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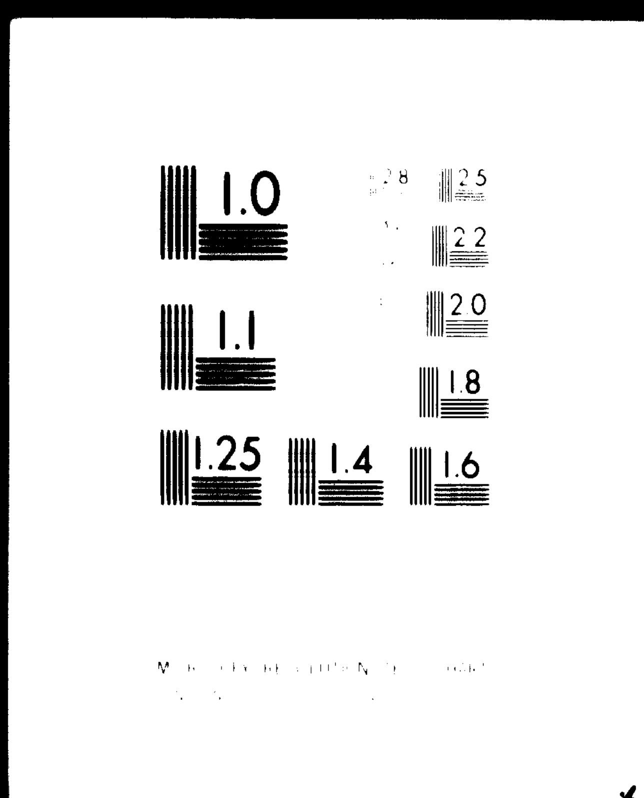

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ASSISTANCE IN THE COMPLETION AND COMMISSIONING OF ETIBANK'S ERGANI COPPER SMELTER EXPANSION

report to

UNITED NATIONS
INDUSTRIAL DEVELOPMENT ORGANIZATION

MARCH 1972

Arthur D. Little, Inc.

UNIDO CONTRACT TURKEY 70/78

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PRINCIPAL PARTICIPANTS IN THE WORK

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I. SUMMARY

A. PURPOSE AND SCOPE

In response to a request by the United Nations Industrial Development Organization (UNIDO), Arthur D. Little, Inc. (ADL) has completed a program of assistance and guidance to Etibank, the minerals development organization in Turkey, with respect to the modernization of the copper production facilities (Ergani Copper Works) at Maden, near Elazig, in Turkey. The modernization included the installation of a fluidized-bed concentrate roaster, a reverberatory furnace to produce copper matte, new converters to convert matte to blister copper, and a new sulfuric acid unit to utilize by-product sulfur dioxide. This installation will substantially replace the existing water-jacketed blast furnace equipment in producing a major portion of the plant output. The use of the old equipment is contemplated only for smelting a small quantity of high grade lump ore.

The program of work for ADL was initially concerned with assistance in the completion of the construction program and later, after a one-year interval, with assistance in the initial operations of the modernized facilities. ADL's program of work concerned the following tasks:

- To identify existing problems with respect to completing construction work on the new reverberatory furnace, and outline solutions;
- to review the design of the fluidized bed concentrate roaster and the reverberatory furnace, particularly with respect to the incorporation of modern practices, techniques, and procedures for the production of matte and outline improvements that were not foreseen in the original design;
- to assess the adequacy of the functions and operations of the concentration and roaster units feeding the reverberatory furnace and also the units further treating the matte produced to blister copper;
- to advise and guide the operating staff in utilizing the roaster and reverberatory furnace equipment to obtain maximum output and quality of product, consistent with efficient operations and reasonable operating costs;
- to review the production costs being experienced with the view toward identifying opportunities to reduce them;
- to train local counterparts to the maximum extent practical while performing the tasks listed above; and

- in conjunction with Etibank, to develop an action plan in which the problems identified are outlined, the interim actions already taken during the course of the work are reported, the specific actions still needed are recommended, and a reasonable schedule of implementation is suggested.

B. APPROACH

The program of work was accomplished by two visits to the smelter; the first from January to mid-February 1971, and the second in January 1972. During the first visit, the plant was nearing the final stages of completion. At that point in time, several major construction tasks had not been completed such as installation of the waste-heat boiler, electrostatic precipitator, by-pass flue, acid plant, materials handling facilities and instrumentation. During the second visit, the new plant was in operation except for the fluid-bed roaster and the sulfuric acid plant.

During the first visit to the field, we reviewed the existing problems with respect to the completion of the smelter. Also, we reviewed the design of the overall plant as well as of the individual units, and suggested several changes which have now been incorporated in the plant. We undertook the major effort in training during the first visit. We conducted separate lecture sessions over a two week period for the graduate engineers and for the operating staff. Actual plant demonstrations on the use of equipment were included wherever possible.

In January 1972, during the second trip to the field, we found the reverberatory furnace on an uncalcined (green) charge. The acid plant was almost complete and could have been started up within a month. Fluid-bed roaster operation had not been started, and the concentrator operation had not yet been normalized. Accordingly, a major part of our effort in the second field visit was directed towards assistance in the start-up and operation of the fluidized bed roaster (a prerequisite to providing adequate feed gas to the acid plant) and to the training of the local staff in this area.

During both our visits to the field, we worked closely with Etibank staff and a counterpart who had been designated by Etibank as a member of our project team. As a result, Etibank was kept informed of our activities and the development of the action plan, and accordingly gained a continuity of experience that they should find useful in implementation of our recommendations.

C. FINDINGS

Our review of overall design of the new plant at Ergani is presented in detail in Chapter III. It shows that the design is appropriate to the feed-ore characteristics and to the production of blister copper, and is in keeping with modern practice in most cases. The major deficiency in the overall plant design, for which corrective action is no longer practical, is related to the low headroom provided in the new converter aisle, which is a consequence of having installed the track for the new overhead crane to connect with that for the old converter aisle.

Several minor deficiencies in the design still exist. The correction of these would facilitate plant operations in terms of increasing operating convenience, of increasing copper recovery, and of reducing operating costs. Certain changes, suggested to the plant management for corrective action during the two field trips have already been incorporated in the plant. A potentially serious deficiency lies in the present design of the slag launders. Access is not convenient and this could lead to a lack of control of tapping rate.

A major factor that caused the delay in the completion of new plant was the installation of a waste-heat boiler of incorrect design, and this required extensive modifications in the field after the boiler had been erected. Another factor was the omission from the original design of a flue by-passing the boiler, but which was corrected later at Etibank's request. The reason for the original omission is not clear.

The new plant is now in operation, except, as of January 1972, for the fluid-bed roaster and the acid plant. Normal manufacturer's assistance for roaster startup is not available because the firm concerned is no longer in business. Acid-plant startup should await the roaster start-up because this provides a major portion of the SO_2 gas stream. Acid plant startup beforehand may be practical, should this become desirable because of the presence on the scene of a UNIDO expert. Successful start-up of the two units in the short term is urgent, since the economy of copper production from the modernized plant is based largely on the operation of these two units.

Once all units are in continuous and controllable operation, the need will arise to adjust, or normalize, the operating conditions so that the equipment performs to its design specifications. The time required for normalization to occur depends on the situation in a particular smelter. At Ergani, the operating staff is faced with almost an entirely new approach to blister copper production, and requires the time to gain the necessary experience. With management support, a minimum period of six months is probably required after start-up of the roaster and acid plant for operation to become normalized.

After normalization, a need should arise to achieve a gradual improvement in the operation of all sections of the plant in order to achieve easy operation, avoid unnecessary materials losses, and decrease operating costs. The installation of a metallurgical and cost accounting system for the entire plant will be the primary management tool to accomplish this task. An evaluation of the plant operating experiences at this point might indicate the desirability of making further changes or equipment additions to the plant.

The details of the three tasks, i.e. startup of the remaining units, normalization of operations, and optimization of operations can be organized into a logical scheduled program of work for implementation, i.e., into an Action Plan. The accomplishment of such an action plan in our view is important, if not vital, if Etibank is to realize a maximum return from the investment in the modernized smelter at Ergani.

D. THE ACTION PLAN

We recommend that Etibank implement the short-term tasks in the action plan outlined below in order to achieve acceptable operation of all the component units of the modernized plant.

On reaching this goal, or possibly even before, we recommend further that Etibank begin a program of work based on the tasks listed in the action plan outlined below as intermediate-term tasks.

Finally, on achieving normalized operation, we recommend that Etibank consider a program of work to optimize the operations based on the tasks listed in the action plan outlined below as long-term tasks.

1. Short-Term Tasks

a. Roaster: The following alterations or modifications would be necessary prior to the start-up of the roaster.

- Install an automatic shut-off for main blower, if bed temperature exceeds 1000°C; or such a temperature that bed sintering can occur;
- simplify compressed air supply to avoid uses of air outside the roaster area;
- relocate orifice plate to measure only the fluidizing air;
- install gate valve in series with butterfly valve to control fluidizing air;
- repair roaster refractory lining to eliminate leakages;
- centralize all operating information in control room by setting up communications with various levels in the roaster installation;
- install U-tube manometers to back up pressure gauges;

- install differential pressure manometer;
- install purge lines on all pressure taps;
- install flow indicators on purge lines;
- install vibrators and/or air lances on calcine bin; and
- hard-surface the elbow in the Wagstaff gun.

b. <u>Reverberatory Furnace and Converter</u>:

- Alter slag launder;
- provide instrumentation for sampling and analysis of reverb off-gases;
- install tight-fitting hoods or aprons near the mouth of the converters to prevent excessive dilution of the SO_2 content of the off-gases;
- provide access to clean converter flux charging chute; and
- provide automatic turnout on the converters.

c. <u>Acid Plant</u>:

We suggest that Etibank should consider starting up the acid plant using converter gases alone i.e. before successful roaster operation is achieved, in order to utilize the assistance of the UNIDO expert.

2. Intermediate-Term Tasks

a. <u>Concentrator</u>:

At present, the concentrator is performing below par. The operating parameters in the following areas might be significant in the performance of the concentrator, and might be of possible interest to the UNIDO expert currently available for assistance to Etibank.

