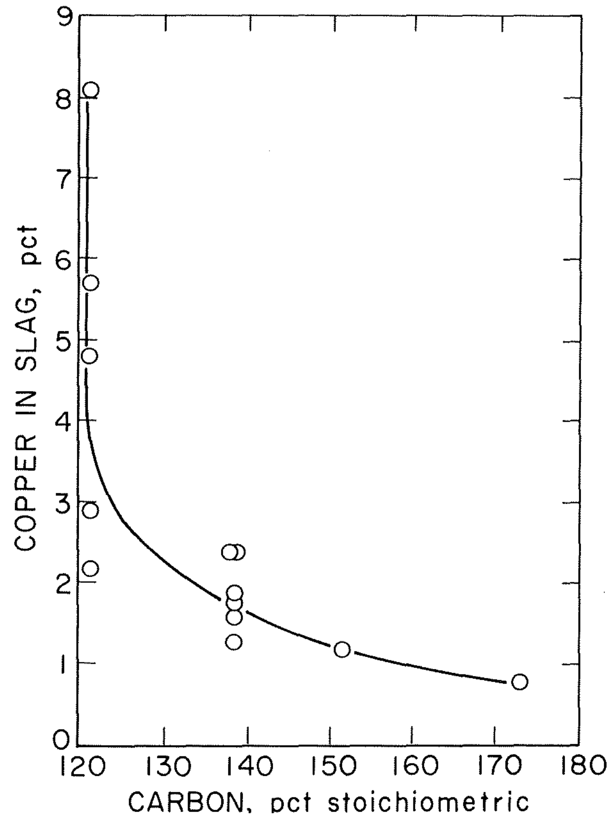

FIGURE 8. - Average Cu in slags versus carbon in carbon-rich charges for three-cycle (six-step) smelting tests.

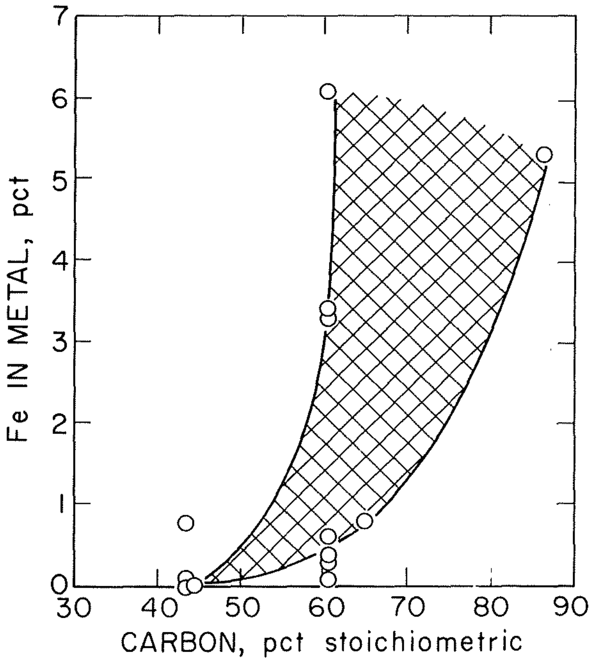

FIGURE 9. - Average Fe in metal versus carbon in carbon-lean charges for three-cycle (six-step) smelting tests.

SMELTING TESTS IN THE 800-kV-A ELECTRIC-ARC FURNACE

Materials Used for Smelting Tests in the 800-kV-A Electric-Arc Furnace

Calcined ASARCO chalcopyrite concentrate (see table 1) was used exclusively for the smelting tests in the 800-kV-A electric-arc furnace. Segregation due to the batch-type nature of the roasting operation caused appreciable variations in the copper and iron contents of the calcined concentrate. Therefore, it was necessary to analyze each barrel (approximately 1,000 lb) of calcine for copper and iron before the material was used to blend a furnace charge. The calcined concentrate was normally less than 4 mesh and had a composition as typified in table 5.

Petroleum and foundry cokes were used as reductants and silica was used as a flux for the smelting tests in the 800-kV-A electric-arc furnace.

TABLE 5. - Principal components of dead roasted concentrate

Component	Pct
Cu	38.0
Fe	29.3
S	1-1.5
SiO_2	3.11
Zn	2.13
Pb	.09
Al_2O_3	.93
CaO	.26
MgO	.03
Bi	.32

Equipment and Procedure for Preparing Charges To Be Smelted in the 800-kV-A Electric-Arc Furnace

The furnace charges were blended in the 8-inch diameter zigzag blender shown in figure 10 by proportioning the blend components into the hopper of a 4-inch bucket elevator that lifted feed into the blender. The blended material was discharged into 55-gallon drums. The entire unit was connected to a 7,500-cfm dust collection system with dampers available to prevent removing fines from the materials being blended.

The charge for each step of a cyclic smelting test in the 800-kV-A electric-arc furnace was eventually based on 1,000 lb of calcined chalcopyrite concentrate. Petroleum coke was added to provide either excess carbon or carbon deficiencies in relation to the carbon required to reduce all of the contained copper from CuO to metal and enough silica was added to make a 65 pct FeO to 35 pct SiO_2 slag.

Equipment and Procedure for Smelting Tests in the 800-kV-A Electric-Arc Furnace

Figure 11 shows the 800-kV-A electric-arc furnace and the ancillary equipment. The elevated hopper shown on the left held the blended furnace charges and was emptied by a variable speed-belt feeder that was used both in conjunction with a horizontal slinger and when the furnace was gravity fed.

Initially, the horizontal slinger located below the elevated hopper in figure 11 was used to throw the blended charge through a water-cooled port in the furnace wall. The cold charge contacted the superheated and turbulent bath within the furnace delta causing instantaneous melting of the charge material. Continuous operation of the slinger system required that an appreciable amount of air be blown across the bottom of the water-cooled port to prevent plugging. To reduce the amount of air entering the system, the blended charge was gravity fed through a hole in the furnace roof during the latter part of the test series. The gravity feeding alleviated the operational problems and the change did not appear to affect the test results.