- Dust control to the crushing plant;
- frequent equipment failures;
- oxidized copper minerals; add Na_2S;
- slimes; deslime prior to rougher flotation;

- ball mill undercapacity; monitor power consumption, change maximum ball size;

- insufficient flotation time;

- improper reagents and dosages; and

- avoid shipping dry pyrite concentrates because of fire hazard.

b. Roaster: Normalization problems in the roaster usually involve materials handling problems, for example:

- Slurry preparation, handling, pumping; change nozzle location;

- bridging of calcine in cyclones; mechanical vibration, poke-holes;

- abrasion in cyclones; ceramic lining;

- pressure balancing system inoperable;

- fluoseal clogging; change bottom plate; change to rotary seals; and

- improper diameter of fluid bed drain; change diameter, air lance.

c. Reverberatory Furnace and Converter: The possible problem areas in the reverb section would involve:

- Excessive fuel consumption;

- charging problems;

- excessive refractory consumption;

- magnetite buildup; and

- slagging in the waste-heat boiler.

On the converter side, a major problem might be encountered in retraining the staff and changing previously established practice to control blowing in response to acid plant requirements, to improve scheduling, and control flux additions.

3. Long-Term Tasks

Ultimately the installation of a well-enforced metallurgical and cost accounting system will be vital regardless of the specific results of the short- and intermediate-term tasks. We strongly recommend that planning of such a system begin as soon as practical. In addition, we suggest giving consideration to the following to improve the plant operations:

- Automatic sampling in the concentrator;
- design larger and deeper moulds for blister casting;
- install running time meters on converters;
- investigate possibility of doubling ladle size;
- improve cementation procedures;
- consider the installation of a pyrite roaster for the acid plant to increase acid production; and
- consider the economics of smelting the high grade ore in the new plant versus smelting it in the old plant.

II. INTRODUCTION

A. PREVIOUS OPERATIONS AT ERGANI

The copper deposit at Ergani is a massive sulfide deposit of the hydrothermal type. The predominant sulfide minerals are chalcopyrite ($CuFeS_2$) and pyrite (FeS_2). The host rock is a diabase or peridotite in the lower sections of the deposit and a serpentine or limonite in the upper sections. From a processing standpoint, the deposit contains two types of ore; a high-grade pyritic ore which can be smelted directly, and a low-grade impregnated ore, which requires concentration. At the end of 1971, the proven reserves were 12.8 million tons with an average grade of 1.72% Cu. The existence of this deposit was known during prehistoric times and evidence has been found of ancient small-scale mining and smelting activity in the area.

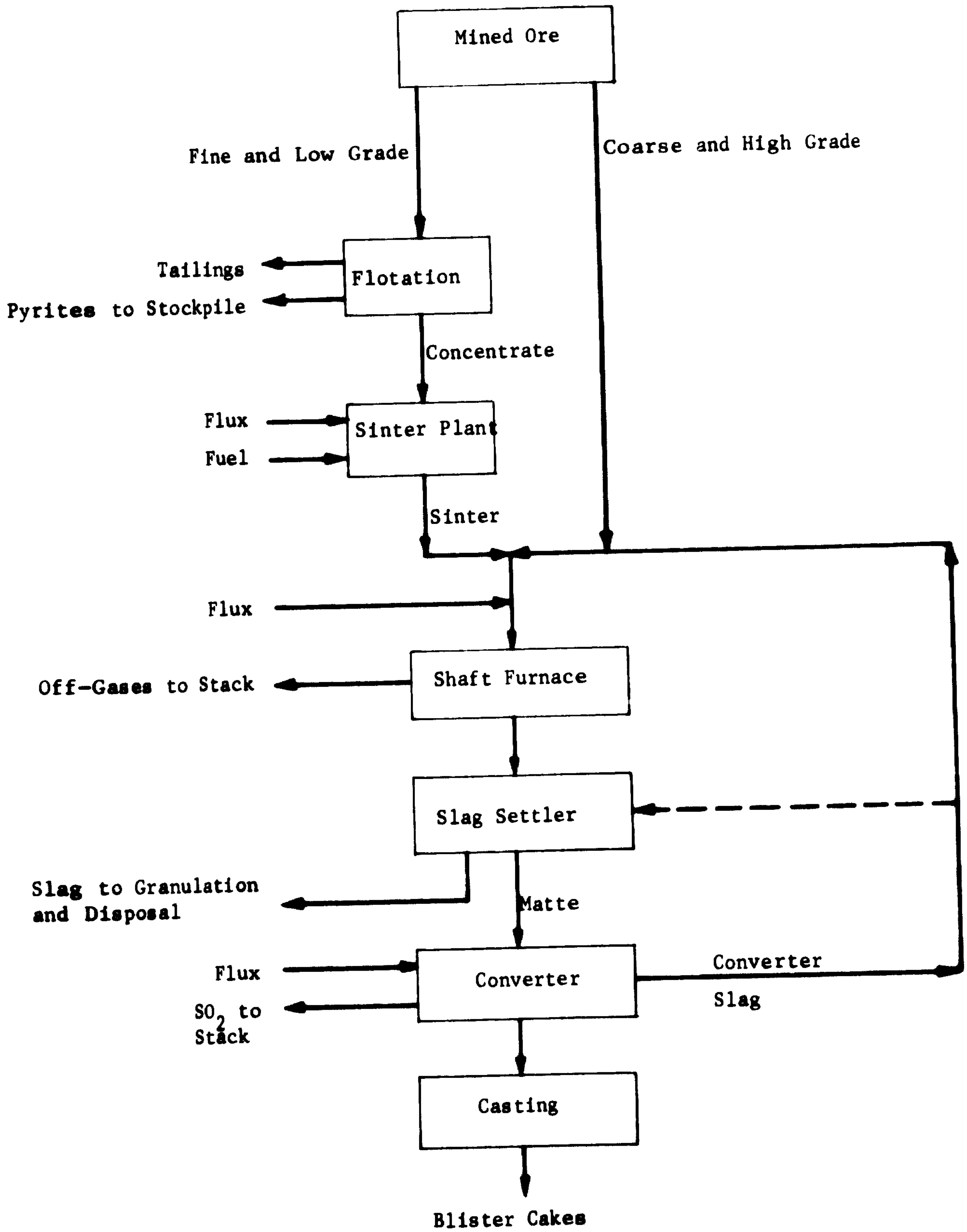
In 1939, a metallurgical plant was constructed at Ergani by a German firm, which was based on the direct smelting of coarse ore, mixed with fluxes and coke, in rectangular water-jacketed shaft furnaces. Initially, the small amount of fine ore available was agglomerated by sintering and fed to the shaft furnaces with the coarse ore. The molten discharge from the shaft furnace was separated into a slag layer and a matte layer in a slag settler; the average grade of matte obtained being 30-40% copper. The matte was converted to blister copper in conventional Pierce-Smith converters.

The original plant was based on the ability of the mine to supply +1-inch ore containing 7% copper. As the ore grade declined with time and as the percentage of fine ore being fed to the plant increased, it was found necessary to beneficiate the ore by flotation and to sinter the concentrates. Accordingly, a 350 tons of ore/day flotation plant was built in 1950. The capacity of this plant was subsequently increased to 750 tons/day in 1955. The flotation plant also produced a by-product pyrite concentrate which has been stockpiled. The fine concentrate was mixed with fuel and flux and sintered.

Before the expansion project just completed was undertaken, the physical plant at Ergani comprised of the 750 tons/day flotation mill for treating fine low grade ore, a sinter plant for agglomerating the flotation concentrate, and a smelter containing two water-jacketed shaft furnaces for smelting sintered agglomerates and coarse ore, along with slag settlers, slag granulation, auxiliary off-gas handling facilities, and four Pierce-Smith converters for converting the matte produced to blister copper. Dust in the gases from the shaft furnaces was removed in a settling chamber after which the gases were vented to the atmosphere via a gallery and a flue. Converter gases passed through a balloon flue before mixing with the shaft furnace off-gases. Cakes of blister copper were cast from ladle and shipped. A schematic diagram of the plant just before modernization is shown in Figure 1.

FIGURE 1

SCHEMATIC FLOWSHEET OF THE PLANT AT ERGANI BEFORE MODERNIZATION



The decision to modernize the smelter was a result of the declining grade of ore which led to lower matte grades and decreased copper production from the existing equipment, and the increasing amount of low grade ore which has to be beneficiated by flotation and sintered. The decrease in copper production resulting from these factors, and in later years from interference by plant modernization activities, is seen in the production statistics of Table 1.

The decision to modernize the Ergani copper works was taken in about 1956. By about 1961, the mining plan had been modernized and was capable of supplying the larger concentrator. The final selection of a processing flowsheet occurred after 1962 when it was decided to build a conventional reverberatory-type smelter capable of producing 18,000 tons/year of blister.

B. THE MODERNIZED OPERATIONS

The present smelter complex comprises the following units and facilities, shown schematically in Figure 2.

- A new concentrator;
- A fluidized bed roaster for roasting the concentrate;
- A deep bath reverberatory furnace fired with fuel oil which has the dual capability of being fed with either the roasted calcines via a Wagstaff gun or with moist concentrate via slinger belts;
- A waste heat boiler for recovering waste heat from the reverberatory furnace gases;
- An electrostatic precipitator for dust removal from the gases after cooling in the waste-heat boiler,
- Two new converters of the Pierce-Smith type; and
- An acid plant for manufacturing sulfuric acid from the off-gases.