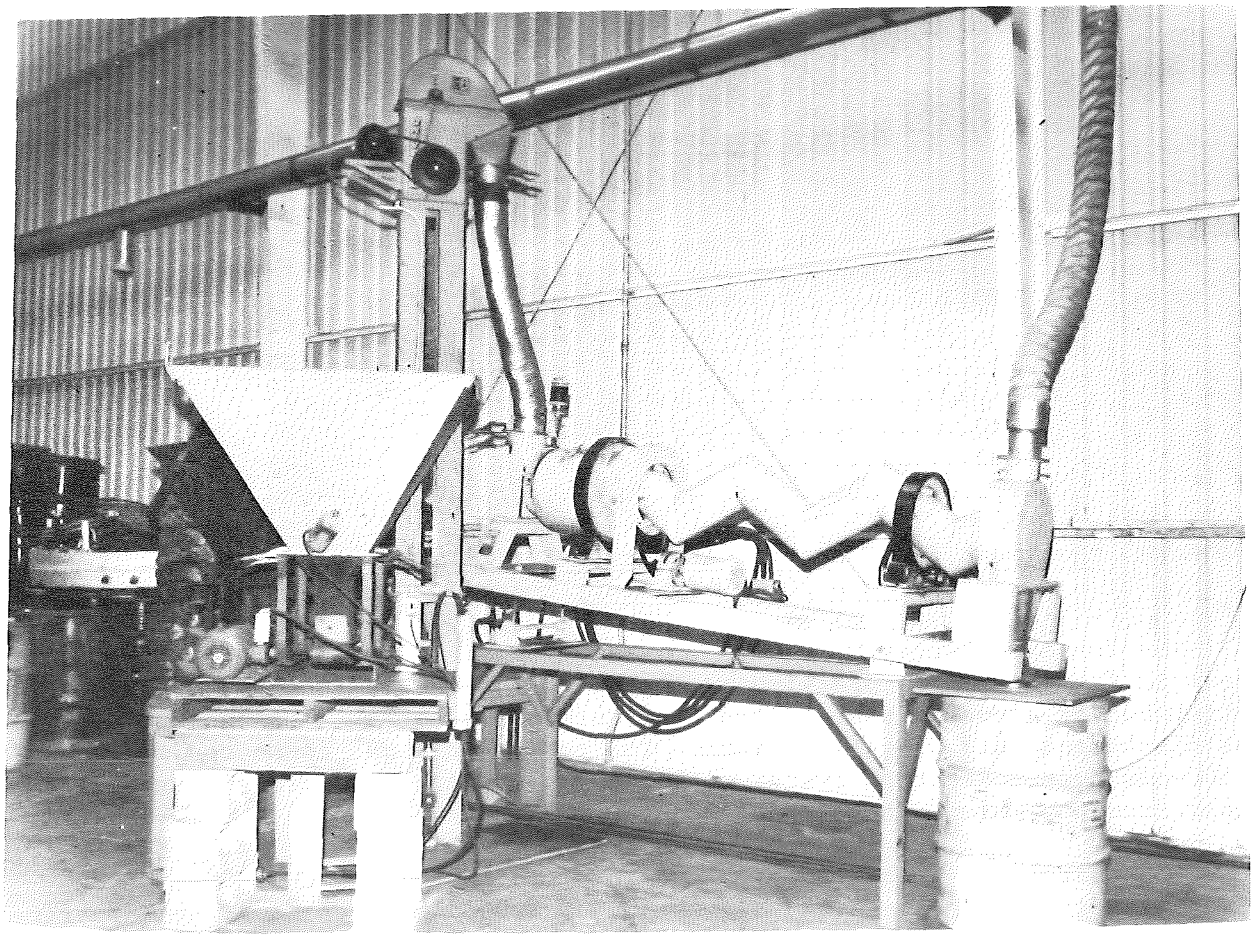

FIGURE 10. - Eight-inch diameter zigzag blender.

The 800-kV-A electric-arc furnace was 35 inches in inside diameter and was powered through three 4-inch-diameter graphite electrodes. The conventional round bottom tilting furnace shell was modified to accommodate the deeper baths that are necessary to successfully remove metal from under the slag layer in the cyclic smelting process. The tilting mechanism was removed from the furnace and the bottom of the shell was extended 18 inches to where a flat bottom was supported by 4-inch channel irons to allow circulation of air between the furnace and the brick foundation. The metal taphole was located at hearth level and the slag taphole was 7 inches up the sidewall. The metal bath portion of the modified furnace shell was lined with basic (MgO), brick and silica bricks were used to line the upper sidewalls. The completed furnace accommodated a molten bath 18 inches deep and weighing approximately 5,000 lb.

The furnace was equipped with a plenum hood and the positive pressure dust-collection system consisted of a cyclone and baghouse downstream from a 7,500-cfm blower.

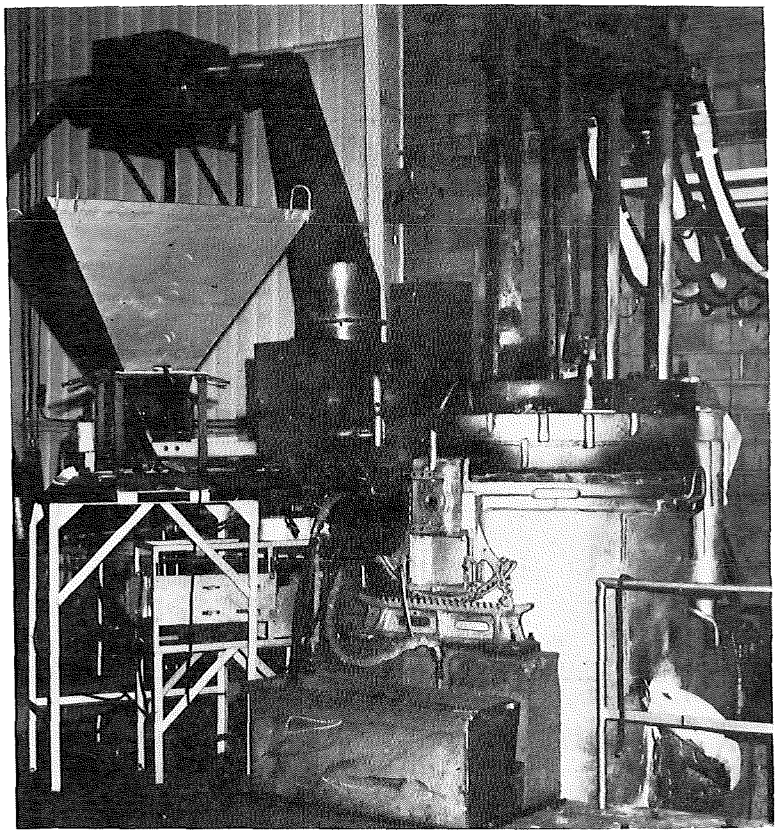

FIGURE 11. - 800-kV-A electric-arc furnace.

Metal products from the smelting tests were poured into molds that formed 50 lb metal pigs and the slags that were removed from the furnace were cooled in conical slag ladles that formed billets containing all of the slag that was poured during each tap.

Techniques for Sampling and Analyzing the Smelting Products From the 800-kV-A Electric-Arc Furnace

An analytical scheme was developed to rapidly approximate the iron content of a small blister copper sample that was taken from the metal bath before the metal was tapped from the furnace. The technique was based on comparing $Fe(OH)_3$ collected on smooth texture filter papers with standards prepared as follows:

1. Dissolve 100 mg of pure iron wire in a mixture of 10 ml of 20 pct HCl and 10 ml of 20 pct HNO_3 and dilute the solution to 1 liter.

2. Prepare aliquotes containing 5, 10, 25, and 50 ml of the standard solution.

3. Precipitate the contained iron in each aliquote with NH_3OH and collect the resulting $Fe(OH)_3$ from each aliquote on a separate filter paper.

The resulting standard precipitates are equivalent to iron contents of 0.05, 0.1, 0.25, and 0.5 weight-percent iron in a 1-gram sample of blister copper.

Analytical Procedure:

1. Dissolve the blister copper sample in dilute HNO_3 and heat the solution to boiling.

2. Add excessive amounts of NH_3OH to precipitate the iron as $Fe(OH)_3$ and to keep the copper amines in solution.

3. Filter the solution rapidly and wash with water.

4. Approximate the iron level in the blister copper by comparing the amount of precipitate with the prepared standards.

The smelting furnace was held at temperature and allowed to equilibrate during the 15 min that were required to prepare the blister copper sample and to perform the analysis. Results of the rapid iron analysis consistently agreed with the quantitative analyses that were eventually performed by standard techniques on the blister copper products.

All of the slag from each tap was crushed in a primary jaw crusher and gyratory mill before the material was split down to a 20 lb sample for additional processing. The slag samples were ground to minus 80 mesh and were thoroughly blended before they were submitted for chemical analyses.

Random drill samples were taken from each one of the metal pigs and a composite sample was prepared from each tap for chemical analyses. In addition to this, one tap was randomly sampled and extensively analyzed to determine the homogenuety and total impurity content of the 50 lb pigs. Two skin samples and seven drill samples that were taken diagonally across one randomly chosen pig were analyzed individually to determine the distribution of gold and silver across the cooling profile of a single pig. A composite of the assayed samples and random drill samples from seven additional pigs were analyzed for total impurities.

One- and Two-Charge Smelting Tests in the 800-kV-A Electric-Arc Furnace

The first half of the smelting tests in the 800-kV-A electric-arc furnace consisted of one- and two-charge tests that were made to break in the newly installed 800-kV-A furnace and develop the operating proficiencies required for a successful cyclic operation. The initial smelting tests included radical changes in feed rates, charge weights, smelting temperatures, and other less significant operating techniques. Although much of the data that were collected from each test could not be correlated with other results, an appreciable amount of significant information was made available as shown in table 6.

TABLE 6. - Data from initial smelting tests in the 800-kV-A electric-arc furnace

Test	Total furnace charge, lb	Pct stoichiometric C	Metal product,[1] pct						Slag, pct				
			Fe	Pb	Zn	S	Ag	Au	Cu	Fe	FeO[2]	SiO_2	FeO/SiO_2
LM-1..	2,127	40	<0.005	-	-	-	-	-	23.5	27.1	34.9	19.1	1.8
LM-2..	1,715	160	4.09	-	-	-	-	-	3.13	39.9	51.3	29.4	1.7
LM-5..	2,234	160	.58	~0.1	~0.1	-	-	-	1.93	46.6	60.0	23.6	2.5
LM-6..	2,727	160	1.36	-	-	0.82	-	-	1.70	46.9	60.3	24.6	2.5
LM-8:[3]													
A...	2,451	130	-	-	-	-	-	-	3.04	46.6	60.0	23.0	2.6
B...	2,090	40	.23	-	-	.8	-	-	5.13	39.6	50.9	21.6	2.4
LM-9:													
A...	1,942	160	.82	-	-	-	-	-	1.56	42.2	54.3	20.6	2.6
B...	1,805	40	.19	-	-	.724	0.001	0.30	4.0	44.8	57.6	21.1	2.7
LM-10:													
A...	2,006	170	-	-	-	-	-	-	3.66	41.8	53.8	28.4	1.9
B...	1,764	30	.003	.005	.050	.686	-	-	9.28	38.9	50.0	24.5	2.0

[1] Balance assumed to be Cu.

[2] Calculated values.

[3] Two-step test: A = slag tap; B = metal tap.