Although the new smelter is located adjoining to the old smelting plant, the new plant is essentially self-contained and minimal use is made of the older facilities. The new plant uses the old flue, gallery, and chimney for conveying the cleaned reverb off-gases to the atmosphere. The same installation can also exhaust roaster or converter gases in case the acid plant is shut down. The new converter aisle is a continuation of the old converter aisle and, if necessary, cranes installed in the older section can travel and be used in the new section. However, the new cranes would be adequate during normal operations. Figure 3 is a schematic-pictorial diagram

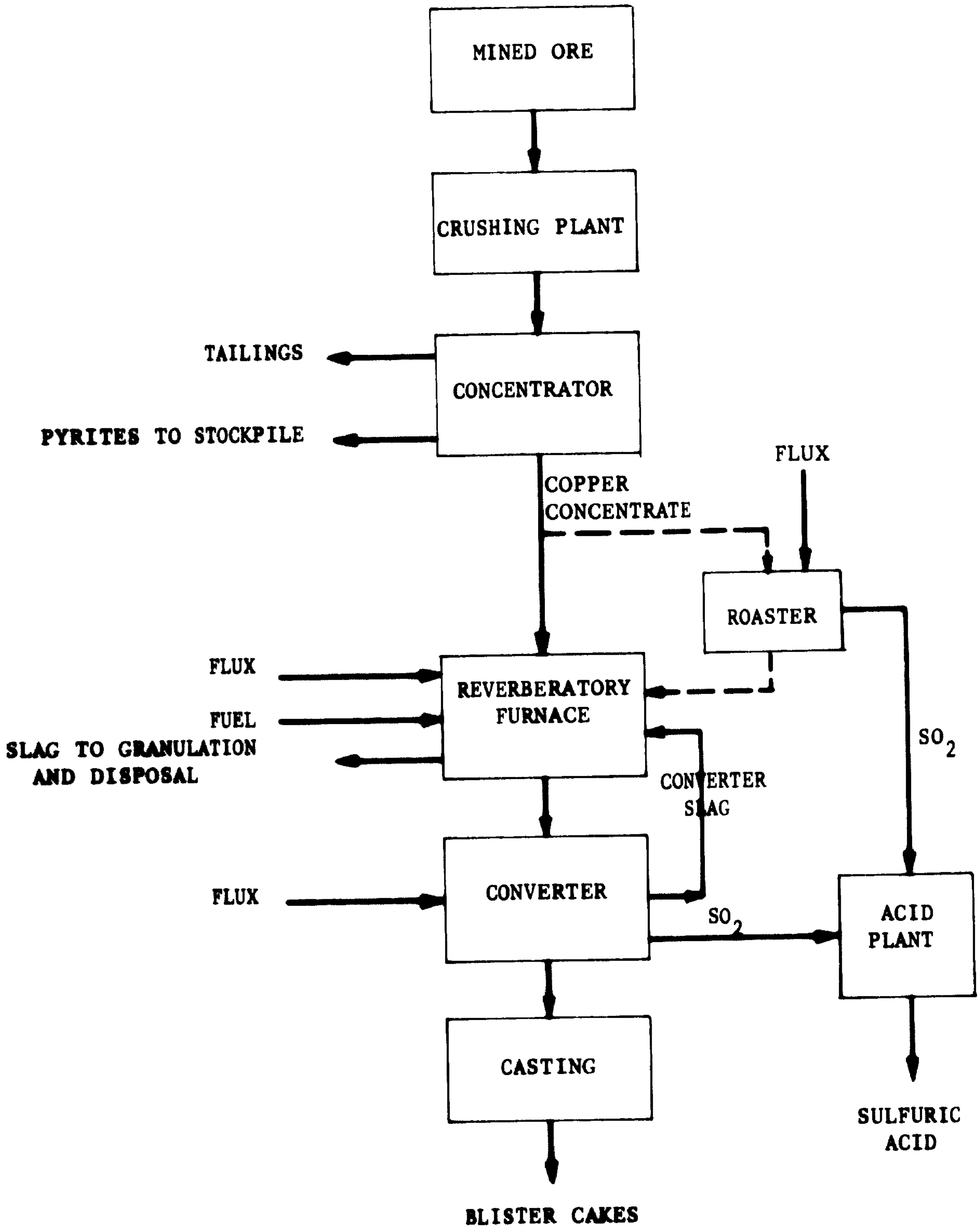
TABLE 1

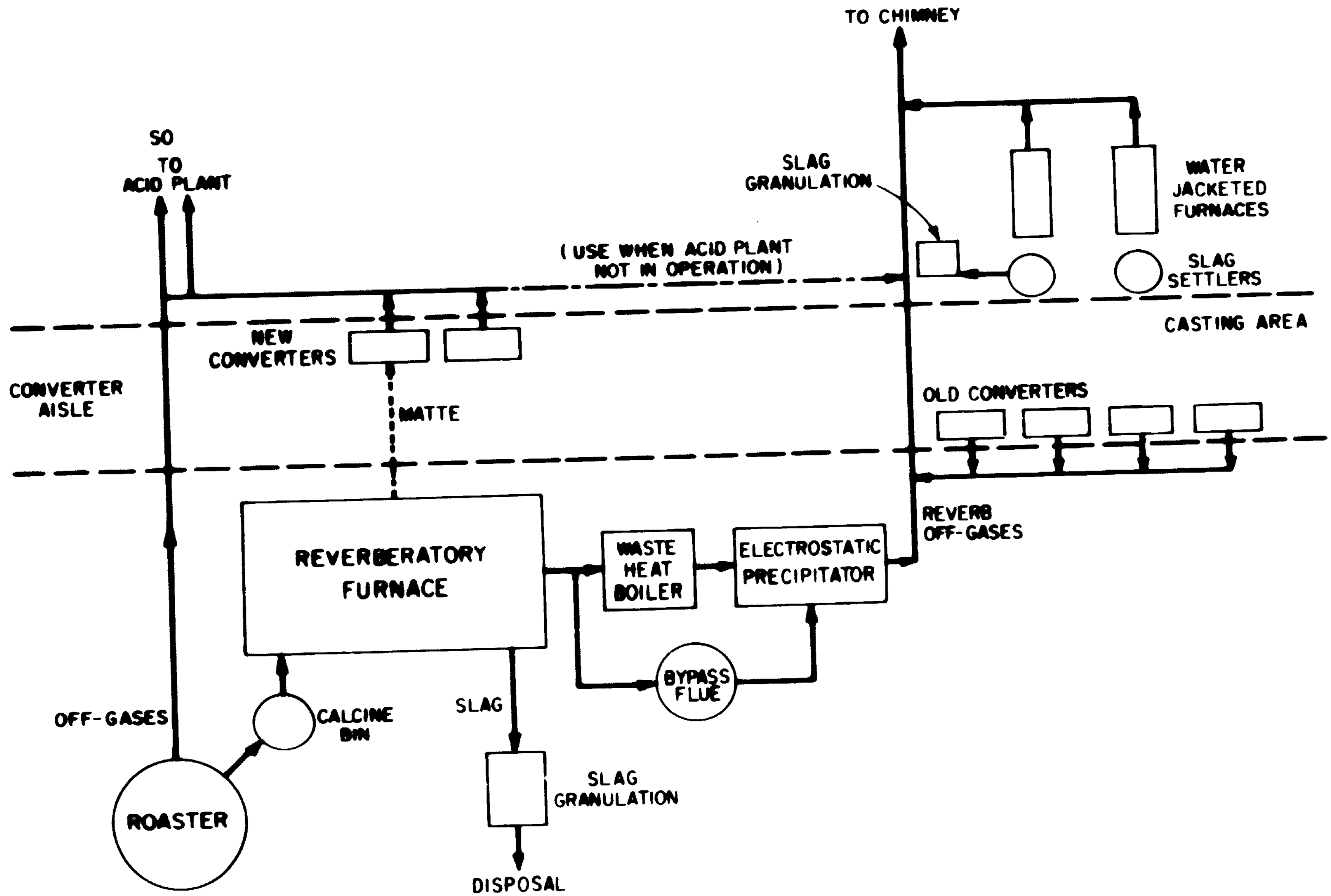
PRODUCTION STATISTICS FOR THE ERGANI SMELTER

Annual Blister Production (metric tons)	Year
17,517	1966
15,750	1967
14,000	1968
10,470	1969
9,540	1970

FIGURE 2

SCHEMATIC FLOWSHEET OF THE NEW PLANT AT ERGANI





SCHEMATIC DIAGRAM SHOWING RELATIVE POSITIONS OF INDIVIDUAL UNITS IN OLD AND NEW SECTIONS OF THE SMELTER

FIGURE 3

that shows relative positions of equipment in the old and new plant.

We understand that the overall construction program was managed by Etibank with the help of the Parsons-Jurden Corporation of New York, who acted in the capacity of a consultant. Furthermore, Parsons-Jurden was responsible for the overall plant design, provided detailed specifications for major items of equipment, and provided assistance in the start-up of certain units such as the reverberatory furnace. Based on the detailed specifications, various units in the plant were supplied by subcontractors from several countries. For example, the concentrator and waste heat boiler were supplied by Turkish companies, the roaster by a British firm, and the acid plant by an Italian firm.

III. THE SMELTER MODERNIZATION AND EXPANSION PROJECT

A. REVIEW OF DESIGN

In the discussion below, we identify elements in the design of the plant that generate questions of their appropriateness and areas where there is still room to incorporate newer developments (in equipment or in operating practice). For sake of clarity, we have included a short description of each component unit in order to frame the prospective problem areas in equipment or in operation that we identified.

1. Process Plant Component Functions:

a. Crushing Plant and Concentrator

In the crushing plant, the mined ore is crushed dry to a size where it can be reduced further by wet grinding in the concentrator. In the concentrator the ore is wet ground to "liberate" (or separate) the grains of sulfide copper-bearing minerals from the siliceous host rock (gangue). At Ergani, this liberation occurs at a nominal grind of minus 65 mesh, when most of the individual particles are either sulfides or gangue material. The minus 65 mesh discharge from the ball mill-classifier circuit is treated by bulk sulfide flotation in order to separate and reject the worthless gangue in the ore which is then discarded as plant tailings. The bulk sulfide concentrate thus obtained is ground further to a nominal minus 200 mesh in order to separate the grains of pyrites from the grains of chalcopyrite. The resultant pulp is treated by selective flotation, during which a high pH of about 11.5 is maintained. This renders the pyrite nonfloatable and separates it from chalcopyrite, the predominant copper mineral at Ergani. The chalcopyrite and pyrite concentrates are separately thickened and filtered. The chalcopyrite concentrate can be fed either directly to the reverberatory furnace when utilizing green-charge smelting or to the fluid-bed roaster for partial roasting when using calcine smelting.

b. Fluidized-Bed Roaster

In the roaster, about 40% of the sulfur in the concentrate is eliminated by oxidation. This leads to higher matte grades in the reverberatory furnace. The roaster off-gases provide a steady, concentrated stream of SO_2-rich gas ideally suited for sulfuric acid manufacture. In a fluidized-bed roaster, particles of silica flux are fluidized with air into which a slurry of chalcopyrite concentrates in water is injected. Oxygen in the fluidizing air reacts with sulfur and iron in the concentrate to form sulfur dioxide (SO_2) and magnetite (Fe_3O_4) respectively, the copper remaining as the sulfide. The heat generated by these oxidation reactions is taken up by the vaporization of the water component of the slurry so that the roaster operates essentially at a constant temperature (about 600°C). The bulk of the roasted concentrate or calcine leaves the reactor with the off-gases and is recovered in primary and secondary cyclones. The coarse material in the reactor drains from the bottom. Both are stored in a calcine bin from which they can be charged intermittently into the reverberatory furnace as required.

c. Reverberatory Furnace

A reverberatory furnace is essentially a rectangular box of refractory which is heated by burning fuel. The reverberatory furnace at Ergani is of the deep-bath type and uses residual fuel oil as fuel. A deep-bath type furnace contains a large amount of liquid material at all times in its crucible. The floor of the crucible is kept cool and "frozen" by air pipes embedded in it, while the sides are protected a a series of water jackets that cool and protect the refractory that is in contact with the molten charge.