Results of the One- and Two-Charge Smelting Tests in the 800-kV-A Electric-Arc Furnace

The initial smelting tests in the 800-kV-A electric-arc furnace indicated that two operating techniques had to be changed in order to improve the copper recovery and the quality of the blister copper that was produced. First, the furnace charges that were based on 1,500 lb of calcine were too large and raised the slag-metal interface up too close to the slag taphole causing large copper prills to be entrained with the slag as it poured from the furnace. Second, the original metal taphole which was 2 square inches through the magnesite brick rapidly erroded to approximately 3 square inches. A high tapping rate caused the metal bath to vortex past the slag-metal interface and some copper-bearing slag was entrained with the blister copper that was being removed from the furnace. The subsequent furnace charges were based on 1,000 lb of calcine to lower the slag-metal interface away from the slag taphole and a water-cooled graphite tapping block with a 1-square-inch taphole was installed at hearth level to restrict the tapping rate of the blister copper. There was never any problem with high iron levels in the blister copper products after the smaller taphole was installed.

The one- and two-charge smelting tests indicated that the system including electrode stability, bath uniformity, feeding system, and the exhaust system, operated best when the blend to be smelted was fed to the furnace at approximately 1,500 lb/hr.

Cyclic Smelting Tests in the 800-kV-A Electric-Arc Furnace

The testing procedures that were established were held as consistent as possible during most of the four-step (two-cycle) smelting tests so that bath compositions, operating temperatures, and associated chemical parameters could be better evaluated. An operating schedule for a typical four-step (two-cycle) smelting test is as follows:

Preheat to 1,000° C.

1. Feed charge A_1 (excess carbon).
2. Hold for 15 min.
3. Tap "throwaway" slag.
4. Sample metal bath.
5. Feed charge B_1 (low carbon).
6. Hold for Fe analysis.
7. Tap metal.

Repeat steps 1 through 7 with charges A_2 and B_2. Empty furnace.

The arc was started across some copper metal on top of about 400 lb of material from charge A_1. The large initial bed was used to protect the furnace bottom during startup. A high copper slag (near 10 pct) was tapped when the furnace was emptied. However, the contained copper was not considered lost because the slag would remain in the furnace for the next step of a continuous operation.

Two smelting tests were designed to improve the utilization of the petroleum coke and lower the copper content of the slag (increase the iron content of the metal bath) after smelting the carbon-rich charges. In lieu of using constant feed and power input rates to maintain an entirely molten system, the carbon-rich charges were "choke-fed" into the furnace. A layer of solids was allowed to build up on top of the bath to improve contact between the calcine and reductant as they started smelting.

Other smelting tests included silicon carbide additions to increase the iron level in the retained metal, slight adjustments in the slag factor (Al_2O_3/CaO) (12), and tapping the "throwaway" slags through a preheated weir. Foundry coke was used in lieu of petroleum coke in tests LM-16 and LM-17. None of the changes that were tried caused any appreciable increase in copper recovery.

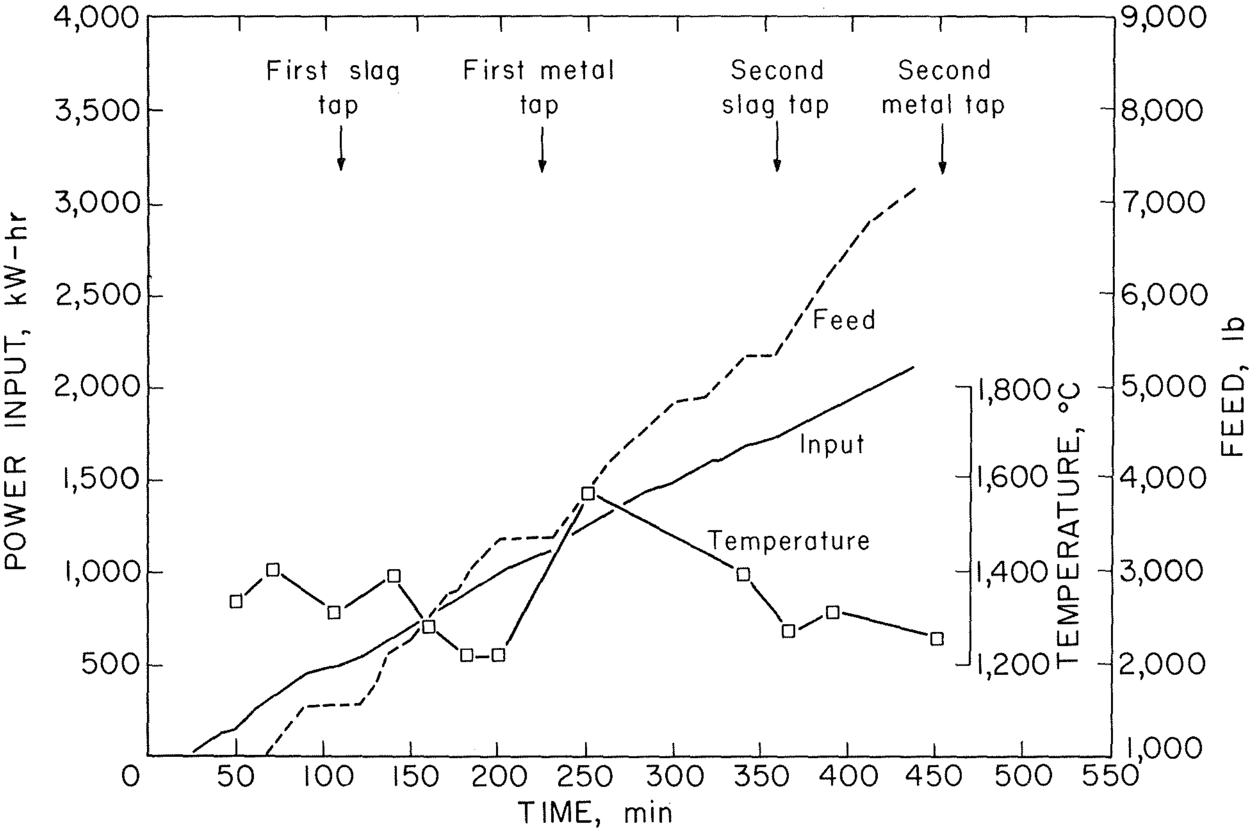

FIGURE 12. - Operating data from a typical smelting test.

Results of Cyclic Smelting Tests in the 800-kV-A Electric-Arc Furnace

Materials data from the four-step (two-cycle) smelting tests are shown in table 7. The copper recoveries shown in the last column are calculated to compare the effects of smelting parameters on a continuous operation. The calculations treat the high copper slags that are inadvertently tapped along with metal during and at the end of an experimental test as material retained in the system. As a result the copper losses are based only on copper contained in the "throwaway" slags. A typical calculation is shown in table 8.

Table 9 contains the operating data from the four-step (two-cycle) smelting tests. The feed rates, average bath temperatures, and power consumptions that are listed were determined by graphical analyses of the test data after it was plotted as shown in figure 12. The plateaus in the feed curve indicate intentional 15-min holding times and experimental delays in tapping the furnace. If the delays were excessive, deductions were made from the power input figures before the representative power consumption value was calculated.

The effects of "choke-feeding" the carbon-rich charges are demonstrated by comparing tests LM-16 and LM-13 as shown in table 10. Test LM-16 is typical of several tests during which feed rates were held constant at near 1,500 lb/hr of blend and power inputs were adequate to keep the entire furnace charge molten. Regardless of the flux additions, the copper content in the "throwaway" slag remained too high. Test LM-13 typifies smelting tests that were run to determine if high feed rates would improve the utilization of the petroleum coke and lower the copper content of the "throwaway" slags. "Choke feeding" lowered the copper content of the lean slags and increased the calculated continuous copper recovery. However, the iron content of the blister copper increased because excess carbon remained in the bath after the slag was removed from the furnace. The quality of the blister copper was not improved by extending the holding time or by taking precautions to remove all of the visible carbon from the furnace before feeding a carbon-deficient charge to the furnace.

TABLE 7. - Materials data from four-step (two-cycle) smelting tests in the 800-kV-A electric-arc furnace