The molten charge in the reverberatory furnace consists of a mixture of molten sulfides (essentially Cu_2S and FeS and referred to as matte), as the lower layer, and a lighter layer of slag. Slags in copper smelting are typically of the fayalite composition; that is, $2FeO \cdot SiO_2$. The off-gases from the reverberatory furnace are cooled in waste-heat boilers, treated in electrostatic precipitators for dust removal, and vented to the atmosphere.

A reverberatory furnace performs two functions. First, it melts the solid charge which then separates to form liquid matte and slag layers. When using green charge smelting, the charge normally contains an excess of sulfur. During matte formation, all the iron in the concentrate reports to the matte which can lead to the production of low-grade (low percentage of copper) mattes. When using calcine smelting, there is a deficiency of sulfur in the charge and some of the iron will report to the slag layer thus resulting in a high-grade matte. Second, the reverberatory furnace treats the viscous, high-magnetite, high-copper slags produced in the converter for additional copper recovery.

The reverberatory furnace slag is tapped intermittently, granulated with water and discarded. Matte from the reverberatory furnace is tapped intermittently into ladles and then fed into a converter with the use of an overhead crane.

d. Converter

A converter transforms matte to iron oxides (which report to the slag) and to metallic copper. The off-gases can be rich in SO_2 and accordingly can be used for sulfuric acid manufacture. A converter is a barrel-shaped vessel with an opening (mouth) near its top and a row of tuyeres along its side. The converter can be rotated to pour out its contents. Converting is a batch operation. Initially, a converter is charged with several ladles of matte from the reverberatory furnace. Air flow through the tuyeres is commenced and the converter is tilted so that the tuyeres are submerged by the matte. Oxygen in the air converts FeS in the matte to FeO while the Cu_2S remains unaffected. The oxidation reactions generate sufficient heat to keep the charge molten so that external heat is not required. Silica is added to the converter to form a slag with the FeO and to keep the iron oxide in its lower oxidation state. The latter, however, cannot be achieved completely so that converter slag is always high in magnetite, Fe_3O_4. The blowing of air is continued until the matte layer is essentially Cu_2S. Then all the slag is skimmed off by pouring and charged back into the reverberatory furnace. This phase of converter operation is referred to as the "slag blow".

After several slag blows during which additional matte is charged and iron is removed in slag, sufficient Cu_2S (or "white metal") is accumulated so that it can then be blown to metallic copper during a "copper blow". The sulfur in Cu_2S is eliminated at this stage.

At Ergani, the metallic copper from the converter is poured into ladles and from which it is cast directly into (blister) copper cakes for shipment.

e. Acid Plant

The combined off-gases from the roaster and converter are rich enough in SO_2 to be treated in the contact acid plant for the production of sulfuric acid.

2. General Layout

The terrain at Ergani is mountainous and only a limited amount of level space adjoining the existing water-jacketed furnace installation was available for the new plant. Considering the level space constraint we believe that the new plant has been constructed in the proper location and all material flows, in and out of the plant, are properly laid out with the exception of the height of the overhead crane. The overhead crane is responsible for all material flows in the converter aisle and performs a number of major functions.

- Matte from the reverberatory furnace is tapped into ladles on one side of the converter aisle which are then carried across the aisle and discharged into the converter.

- The slag from the converter is poured into ladles and charged back into the reverberatory furnace.

- When converting is complete, copper is poured from the converters into ladles and taken to the blister casting area. At Ergani, the facilities for casting blister cakes are at one end of the converter aisle.

The converter aisle in the new plant is an extension of the converter aisle in the old plant, and as a result, the track for the overhead crane in the new section is an extension of the older track. The height of this track is too low when compared to modern practice, and this results in a number of disadvantageous consequences.

- The low height of the overhead crane limits the capacity of the ladle that can be used to transfer matte or slag. As a result, we found that an unusually small size of ladle has to be used at Ergani. For example, the converter operating schedule suggested by Parsons-Jurden calls for at least two ladles of matte to be charged when the converters are turned down to accept matte. If the overhead crane were higher, one ladle of twice the capacity could be used once, thereby reducing heat losses and the amount of skulls, the frozen material sticking to the sides of the ladle which has to be chipped out in order to be recycled.

- A pit in the floor is necessary in front of each converter to hold the ladle so that the molten charge in the converter can be poured into it. Modern practice has evolved away from the installation of pits by raising both the overhead-crane track elevation and the level of the converters themselves. Most converters have an automatic safety mechanism so that in case of power failure (and stoppage of air supply) the converter can be turned down and emptied or at least turned down enough to bring the tuyere openings above the bath to prevent freezing. If the converter charge is thus dumped into a pit, it is much harder, if not impossible, to recover especially if the dumped charge is malleable metallic copper rather than brittle matte.

- Additional avoidable lifts can also decrease plant throughput and the availability of the overhead cranes for other purposes.

The obvious remedy here would be to increase the height of the overhead crane track and also the height of the converters. Since both these changes are very expensive in terms of cost and loss of production, we believe that they could not be justified by the extent of possible improvement in operations. But, since the new overhead crane is capable of lifting twice the weight of the current ladles, we suggest that Etibank investigate the possibility of designing a ladle twice the present size which could be used within the limited headroom.

3. Receiving and Storage Facilities

The facilities for receiving and storage of materials such as flue, flux, spare parts and miscellaneous maintenance materials are adequate. During the start-up of the new plant, spare parts for certain items, especially those required in the concentrator (crusher machinery, screens, pumps, etc.), were not available. But, such problems are to be expected during initial start-up operations. However, once the plant operation is normalized, we suggest that careful maintenance and replacement records be kept. Then, an appropriate maintenance and replacement policy can be worked out by the plant management, and measures necessary for maintaining a proper inventory of spare parts can be implemented. In such a policy formulation, the fact that many parts are imported, and delays in shipments can be beyond the control of the plant management, should be taken into account.

4. Individual Units

a. Mining

A detailed study of the mining plan, mining equipment and reserves was outside the scope of this effort. However, based on detailed discussions with the plant management, we believe that the mine is capable of supplying the necessary tonnage to the crushing plant and concentrator.

b. Concentration

The flowsheet selected for the crushing plant and concentrator is typical of flowsheets used for treating ores of this type. The crushing plant has three stages of crushing--the primary and secondary stages being in open circuit, and the tertiary stages in closed circuit. The crushed ore is stored in fine ore bins. In the concentrator, the crushed ore is wet ground in one of two rod mills in open circuit. The rod mill discharge is then ground by three ball mills in closed circuit. The resultant slurry is then treated by flotation.

The flotation lines consist of four banks of "rougher" cells each containing ten cells. The rougher concentrate is treated by one bank of "cleaner" cells, whose concentrate is reground prior to selective flotation in order to separate the pyrite from the chalcopyrite. Selective flotation is achieved by maintaining a high pH (about 11.5) which depresses the pyrite. The selective flotation section consists of two parallel lines of cleaning and recleaning stages and a scavenger stage for decopperizing the pyrite tails. The copper concentrate and the pyrite concentrate are thickened separately and filtered.

We believe that the design of the crushing plant and concentrator is basically sound and we have no major recommendations regarding changes in this area. We believe it useful to make suggestions and observations as follows for consideration by the UNIDO expert who is now (March 1972) in Turkey with a mission to advise the Ergani plant on the flotation of the copper concentrates, and who is expected to complete his mission by December 1972.

- The crushing plant is dustier than is usual in installations of this type. This dust should be controlled by water sprays or by properly ventilating the crushing plant. In the meantime, the personnel in this section of the plant should be provided with respirators for their protection.

- Based on exploratory calculations, it appears to us that the three primary ball mills in the grinding plant might be undersized and might not be able to handle the rated tonnage of the plant and produce the required mesh of grind. These calculations, however, are based on handbook data and are quite approximate (perhaps an error of ± 20%). Therefore, whether or not the primary ball mills constitute a real bottleneck can be proven only after operating the concentrator in a normal fashion over a period of several months. Should the ball mills prove to be a capacity bottleneck, some relief can probably be obtained upstream by crushing the ore to a finer size in the crushing plant. However, this might require additional screening equipment.

- In the operation of the ball mills, the total power drawn by each mill should also be monitored as a function of the feed rate. This would enable the operating staff to determine the maximum capacity of each ball mill and achieve the maximum throughput through them, should they be found to be a bottleneck.

- In the concentrator, the design shown on the flowsheet does not allow for automatic sampling for metallurgical accounting and for process control. A certain amount of automatic sampling equipment might improve the sampling procedure and hence the ability of the operating staff to improve the control of the plant operations.

- We notice that the plant maintains an inventory of steel balls of various sizes for use as make-up. In an operating ball mill, the ball sizes quickly reach an equilibrium size distribution and it is necessary only to replace the largest sized balls as these are worn down. If only the largest sized balls are used for make-up, unnecessary supply and inventory costs would be eliminated or decreased.

During our second visit to the plant (in January 1972), the crushing plant and concentrator were in operation, but the operation had yet to be normalized. The most obvious problem at that time was extremely high wear rates on certain equipment and the lack of spare parts for replacements which led to frequent plant shutdowns, so that the circuits rarely had a chance to reach equilibrium. However, even when allowance is made for the effects of frequent shutdowns, it appears that copper recoveries have been low (a high loss in the tailings) and selective flotation has not produced concentrates of adequately high copper content.

In many instances, it appears that plant performance (i.e., copper recovery) improves significantly at low throughputs, which indicates that one or more of the equipment components may be overloaded when the plant operates at its rated throughput. The plant has been fed with high grade ore in order to obtain a reasonable volume of concentrate but obviously, this practice cannot be continued very long without incurring a serious loss of ore reserves.