Test	Charge composition						Metal										Slag									
									Analysis[1]										Analysis, pct							
	Calcine, lb	Coke, lb	Stoichiometric C, pct	SiO_2 lb	CaO, lb	Cu, lb	Tap	Weight, lb	Fe, pct	C, pct	S, pct	Zn, pct	Pb, pct	Bi, pct	Au, ppm	Ag, pct	Tap	Weight, lb	Cu	Fe	SiO_2	CaO	Al_2O_3	MgO	S	Recovery,[2] pct Cu
LM-11:																										
A...	1,492	204	170	248	-	63	-	-	-	-	-	-	-	-	-	-	Slag A.......	363	3.95	43.7	27.9	2.17	3.21	0.47	ND	93.6
B...	1,502	35	30	271	-	-	Pigs A	490	0.002	ND	0.78	ND	ND	ND	7.5	0.32	Metal A......	143	11.1	37.6	26.2	1.87	3.87	.43	ND	
C...	1,506	193	170	274	-	-	-	-	-	-	-	-	-	-	-	-	Slag B.......	1,738	3.78	40.7	28.5	2.34	6.65	.50	ND	
D...	1,515	35	30	276	-	-	Pigs B	1,232	.003	ND	.79	ND	ND	ND	6.3	.32	Metal B......	[3]1,861	11.5	38.3	30.2	2.17	4.61	.48	ND	
																	Dust.........	81	24.5	26.9	5.38	.41	1.25	.06	ND	
LM-12:																										
A...	991	131	170	166	29.7	90	-	-	-	-	-	-	-	-	-	-	Slag A.......	370	5.11	41.6	28.2	1.22	4.22	1.62	0.248	96.4
B...	996	23.5	30	164	29.9	-	Pigs A	442	.001	ND	.703	0.015	0.002	ND	ND	ND	Metal A......	498	12.0	38.1	24.6	1.85	3.49	1.28	.353	
C...	995	129	170	179	29.8	-	-	-	-	-	-	-	-	-	-	-	Slag B.......	808	1.4	42.6	30.3	3.50	6.02	.71	.564	
D...	445	10.2	30	80	13.4	-	Pigs B	653	.36	ND	ND	ND	ND	ND	ND	ND	Metal B......	877	14.4	28.5	21.7	.42	ND	ND	2.63	
LM-13:																										
A...	1,000	141	170	168	31	227	-	-	-	-	-	-	-	-	-	-	Slag A.......	307	1.51	43.1	28.5	5.73	2.49	1.93	.503	98.4
B...	1,000	24.8	30	168	31	-	Pigs A	820	.87	ND	.776	ND	ND	ND	ND	ND	Metal A......	378	1.72	41.9	28.2	5.70	5.06	1.39	.668	
C...	1,000	141	170	168	31	-	-	-	-	-	-	-	-	-	-	-	Slag B.......	698	1.17	40.8	28.1	5.81	5.34	2.87	.438	
D...	858	20.4	30	150	35	-	Pigs B	567	1.72	ND	.779	ND	ND	ND	ND	ND	Metal B......	1,444	15.3	35.0	22.8	4.88	3.38	1.37	3.86	
LM-15:																										
A...	1,005	127	170	186	42	149	-	-	-	-	-	-	-	-	-	-	Slag A.......	485	1.23	31.5	33.8	6.83	8.90	6.92	.449	91.7
B...	1,000	21.1	30	172	42	-	Pigs A	475	.25	ND	.635	.05	.01	0.49	9	.09	None.........	-	-	-	-	-	-	-	-	
C...	1,000	133	170	170	42	-	-	-	-	-	-	-	-	-	-	-	Slag B.......	1,070	5.18	39.2	26.3	5.49	3.47	2.95	.128	
D...	389	9.4	30	66	16	-	Pigs B	638	1.6	ND	.807	.23	.06	.49	9	.09	Metal B......	1,290	1.58	34.7	26.7	5.02	14.8	2.86	.372	
LM-16:																										
A...	992	117	170	181	42	100	-	-	-	-	-	-	-	-	-	-	Slag A.......	389	2.38	40.7	29.3	5.64	4.58	4.38	ND	95.3
B...	1,000	23.5	30	183	42	-	Pigs A	565	.005	ND	ND	<.05	.02	1.0	ND	ND	Metal A......	565	15.6	44.5	24.2	4.48	3.54	3.21	ND	
C...	984	115	170	180	41	-	-	-	-	-	-	-	-	-	-	-	Slag B.......	1,151	2.94	37.9	31.2	6.05	4.19	8.90	ND	
D...	995	20.7	30	182	42	-	Pigs B	392	.005	ND	ND	<.05	.04	1.5	ND	ND	Metal B......	1,349	15.2	42.9	22.8	4.69	2.58	7.86	ND	
																	Baghouse dust		19.0	26.7	6.29	.20	.62	.40	(4)	
LM-17:																										
A...	996	132	175	166	42	121	-	-	-	-	-	-	-	-	-	-	Slag A.......	417	1.77	42.4	28.6	6.45	3.14	1.23	ND	96.9
B...	1,008	19.6	25	179	42	-	Pigs A	439	.48	ND	.70	.26	.09	.44	ND	ND	Metal A......	49	2.72	42.1	29.3	4.11	2.50	1.80	ND	
C...	1,000	125	175	185	42	-	-	-	-	-	-	-	-	-	-	-	Slag B.......	1,120	1.75	41.0	29.8	5.77	4.50	1.85	ND	
D...	1,000	19.6	25	171	42	-	Pigs B	879	.14	ND	.79	.13	.07	.35	ND	ND	Metal B......	1,256	5.22	43.4	26.8	5.13	3.46	1.41	ND	
LM-18:																										
A...	994	132	175	166	42	50	-	-	-	-	-	-	-	-	-	-	Slag A.......	399	3.11	43.4	27.8	7.49	3.31	.95	.25	89.0
B...	997	15.1	25	198	31	-	Pigs A	551	.035	0.36	.52	.012	.02	.95	ND	ND	Metal A......	880	12.8	42.8	25.5	5.43	2.98	.84	.084	
C...	1,000	112	175	199	31	-	-	-	-	-	-	-	-	-	-	-	Slag B.......	861	7.21	41.7	27.5	4.94	4.93	1.11	.045	
D...	1,007	15.7	25	193	30	-	Pigs B	404	.026	.39	.67	<.01	<.01	1.55	ND	ND	Metal B......	1,222	12.5	42.4	26.0	4.83	2.83	.73	.06	
LM-19:																										
A...	1,096	127	170	167	27	142	-	-	-	-	-	-	-	-	-	-	Slag A.......	437	2.56	40.7	32.2	3.51	2.63	1.68	.417	95.9
B...	1,000	21.6	30	190	30.7	-	Pigs A	585	.03	.158	.994	.009	.03	.82	ND	ND	None.........	-	-	-	-	-	-	-	-	
C...	995	135	170	178	30.5	-	-	-	-	-	-	-	-	-	-	-	Slag B.......	1,221	2.02	41.7	30.4	4.18	2.34	2.62	.146	
D...	995	23	30	176	30.5	-	Pigs B	641	<.01	.174	1.07	.02	.04	.64	ND	ND	Metal B......	1,503	11.4	36.5	26.8	4.22	2.02	1.67	.070	
LM-20:																										
A...	1,000	126	170	178	30	40	-	-	-	-	-	-	-	-	-	-	Slag A.......	269	1.25	40.5	33.0	5.60	2.76	1.12	ND	96.6
B...	1,010	23	30	180	31	-	Pigs A	527	.07	.338	ND	.12	.09	.57	ND	ND	Metal A......	540	2.84	42.9	27.6	4.80	1.64	.94	ND	
C...	1,016	122	170	198	31	-	-	-	-	-	-	-	-	-	-	-	Slag B.......	797	2.44	41.8	30.7	4.96	1.93	.82	ND	
D...	1,000	20	30	202	31	-	Pigs B	635	.07	.028	ND	.08	.12	1.12	ND	ND	Metal B......	1,395	2.69	43.1	29.4	4.22	1.87	.84	ND	
																	White metal..	72	79.9	ND	ND	ND	ND	ND	ND	
LM-21:																										
A...	847	95.7	170	163	26	120	-	-	-	-	-	-	-	-	-	-	Slag A.......	197	3.64	40.4	29.7	6.61	4.11	2.26	.515	93.5
B...	838	17.9	30	151	26	-	Pigs A	405	.003	.391	.836	.1	.06	2.2	ND	ND	None.........	-	-	-	-	-	-	-	-	
C...	819	104.7	170	150	25	-	-	-	-	-	-	-	-	-	-	-	Slag B.......	1,171	3.33	40.3	26.8	4.95	2.70	1.64	.114	
D...	891	19.9	30	165	30	-	Pigs B	641	.002	.399	.738	.03	.04	2.0	ND	ND	Metal B......	1,142	7.80	39.5	24.5	4.65	1.89	1.18	.10	

ND Not detected.

[1] Balance assumed to be Cu.

[2] Calculated value of Cu recovered as metal for a continuous operation.

[3] Includes splattered materials and cleanup.

[4] 33 Zn; 34 Pb; and 4.9 Bi.

TABLE 8. - Copper recovery calculated for a continuous operation[1]

Test data:
Cu charged to furnace = 1,734 lb
Cu lost from system:
Slag tap 1 = 307 at 1.51 pct Cu
Slag tap 2 = 698 at 1.17 pct Cu
Last metal tap (slag) = 1,408 at 15.3 pct Cu

Average slag tap Cu losses:
(307) (0.0151) = 4.64 lb
(698) (0.0117) = 8.17 lb
Total Cu lost = 12.80 lb Average = 1.27 Cu

True Cu loss from last tap:
1,408 - (1,408) (0.153) = 1,192.58 lb of slag in last tap
1,192.58 + contained Cu = new slag total = A
1,192.58 + (0.0127) (A) = A, A = 1,207.92 lb
Contained Cu = (0.0127) (1,207.92) = 15.3405 lb

Total Cu lost = 12.8023 + 15.3405 = 28.14 lb

$$\text{Continuous copper recovery} = \frac{[1{,}734 - 28.14]}{1{,}734}\,(100) = 98.37 \text{ pct}$$

---

[1] Calculated from LM-13 copper losses.

TABLE 9. - Operating data from four-step (two-cycle) smelting tests in the 800-kV-A electric-arc furnace

Test	Feed rate, lb/hr	Average bath temperature, ° C	Power consumption, kW-hr/ton of blend	Electrode consumption, lb/ton of blend
LM-11	1,320	1,360	554	15
LM-12	930	1,457	645	30
LM-13	1,627	1,375	525	17
LM-15	1,778	1,450	565	31
LM-16	1,608	1,400	599	28
LM-17	1,589	1,338	512	17
LM-18	1,959	1,290	408	14
LM-19	1,492	1,370	478	13
LM-20	1,444	1,170	461	11
LM-21	1,433	1,380	470	16

TABLE 10. - Smelting test results for "uniform-feeding" and "choke-feeding" carbon-rich furnace charges

Test	Iron in blister copper, pct	Cu in "throwaway" slag, pct	Continuous copper recovery, pct
LM-16..............	0.005	2.04	95.26
LM-13..............	1.22	1.27	98.37

DISCUSSION

The project objectives were to produce a blister copper containing less than 0.5 pct iron. Results from initial smelting tests in the 50-kV-A electric-arc furnace (figs. 5 and 6) indicated that the chemical equilibriums, dictated by various carbon additions, make the recovery of a high-quality blister copper easier to achieve than reducing the copper content of the "throwaway" slags to 1 pct or less. This was confirmed by the cyclic smelting tests in the 50- and 800-kV-A electric-arc furnace.

Reducing the copper content in the "throwaway" slags was emphasized throughout the investigation. The lowest copper levels achieved in any one tap were 0.80 and 1.17 pct during the cyclic smelting tests in the 50- and 800-kV-A electric-arc furnaces, respectively. The average copper contents in the "throwaway" slags during a four-step (two-cycle) smelting test in the 800-kV-A electric-arc furnace ranged from 1.27 to 5.91 pct. Most of the contained copper could be recovered by froth flotation as described in appendix A. Failure to reach the less than 1 pct copper level was partially attributed to the configuration and inherent bath condition of the 800-kV-A electric-arc furnace.

Metallographic and electron microprobe examinations of the "throwaway" slags indicated that most of the copper in the slags was present as metallic prills as shown in figure 13. The electron microscan indicates that very

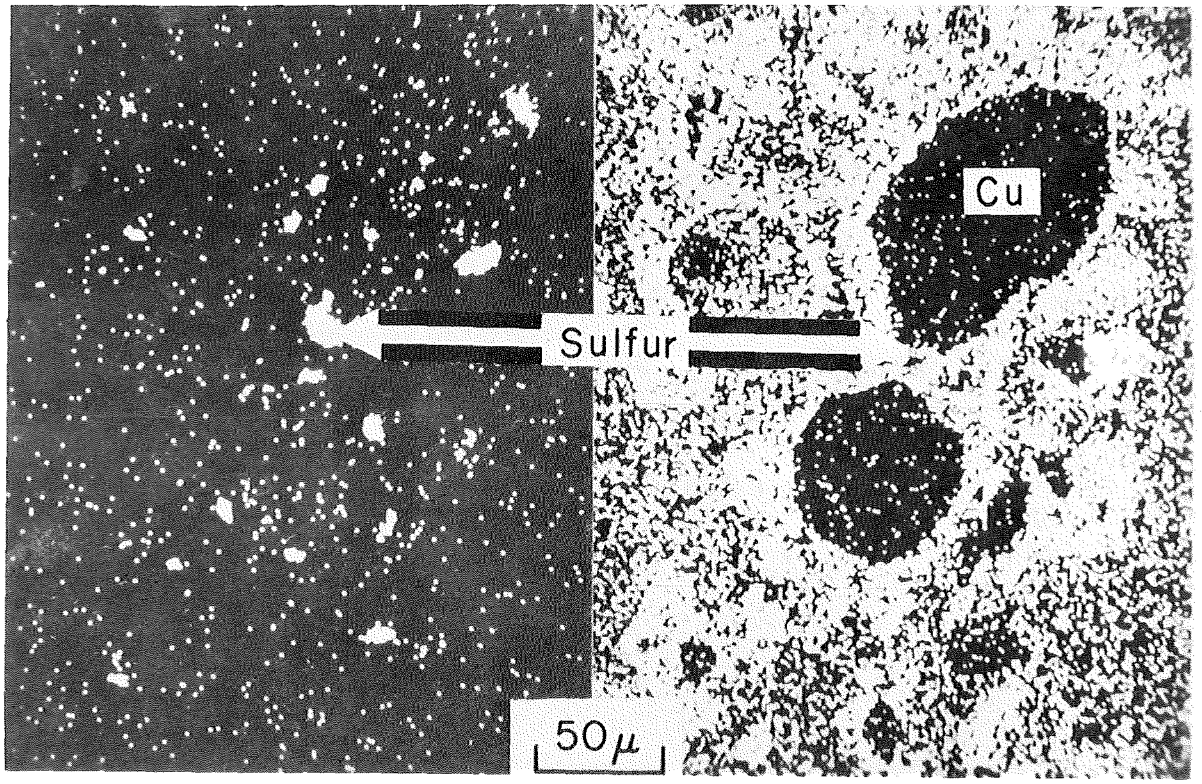

FIGURE 13. - "Throwaway" slag.

little sulfur was absorbed in the copper prills but collected in isolated spots on the prill surfaces. It is anticipated that an improved furnace design locating the tapholes farther away from the turbulent electrode delta would allow the larger copper prills that were observed to precipitate before tapping "throwaway" slags from the furnace.