We understand that representatives of the sub-contractor to Etibank, who installed the concentrator, were unable to suggest changes to bring the performance of the concentrator up to requirements. As noted above, a UNIDO expert in mineral beneficiation is in Turkey until December 1972 to assist Etibank with its concentrator operations and we would expect that considerable attention is now focused on this problem. We would suggest that the first step in improving concentrator performance should be the identification of the specific problem areas. The procedure we suggested to the plant management should still be applicable. It is based on our suspicions that the specific problem areas are one or more of the following:

- A high frequency of mechanical failures in the equipment which leads to frequent shutdowns;

- The presence of oxidized ore which would require the addition of sodium sulfide to improve flotation recoveries;

- The presence of slimes which would require desliming cyclones prior to rougher flotation;

- Ball mill under-capacity which may result in inadequate grinding;

- An inadequate retention time in flotation cells, which does not permit thorough separation; and

- A non-optimum choice of flotation reagents and pH regimes for bulk sulfide flotation and selective chalcopyrite flotation.

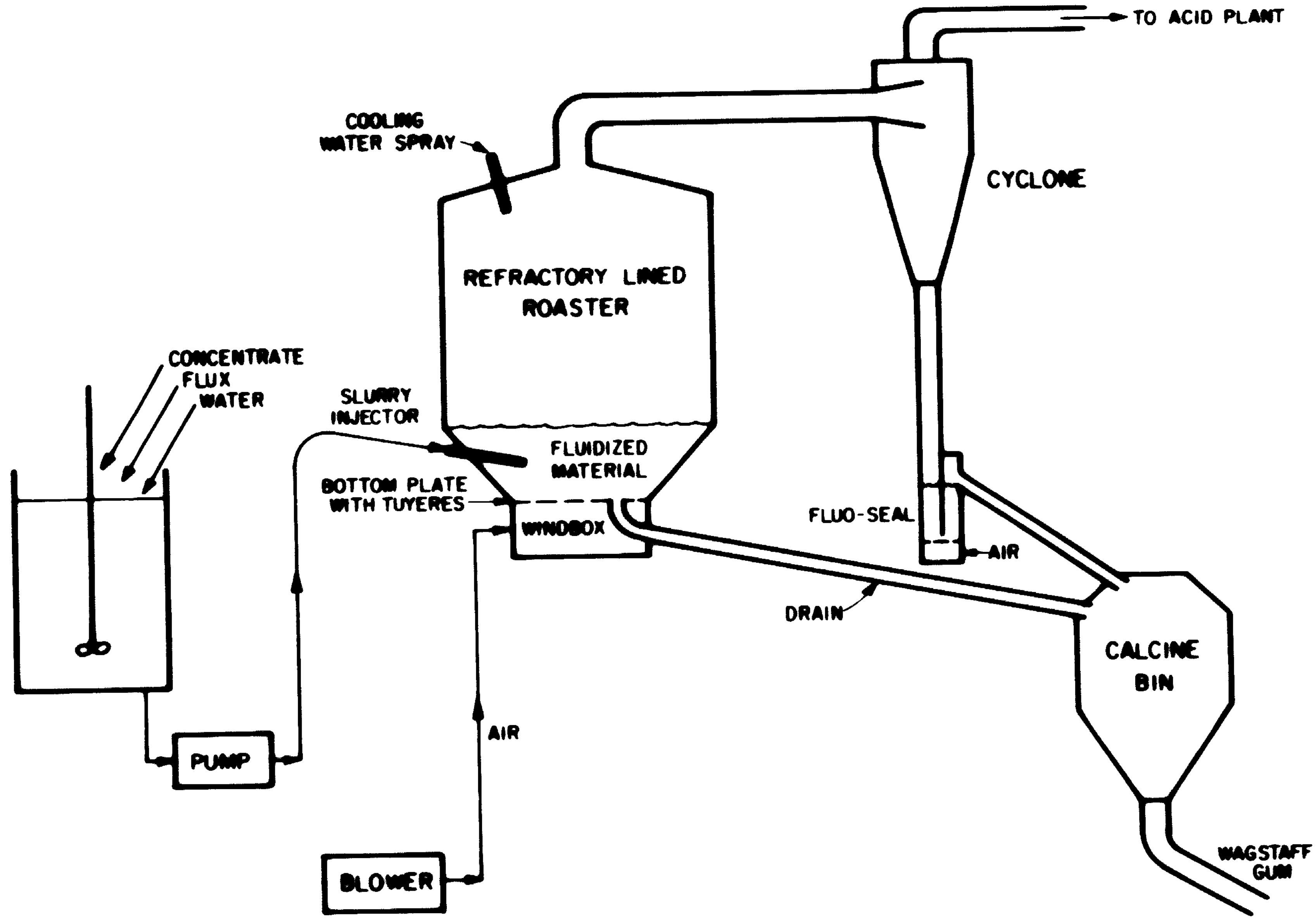
The procedure we suggested was to take two identical samples of mill pulp, subject one to the standard flotation treatment in the laboratory flotation cell at Ergani (the "control" sample) and subject the other sample of pulp to a specific treatment program involving desliming, sodium sulfide treatment, additional grinding, etc., in order to identify the significant variables affecting the copper recovery in the flotation plant.

c. Fluid Bed Roaster

The fluidized bed roaster is a critical part of the physical plant at Ergani, because the concentrate grade is normally fairly low and a certain degree of sulfur elimination is necessary in order to produce a high-grade matte. Also, partial roasting produces a steady, concentrated stream of sulfur dioxide which is ideally suited to sulfuric acid manufacture.

Figure 4 is a schematic-pictorial diagram of the important components of the roaster installation at Ergani. It consists of a reactor vessel in which minus 28-mesh silica (flux) is fluidized by injecting low pressure air from the windbox below via tuyeres. When properly fluidized, the sand acts very much like a boiling liquid and is characterized by "liquid" properties such as possessing a hydrostatic pressure.

The concentrate filter-cake is reslurried in water, mixed with about 10% by-weight of coarse silica flux and the slurry is injected into the fluidized bed reactor. Within the reactor, the oxygen in the fluidizing air reacts with sulfur in the concentrate to form SO_2, and iron is oxidized to Fe_3O_4. Both these reactions



SCHEMATIC-PICTORIAL DIAGRAM OF FLUIDIZED BED ROASTER

FIGURE 4

release heat which is absorbed by the evaporation of the water component of the slurry. Hence, the reactor operates at a constant temperature--at around 600°C.

The fine roasted concentrates are carried with the fluidizing gases out of the top of the reactor, and are separated from the gas stream by two stages of cyclone separators which recover almost 95% of the solids in the stream. The remaining solids are recovered in the scrubbing system ahead of the acid plant and recycled to the slurry tank.

Silica, which comprises the fluidized bed material, drains from the bottom and is combined with the cyclone discharge in the calcine bin. The hot calcines in the calcine bin are fed to the reverberatory furnace on an intermittent basis through a Wagstaff gun, which is essentially a hollow tube with a water cooled end.

At the time of our visit in January 1972, the roaster at Ergani had not yet been operated successfully. We understand that the subcontractor, who supplied the roaster equipment, became bankrupt before the roaster installation was completed and now cannot be relied upon for assistance in starting and commissioning the unit.

In general, the major problems encountered in operating fluidized-bed roasters are related to breakdowns or to bottlenecks in the materials handling facilities on the input or the output side. The plant management, prior to our arrival in the field in January 1972, had made the first attempt to commission the roaster. During our stay, we assisted the plant management in the second attempt. The discussion below is based on these experiences and on an inspection of the design drawing and the equipment installation itself. We cover first potential or real inadequacies in design that might require alterations of equipment after initial operating experience is available. We then discuss the specific recommendations we made to the plant management during our second visit to the field.

The inadequacies in design and equipment are:

- <u>Preparation of Slurry</u> Although the slurry preparation and handling system appears to be adequate, the slurry is supposed to contain approximately 79% solids and this might conceivably cause problems in pumping. On the other hand, based on observations of slurry pump operation at the plant, we would tend to discount potential pump problems, but not potential problems involving clogging in stagnation-prone areas (i.e., blind spots) in piping or high wear at elbows. Our examination of the piping does not reveal any obvious blind spots, and erosion wear data would only become available after the roaster has been in operation for several months.

- Injection of Slurry The nozzles which inject the slurry into the fluidized bed are located below the nominal level of the bed and might consequently suffer from abrasion and/or overheating. If nozzle location proves to be a source of a problem, we would recommend that they be moved so that the slurry is injected almost horizontally between a level in line with the top of the fluidized bed (this would normally be the level where the tapering bottom section of the reaction ends and the reactor reaches its maximum inside diameter) and a level a few inches above.

- Cyclones The primary and secondary cyclones are not lined with refractory. We checked with several U.S. manufacturers of such equipment and found that when cyclones operate up to around 600°C, a refractory lining is not normally used, but is definitely used above such a temperature. Since the operating temperature at Ergani would be on the border line, it is possible that the cyclone life could be shortened significantly by heat damage and erosion. If operating experience shows that considerable abrasion occurs, the cylcones can be lined with a castable abrasion resistant refractory, for example, "Aludur" abrasion resistant ceramic as manufactured by Norton Refractories in the U.S.A.

  There is a possibility of dust bridging and accumulating at the apex of the cyclone above the seals. This could be overcome by installing mechanical vibrators, by banging the cyclones from the outside manually, and/or providing poke holes for cleaning near the apex of the cyclone.

  The supplier of the equipment has installed a rather elaborate automatic pressure balancing device for controlling cyclone performance. We are skeptical about its ability to perform successfully on a continuing basis, and suggest that pressure balancing be done manually during start-up and the early phases of operation. Then the performance of the equipment on an automatic pressure balance basis can be verified.

Each cyclone is provided with a "fluo-seal" at the bottom of its leg. Fluoseals are basically U-tubes whose legs contain calcine fluidized with air. In operation, the U-tube maintains a gas pressure seal like a manometer. The roaster calcine discharging onto the top of one leg creates a hydrostatic head causing solids to overflow from the other. The fluidizing air sweeps the SO_2 gas entraining the discharging calcines back into the roaster; hence, SO_2 does not escape from the bottom of the cyclone along with the calcines. In the operation of the fluo-seals, minimum air should be used for fluidization so as to avoid additional oxidation of the calcine and consequent overheating and sintering.