Chemical analyses of the metallic copper products from smelting tests in the 50- and 800-kV-A electric-arc furnaces indicated that there was very little difficulty in maintaining quality. The total impurities analysis shown in table 11 indicates that essentially all of the gold and silver contained in the starting material reported to the copper metal. The high bismuth content of the blister copper was rather disappointing. However, the as-received ASARCO concentrate (table 1) contained a prohibitive 0.67 pct bismuth. Fifty-five percent of it was removed in the dead roasting and electric furnace smelting which is approximately the same removal as realized in conventional smelting processes.

TABLE 11. - Total impurities analysis of blister copper[1]

Element	Average content	Range
Bi....................pct..	1.0	-
Fe....................pct..	.001	0.001 -0.002
Ni....................pct..	.04	-
O_2... ....pct..	.1003	.0957- .1050
Pb....................pct..	.002	.002 - .003
S.....................pct..	.703	.669 - .725
Sn....................pct..	<.01	-
Zn....................pct..	.015	.009 - .039
Ag....................pct..	.37	.33 - .38
Au....................ppm..	9	5-15

[1] Total impurities = 2.23 pct.

Graphical analyses of the operating parameters and recovery data indicated that both intentional and inadvertent variations in the test parameters affected the copper recovery more than chemical changes that were made in the furnace charge materials. Because power consumption values reflect both operating temperatures and retention times, they are the individual parameter that best represents the actual smelting conditions. The continuous copper recoveries from the four-step (two-cycle) tests in the 800-kV-A electric-arc furnace are plotted versus their respective power consumption values in figure 14. The resulting curve indicates that the optimum smelting conditions would be at a power input of approximately 520 kW-hr/ton. Although there is considerable scatter in the data, the mean bath temperature and feed rate that were associated with the 520-kW-hr/ton power input were 1,350° to 1,375° C and 1,500 to 1,600 lb/hr, respectively. The optimum feed rate is equivalent to smelting 225 lb of blend per square foot of bath surface per hr.

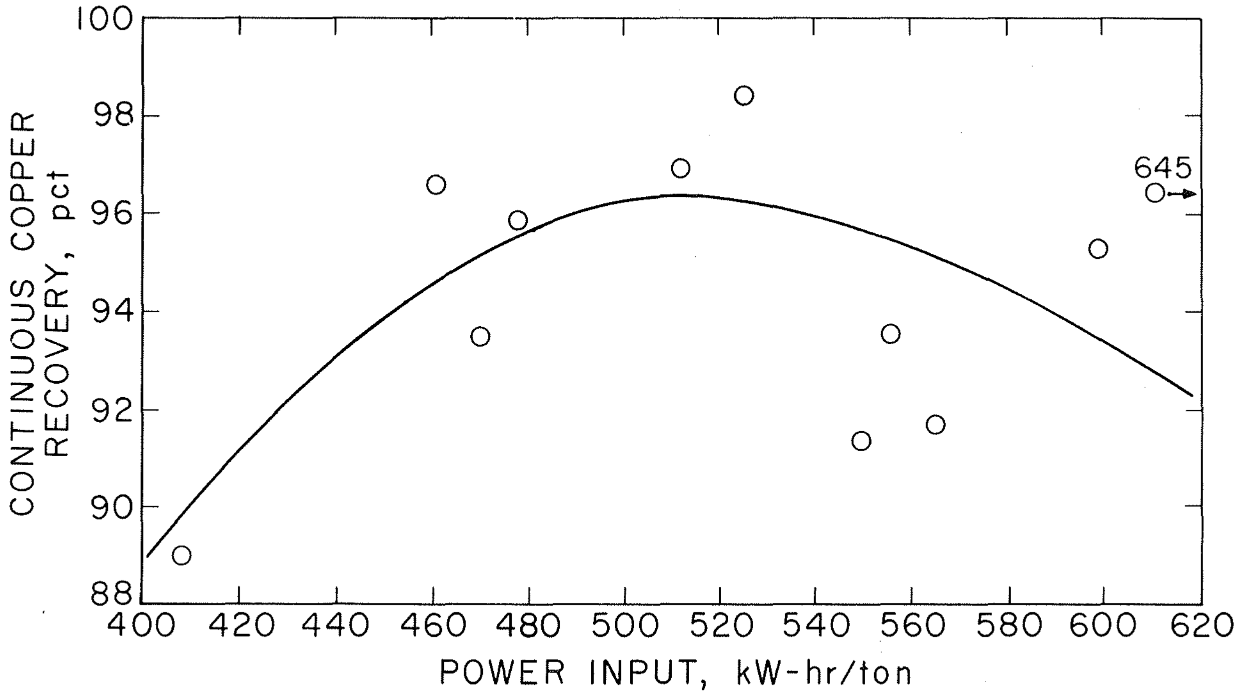

FIGURE 14. - Continuous copper recovery versus power input.

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APPENDIX.--FROTH FLOTATION OF SMELTING FURNACE SLAGS

A brief study was made to determine the feasibility of recovering additional copper by froth flotation from the "throwaway" slags. By using three flotation stages, 84.3 pct of the copper in the slag was recovered in a concentrate containing 12.42 pct copper. "Throwaway" slag from test LM-13 was used in the study and the froth flotation results indicated a potential for increasing the continuous copper recovery from 98.4 to more than 99.5 pct.

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