The fluo-seals were supplied as pre-assembled units by the subcontractor who used a ceramic porous plate for distribution of the fluidizing air. We found that these plates were badly cracked, perhaps during shipment, and these were replaced with an iron plate perforated with small diameter holes that we helped the plant engineers design. This design will have to be verified by its performance during operation. Moreover, fluo-seal valves are conventionally used for handling calcine coarser than that resulting from the Ergani concentrate. Mechanical-type valves, for example, rotary vane feeders or flap valves, are used for fine-sized calcine. Hence, there is a possibility that the fluo-seals will not work properly, especially those installed for the secondary cyclones, and may have to be replaced by a mechanical device.

- Bed Discharge The coarse solids are discharged from the bed through a bottom drain flush with the bottom-plate tuyeres. Such a design might not adequately control the bed height, in which case the installation of a bed overflow pipe would become necessary. Other possible operating problems with such a drain can arise from an improper diameter for the drain.

This can lead to either excessive or insufficient removal of coarse bed material and bridging and hanging at the elbow in the drain may occur. After sufficient operating experience has been accumulated, the drain diameter could be changed, if necessary. If there are excessive problems with hanging at the elbow, an air lance would have to be provided.

Our recommendations to the plant management for changes in equipment prior to the next attempt to start-up the fluid bed unit are:

- <u>Input Air Systems</u> The roaster unit has three sources of compressed air. Low-pressure air is provided by the blower which provides the fluidizing air. High pressure air is provided by two compressors. These compressors are not sufficient for all process air requirements and hence additional high or medium pressure air has to be obtained from other parts of the plant. At the present time, these three supply systems have been interconnected at many points, and are used for purposes other than roaster operation. These interconnections should be eliminated so that the entire compressed air distribution system is simplified and specifically used as a process unit for supplying all of the roaster requirements. We discussed several alternative approaches to achieve this with the plant management. The discontinuation of the second start-up trial before achieving fluidization resulted from the failure of the slurry pumps to operate and it appears that this failure was directly related to problems in compressed-air supply.

  Other changes in the input air systems are:

  - Relocate the orifice plate so that the air entering the fluidized bed roaster is measured. At its present location, the orifice plate measures the total air handled by the blower, some of which is diverted to the oil burner (secondary air) during start-up.

  - Install a gate valve to stop air flow into the bottom of the fluidized bed when this is necessary. At present there is a butterfly valve for this purpose which does not close properly. The butterfly valve, however, should not be removed.

- During the second start-up trial, the blower motor overloaded several times. The reason for this should be investigated.

- <u>Flux Leakage</u> There are several holes in the refractory lining of the roaster through which fine particles of flux leaked out during start-up. These should be plugged with castable refractory.

- <u>Instrumentation</u> Although the roaster installation is elaborately instrumented, it is not adequate for several reasons. The utility of some of the instrumentation (for example, the automatic system for control of cyclones mentioned above) is open to question. Information is not available centrally in the control room but at four different levels in the roaster installation. This information must be channeled to a central point so that quick operating decisions can be made, especially during start-up. We discussed a simple system based on bells and lights, for communicating with these levels using prearranged codes.

  The Ergani smelter does not have adequate facilities or trained personnel for servicing much of the sophisticated instrumentation installed in the control room at the roaster. Because the acid plant also has sophisticated instrumentation, we believe that the full-time services of a properly equipped instrumentation engineer is justified, and will be required once both the units start operations. In the meantime, we suggested using U-tube manometers to back up and verify all pressure measurements.

  During the second start-up trial, a differential pressure manometer was installed to measure the pressure difference between two points under the level of the fluidized bed. Observation of the character of the variations in this differential pressure are a reliable indication of fluidization and readings can also be used to indicate the density of the fluidized bed.

Purge lines were installed on all pressure taps to prevent condensation and accumulation of fine dust in the line. It will be necessary to install flow indicators in each line so that an excessive flow of purge air does not mask the pressure measurement.

- <u>Calcine Bin</u> The calcine bin is potentially a serious bottleneck, since when smelting calcine all the solid charge enters the reverberatory furnace from this one source. The capacity of the calcine bin is twelve tons which means that when the roaster is operating normally, the empty bin would be filled in about 40 minutes. The normal charging rate into the reverb furnace would be about 18 tons per hour, or 9 tons per half hour when charging is done every half hour. Thus, the bin already contains 9 tons when any problems in bin discharging are detected and only 10 minutes are available for achieving discharge before the bin is full and the roaster has to be shut down.

  There are two potential areas where such problems can occur. The calcine bin does not have mechanical vibrators and considerable difficulty might be encountered as a result of hanging at the apex of the bin. We strongly recommend the immediate installation of one or two pneumatic vibrators. The second potential problem area is at the elbow where the vertically descending solids change direction and travel down the sloping Wagstaff gun. Hard surfacing the elbow at this point with a very hard abrasion resistant material plus perhaps provision of air lances would overcome wear and sticking problems in this area. Careful maintenance of the Wagstaff gun equipment is indicated.

- <u>Automatic Shutoff</u> Under normal operating conditions, the roaster feed contains sulfur in excess of that required to combine with the oxygen in the fluidizing air from the point of view of calcine composition. Hence, all the oxygen in the input air is used up and the roaster off-gases contain little or no free oxygen. The easiest method for controlling the fluid-bed temperature after a steady-state operation has been achieved is to <u>increase</u> the slurry feed rate slightly to cool the bed and <u>decrease</u> the feed rate slightly to increase the bed temperature.

The roaster also has an automatic temperature controller which actuates water sprays in the roof of the roaster in order to cool the fluidized bed. This arrangement should be adequate for controlling normal temperature excursions. However, we strongly recommended that in addition to this, an automatic device be installed that shuts off the <u>main blower</u> any time the fluidized bed temperature reaches 1000-1050°C. This would prevent sintering or melting of the bed material in case there is a temperature excursion which for some reason is not controlled by activating the water sprays in the roof of the roaster. With certain concentrates, sticking and defluidization problems can also occur at temperatures around 700°C. These usually relate to the thermal decomposition of pyrite to FeS and elemental sulfur.

d. <u>Reverberatory Furnace and Accessories</u>

As noted earlier, the reverberatory furnace melts the charge which separates into two layers, matte and slag. Converter slag is also charged into the reverb in order to recover its copper content. Molten matte is tapped and converted to produce blister copper, while the slag is discarded. We have comments on certain components of the reverberatory furnace.

The reverberatory furnace at Ergani is of conventional design. An unusual aspect of the design is the fact that it has a dual capability with provisions for both calcine charging and green-feed charging; and this leads to cramped handling facilities, especially in the vicinity of the Wagstaff gun, and results in a certain degree of redundancy. Although dual capacity has been used by Phelps Dodge at Morenci (Arizona) for charging dry hearth-type reverberatory furnaces, this resulted from changes in operating practice at the smelter and was not part of the original design concept. Dual capability is rarely provided in modern reverb design.

We do not expect operating problems to result from dual capability, and once the fluid-bed roaster and acid plant start operating, the green charging capability will no longer be required on a regular basis. Rather, this capability would only be reactivated if the roaster is down for repairs and possibly also for handling reverts (i.e., copper containing residues).

The furnace roof is made of suspended basic refractory. The specifications for this roof called for panelized construction so that if an area of refractory needed replacement, an entire panel could be lifted out of place by an overhead crane and replaced with a new panel. Many recent reverberatory furnaces utilize suspended roofs with panelized construction. We find that the construction at Ergani is such that although groups of refractory bricks are in fact suspended as panels, the clearances provided for in the structural steel design of the panel supports does not permit refractory to be lifted or replaced as panels. Because of this, any roof repairs and brick replacement would have to be done on the more tedious brick-by-brick basis.

We understand that the burner specification called for the burners to be recessed away from the firing wall so that the burners could aspirate in a certain amount of tertiary air. At present, (January 1972), the burners have been installed flush with the wall, but appear to operate so far without any problems. Should it be determined in the future that burners suffer from overheating and excessive coking of the fuel oil stream, it might be necessary to move them 6-8 inches (15-20 cm) behind the firing end wall surface, to permit the inspiration of additional air by the furnace draft.

We understand that the original design did not incorporate a flue bypassing the waste-heat boiler, which would have enabled the plant to keep the reverb hot and minimize refractory damage while the waste heat boiler was down for repairs and isolated from the system. Such a bypass flue was subsequently designed at Etibank's request, and the additional time to install it contributed towards the delay in starting up the new plant.

Inadequate attention to the waste-heat boiler design was a major reason for the delay in starting up the plant. In general, waste-heat boilers in metallurgical-furnace service are subjected to much heavier dust loadings than is the case in other applications of this equipment, or even is the case for conventional coal fired boilers. In addition, inlet gas temperatures may be higher and the entrained particles may be at the point of fusion, or molten. Thus, metallurgical-service waste heat boilers require a more open spacing in the convection sections and in the screen sections. The tube arrangement in these sections should provide easy access for cleaning tube surfaces such as by rather elaborate soot-blowing apparatus. Adequate provision should be made for removing accumulations of solids loosened by the soot blowers.

The initial type of waste-heat boiler installed did not suit such requirements so that major time-consuming alterations had to be undertaken in the field.

The waste-heat boiler as now installed utilizes "bayonet-type" tubes; that is, water tubes in which cold water descends through a tube centered within the boiler tube proper, and then is heated as it ascends the annulus formed by the two tubes. The disadvantage of this type of construction as compared to the more usual U-tube construction is that the annulus where mineral deposits might accumulate cannot be cleaned by pulling through a reamer. Therefore, the boiler feed water should be of extremely high purity (essentially 100% condensate).

The proper "burning-in" of the floor (or crucible) of a reverberatory furnace is important if continued, trouble-free operations are to be achieved. At Etibank's request, we submitted a detailed procedure for the burning-in process in time for implementation and this is presented here as Appendix B. We do not know the details of the procedure actually used, but we understand it differed slightly from that given in Appendix B.

As a result of our initial visit to the field in January 1971, we recommended the following changes for implementation before commissioning the furnace. Most of these have been incorporated.

- Alteration of matte and slag launders in order to permit easy access to the tap holes;
- Installation of a fuel return line on burner oil supply to avoid dead ends and freezing of the oil;
- Instrumentation for monitoring and recording of furnace draft;
- Compressed air outlets near the roof of the reverberatory furnace for roof cleaning with air lances; and
- Holes in roof refractory for measuring slag and matte levels with a dip stick.

In operating a deep-bath type of furnace, the molten charge in the reverberatory furnace acts as a thermal flywheel to regulate temperature by providing a large reservoir of heat. Hence, it is important to maintain adequate depth of matte and slag at all times. When the depth of matte is adequate, magnetite accumulations on the bottom of the furnace are minimal, especially when matte grade is above 30%, which permits the magnetite to float between the slag and the matte layers. In general, magnetite accumulation is less of a problem with deep-bath type furnaces than with other types (e.g., dry hearth). But eventually the magnetite will adhere to the furnace bottom especially when too much matte is tapped and the matte layer is thin.

The side walls should be protected from deterioration by fettling* with coarse flux charged via the slinger belts.

With certain types of calcines, foaming of the slag (serious enough sometimes to fill the furnace volume) can occur after the calcines have been charged from the Wagstaff gun. The foaming is a result of gas evolution during the reduction of magnetite in the calcine by reaction with the matte:

$$3Fe_3O_4 + FeS = 10FeO + SO_2$$

The exact causes of foaming are not well understood, but plant experience indicates that viscous, highly siliceous slags are more susceptible to foaming.

e. Converters

Because the old smelter at Ergani included converters of identical size and design, this is one section of the new plant with which the operating staff is familiar. However, there are several aspects of operating converters in association with reverberatory smelting and with acid production which will require the operating staff to "unlearn" old habits and to learn new ones.

- The hoods on the new converters do not fit sufficiently tight. Accordingly, we expect that, unless properly fitting hoods or aprons are provided, the converter gases will be considerably diluted unnecessarily by an influx of cold air through the gap between converter and hood. This is a major shortcoming in the converter section, and it should be corrected prior to the start-up of the acid plant.

- The new converters have provisions for feeding silica flux continuously via a belt conveyor and a discharge chute while the converter is on the

* The addition of coarse material such as ore or flux to protect sidewalls from the heat

slag blow. The chute is located inside the hood area and we expect that a problem might be encountered during operations as a result of metal accretions on this chute that impede the flow of flux. A platform for a laborer should be provided at the chute level, so the metal accretions and other blockages can be chipped out.

- Converters almost always have emergency-power systems that rotate the converters in case of power failure to prevent metal freezing in the tuyeres. In most cases this is done automatically. We understand that in both the old and the new sections of the converter aisle at Ergani, emergency converter turnout is actuated by a switch which requires operator intervention. Furthermore, because of the limited current output of storage-battery system used at Ergani, only one converter can be turned out at a time. We believe this to be a potentially dangerous situation, and accordingly recommend that the converter turnout system be made completely automatic, and that additional storage battery capacity be added to permit all the converters to be turned out at the same time.

- On the other hand, because we have noticed that the number of operating labor in the converter aisle who can be exposed to a real hazard during an automatic turnout is unusually high, the automatic turnout system should be designed to bring only the tuyeres above the liquid level, since the matte or slag backflowing into the tuyeres could freeze almost instantaneously. The final emptying of the converters could be done manually. The operating labor should be trained to conduct their work without intruding into the converter aisle except when absolutely necessary.

- Converting by its very nature is a batch operation, whereas an acid plant operates best when its input consists of a constant volume of gas of more or less constant chemical composition. In order to achieve such a constancy, the converter staff will have to schedule converter blows and to control blowing rates, a requirement which they are not used to observing. This requirement could require considerable retraining of staff and direct supervision of their activities for a time. Cooperation between converter section staff and acid plant staff to minimize variations in the gas stream is vitally important.

- The presence of excessive magnetite in the converter slag is usually a measure of improper converter operation. Magnetite in converter slag is much more detrimental to plant operation when the converter slag is charged into a reverberatory furnace, than when this slag is charged into a settler or cooled, crushed and charged back into the water-jacketed blast furnace. Thus, the reverb is less forgiving of bad converter practice than is the blast furnace. Although magnetite accumulations are less of a problem in deep-bath reverberatory smelting than in other types of furnaces, magnetite can and will invariably accumulate on the bottom of a reverberatory furnace. This will decrease the matte storage capacity and increase the frequency of tapping.

  We understand that in the old plant, the converter slags were not analyzed for magnetite. In the new plant, the magnetite content of converter slags should be determined on a regular basis, in addition to the copper analyses (required for proper material balances around the reverberatory furnace), so that the expertise of individual operators in terms of minimizing magnetite in converter slags can be assessed and checked on a day-to-day basis. The operating parameter to achieve such a skill is the amount of flux charged. It appears that practice for the converters in the old plant was based on using the minimum flux. For good reverb-converter operations, the converter slag should be as high in silica as possible, (28-30%). To obtain this, larger flux additions will be required. Also, silica should be charged as soon as the matte temperature permits its melting so that its presence suppresses excessive magnetite formation.

- The new plant at Ergani has ample converter capacity except when using green charge smelting with low-grade concentrates, which can result in matte grades considerably below 20% copper. If the acid plant is operating on gas produced under these conditions, it is essential that all the matte processed in the smelter be converted in the new converter aisle, to produce a maximum SO_2 supply since the converter off-gases from the older section of the converter aisle are not fed to the acid plant, but are vented directly to the atmosphere via the stack. Therefore, the matte volumes will be unusually high, and it will become important to operate the new converters for a maximum time on blow. A good way of evaluating the efficiency of converter blowing to maximize time on blow is to connect elapsed time, or running time, meters which are actuated by air pressure. Thus, the meters measure the fraction of time during each shift that the converter was actually blowing. A good target for effective converter aisle operation would be to utilize the converters in blow 75% of the time.

f. Blister Cake Casting

When the copper blow is complete, the molten copper is poured from the converter into a ladle. Cakes of blister copper are cast directly from this ladle and, after cooling and removal from the molds, they are stored prior to shipping. We believe that with the present system of blister cake casting, metal losses are needlessly high because the operation is inefficient as noted below.

- The presently used molds are small and shallow. As a result, each blister cake is formed with a considerable amount of fin area which is broken off during handling both in the plant and almost certainly during shipment. This is a source of copper loss.

- The blister cakes from such molds are light enough to be carried by a single person, and this can lead to losses by theft and by pilferage. Most refineries around the world, which purchase blister cake, will accept cakes up to as much as 3,000 kg each.

- There is a lot of splashing and spillage under present casting procedure which requires more labor and remelting of spillages which could be avoided by improving the casting procedure.

We suggest that the blister cake size be increased to the extent Etibank's marketing position permits, but at least to the extent that the size can no longer be carried by one person. Twice or three times (say 1,000 Kg.) its present volume of cake would meet this criteria. We also suggest that new molds be designed so that the necessary cake size can be obtained by filling the mold cavity only a part of the way. For example, anodes are cast on an anode wheel by such a method in order to obtain sharp clean edges. A deeper mold will require a properly prepared mold wash so that the blister cakes can be removed easily from the molds. A thick slurry of finely ground silica such as is normally used as mold wash in casting of anodes from blister copper may be a candidate mold wash.

g. Sulfuric Acid Plant

A detailed study of the sulfuric acid plant was outside the scope of the present effort. We find, however, that the acid plant is adequate and is oversized as is normal when the plant has to handle varying volumes and concentrations of gas produced by a converter. The plant is adequately instrumented. Etibank is concerned that the acid plant can only perform below its rated capacity because there is not sufficient sulfur in the feed, and requested that we check this. Our calculations, presented in Appendix C, support their concern. If the sulfuric acid shortage in Turkey demands the operation of the Ergani acid plant at full capacity, an additional source of SO_2 will have to be utilized, such could be provided technically by a pyrite roaster or an elemental sulfur burner. Since pyrite concentrate is produced at the smelter concentrator, this seems to be the appropriate source of additional sulfur units.

5. Shipping Facilities

When the new plant achieves normal operation, both blister copper and sulfuric acid will be shipped by rail to the market. In addition, the pyrite concentrate will be shipped by rail to the market. In addition, the pyrite concentrate will be shipped by rail to another sulfuric acid plant in a fertilizer complex some distance from the Ergani smelter. During our first visit to the field we were asked by the plant management to examine the shipping facilities to decide whether they were adequate to handle the increased volume of material coming from Ergani because of the modernized smelter. Because detailed data on material handling capacities, loading/unloading rates and especially railroad perforamnce were not available, a responsive reply to this request was not practical. But, our cursory examination shows that there is adequate storage capacity for all three items at the plant, and the bottleneck, if any, would have to do with the availability and reliability of railroad services.

During our first visit to the field, it was mentioned that the pyrite concentrates would be dried prior to shipment. We strongly recommend against this approach because dry concentrates are highly susceptible to spontaneous combustion. This phenomenon cannot be entirely avoided even with wet concentrates at times. Furthermore, considerable dusting and loss of material would occur if moisture content were below about 6.5%. We recommend the concentrates be shipped in their wet non-aerated state to the acid plant, and the pyrites be dried just prior to roasting.

6. Maintenance Facilities

We find the maintenance facilities at Ergani to be quite extensive as is usual with a plant location as remote as is Maden.

The machine shop and foundry are extensive. The staff is resourceful and capable of quality work as was proven during the second startup trial on the fluid bed roaster when an urgent need for a perforated steel plate arose.

The deficiency in the maintenance area appears to lie in the area of spare parts availability and inventory control. These problems are aggravated during periods of plant start-up. Therefore, it is essential that careful maintenance and replacement records be kept so that an adequate maintenance and replacement policy can be worked out and implemented. We understand that purchasing of spare parts is performed by Etibank's head office in Ankara. The procedure should be reviewed to determine that no unncessary delays are introduced, and to enable the plant management to exercise appropriate inventory control, if this does not now exist.

7. Others

The plant at Ergani recovers a small quantity of cement copper from various aqueous streams by precipitation with scrap iron. We noticed launders for contacting the dilute streams with scrap in three locations. At all three locations, the solution flow rate through the launders appeared to be very low. This type of contacting leads to excessively high iron consumption, as high as ten times the stoichiometric amount. The iron consumption could be decreased by increasing the rate of solution flow. The easiest way of doing this with the present launders would be to install pipes in the bottom of the launders provided with nozzles, or orifices, at selected locations so that the copper solution is injected at a high velocity into the pile of scrap iron. This construction will have to be of stainless steel.

B. THE CONSTRUCTION PROGRAM

We understand that the construction program to modernize the Ergani smelter was managed by Etibank utilizing design and consulting services provided by the Parsons-Jurden Corporation. Based on our first field trip, (January - February 1971), we believe that the completion of the plant was affected by overlapping delays in two critical areas, both of which postponed the start-up date for the reverberatory furnace. The first area involved the installation of a wrong type of waste-heat boiler by the sub-contractor which then had to undergo extensive modifications in the field. The second was the delay incurred by a late decision to install the by-pass flue. By using a by-pass flue, the reverberatory furnace can continue operations (usually on a curtailed basis) when the waste heat boiler is shut down for repairs. We understand that the initial design of the plant did not include a by-pass flue, and that this was designed later at Etibank's request. The time periods required to alter the waste heat boiler installation and to install the by-pass flue overlapped. Hence, the total delay was not as serious as it might have been, had each delay occurred separately.

C. OPERATIONS

During our first visit to the field, before the plant construction was complete, we undertook an intensive program to train the staff to operate the new smelter. The details of this program are presented in Appendix A.

During our second visit (January 1972) the plant was in operation but operations were far from normal, especially with respect to the fluid-bed roaster, which after one attempt had not yet operated satisfactorily. Our efforts concentrated on providing assistance in starting up the roaster. We helped conduct a second start-up trial which was discontinued because of the slurry pump failure. But, we developed detailed instructions for the engineer-in-charge for use in the next trial. The procedure we suggested for roaster start-up is summarized in Appendix B.

After the roaster and the acid plant are operating in a normal fashion and the concentrator efficiency has been improved, we expect that the need for additional instrumentation or facilities will become clear to enable proper sampling for process control. Also, at that time we strongly recommend that an adequate metallurgical accounting system be installed to assess the operation of the smelter in terms of copper losses, overall and by departments and to measure the effects of changes in operational practices undertaken by the staff to improve the operations.

IV. PRODUCTION COSTS

A. OPERATING COST FACTORS IN NORMALIZED PRODUCTION

The new plant at Ergani is still in the starting-up stages and at the time of our visit in January 1972 had not yet reached normalized production. For example, the concentrator was still performing below its design basis, both in terms of copper recovery and in the grade of concentrate produced. The fluidized-bed roaster system had not operated as yet, although start-up trials had been conducted. The reverberatory furnace was in operation but appeared to consume excessively high quantities of fuel. Start-up trials on the acid plant had yet to commence.

Under such conditions, a scrutiny and analysis of the components of the actual operating costs incurred (a part of our original work plan) can be misleading. Because of this, we present below a discussion of typical raw material and energy consumptions encountered in smelting operations. Depending on specific conditions at any plant location, these unit consumptions might vary by 10-20% from the values stated in this section. The values are, therefore, presented to serve as a target to be approached, achieved or surpassed after the plant start-up problems have been successfully overcome and normal production has been achieved. We have retained the customary English units in this section in order to emphasize the fact that the numerical values presented below are "rules-of-thumb" by which any smelting operation is gauged.

1. Fuel Oil

In a reverberatory furnace, the quantity of fuel consumed is evaluated on the basis of heat input per ton of solid charge (green concentrate + flux or calcine + flux). Typical values are 4 to 8 million Btu per metric ton of charge in green feed smelting and 2.5 to 5 million Btu per metric ton of charge in calcine smelting. An efficient waste-heat boiler will recover around 45-50% of the heat content in the reverberatory furnace-off-gases.

2. Flux Consumption

The silica flux and limestone requirements are largely dictated by the impurities in the concentrate. Silica combines with iron to form iron silicates, typically fayalite ($2FeO \cdot SiO_2$) and limestone is added for viscosity control. A typical reverberatory furnace slag will contain 30-40% silica and 3-10% CaO. The total silica requirement is calculated on the basis of a preselected "silicate degree" for the slag. The silicate degree (sum of weight of oxygen in acidic oxides such as SiO_2 divided by weight in consistent units of oxygen in basic oxides such as CaO, FeO) ranges in reverb slags from 1.5 to 2 with 1.7 as a typical value. Thus, the total silica requirement can be calculated on the basis of the weight of the slag ("slag volume"). Slags high in alumina (contributed by the gangue components of the ore and siliceous flux) usually require higher amounts of CaO. Some examples that illustrate such variations are:

	SiO_2	CaO	Al_2O_3
Range cited above	30-40	3-10	-
El Paso, Texas	40.1	10.1	6.8
Tsumeb, Southwest Africa	27.2	17.5	4
Noranda, Quebec	36-39	10-20	6.0-8.8

3. Refractory Consumption

Consumption of refractory in the reverberatory furnace can be highly variable, and low consumption is achieved only when the reverberatory furnace can be operated for extended periods of time, under very steady conditions, and by minimizing thermal shock to the refractory. Average values for refractory consumption can range from 1 to 5 lb. per metric ton of charge. Much of the consumption occurs in the roof. A typical value for refractory consumption for reverb roof alone would be 0.75 lb/metric ton of solid charge.

Converter refractory consumption is a measure of the skill of individual converter operators. In several smelters around the world, the converter refractory is coated with a layer of magnetite produced by blowing a charge of matte without the addition of flux. The utility of this approach in extending the life of basic converter refractory is still being debated. A copy of a detailed written discussion on refractory consumption in converters at various plants around the world (*) was presented to the plant management during the field trip. Refractory consumption in the converter with good operating practice averages from 5 to 9 lb. per metric ton of blister.

4. Water

Water consumption is basically a function of the specific flowsheet of the smelter and the allowances made in the flowsheet for water recycle and reuse. For example, smelters (and

*F.E. Lathe and L. Hodnett, "Data Regarding Converting Practice in Various Countries", Trans. Met. Soc. AIME, 212, 603-17 (1958).

concentrators) in the Western United States and other arid regions operate on almost 100% recycle on plant water. Total recycle is not necessary in other parts of the world. Water consumption in smelting averages at approximately 4000 gallons per metric ton of copper. In acid plants, about 8000 gallons are required per metric ton of 98% H_2SO_4.

5. Labor

The unit costs of labor, i.e., direct operating labor, supervision and administration, can vary widely in different parts of the world. Generally, less labor quantity is used in countries where unit cost of labor is high than in countries where the unit cost of labor is low. As a rule, however, the fraction of the total direct operating costs contributed by the total cost of labor is less variable when comparing smelters throughout the world. Thus, smelter and acid plant labor should contribute about 25-35% of the total direct operating cost in copper smelting. This labor cost component would include the cost of social benefits.

B. OPPORTUNITIES FOR COST REDUCTION

The management at the smelter has two basic tasks. The first is to overcome start-up problems, and evaluate the operation of the plant to learn rapidly the idiosyncracies of equipment in the new plant, thereby shortening the time required to normalize production. The second is to reduce the expenses involved in the operation of the plant systematically after normalized operation has been achieved. We suggest a procedure for accomplishing these two tasks in the Action Plan, which is presented in detail in Chapter V.

An invaluable and essential tool for evaluating the operation of a plant is to enforce a system of metallurgical accounting (or the production of detailed material balances indicating the position or final disposition of all plant inputs), and to use the results to compare and evaluate differences detected in the plant performance with the guidelines and the factors discussed so far in this chapter.

Metallurgical accounting (and, consequently, cost accounting) is practiced in the copper industry much more rigorously by the custom smelters*, than by captive smelters such as Ergani, which process only its own concentrates.

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*Smelters that smelt concentrates from various different sources owned by others for a fee, frequently returning the copper and associated precious metals to the owner of the concentrates.

Custom smelters are extremely cost conscious since profit derives from the sale of services rather than the products.

Nevertheless, a metallurgical accounting system is fairly inexpensive to enforce and can, in our opinion, provide high returns in terms of improving plant performance to reduce costs and in terms of keeping a check on the plant losses. A typical metallurgical accounting system is operated in two parts:

- Sampling slags, matte and blister copper by shift and by day and compiling the weights of key charge material flowing to various units; and

- Compiling the appropriate results of such sampling monthly and yearly in terms of the data normally used by the accounting department for their cost reports, historic production data, and for cost accounting

A useful tool, also, is to keep a record of plant delays to pinpoint the causes of lost time for production. Methods for proper sampling and for estimating material balances in the concentrator are discussed widely in the literature and the plant management is aware of these publications. Systems for estimating material balances, or metallurgical accounting systems, in smelters are usually unpublicized. Because of this, in the discussion below we place primary emphasis on metallurgical accounting in the smelter.

At Ergani, the material inputs to the reverb can be obtained from the belt conveyor scales. The data from this source can be complemented by taking volumetric measurements (e.g., the number of buckets of green charge charged into the furance) when a weight measurement is not possible. As a rule, volumetric measurements lead to a greater margin of error. Similarly, the quantity of silica or other flux used in the reverberatory or converters can also be measured. There is also the circulating slag load to the reverberatory furnace from the converter that has to be taken into account.

At several smelters, a magnetite balance is performed around the reverb especially when using calcine charges in order to monitor the extent of reduction of magnetite by the matte and/or to estimate the accumulation of the magnetite on the furnace bottom. The results of careful maintenance of magnetite balance records around the furnace can frequently indicate magnetite accumulation.

The data thus collected are used to monitor the losses of copper in various handling operations and to assess the efficiency of the operation. Two measures of reverberatory furnace efficiency are the fuel oil used per ton of solid charge and the steam produced per unit of fuel oil. Slag composition control is of primary importance since the loss of copper in smelting occurs mainly via the reverberatory furnace slag.

After the inputs into the process have been measured in the manner noted above, the outputs must also be measured and calculated. The primary outputs are copper as blister, slag from the reverberatory furnace, and flue dust lost up the chimney. The loss of copper, the valuable material being recovered by the smelter, is a primary concern in a materials balance.

A critical examination of losses and unit consumptions of raw materials, product and excessive inventories, can then be conducted by converting these results to the equivalent cash value. This procedure is a cost-accounting function and identifies the critical areas that need improvement in order to reduce costs, and the justification for new capital expenditures when these are involved.

The procedure does have some difficulties, however,that can reduce the accuracy of the results. Since it is difficult to weight molten products, it is difficult to obtain an accurate measure of reverberatory slag. A good basis for estimating quantity can be calculation from the inputs of solid materials to the reverb and converter, and from the analysis of slag. An iron and silica balance would be made from such data.

V. THE ACTION PLAN

The discussion in the preceding chapters indicates that a number of tasks need to be accomplished at Ergani in order to achieve efficient copper production, and that these tasks should be arranged by priority to accomplish. The short-term tasks that require immediate attention are the start-up of the fluidized-bed roaster and the acid plant. In the intermediate-term, after the start-up has been successfully accomplished, the overall plant operation will have to be normalized so that the equipment components perform to their design specifications. In the long-term, gradual improvement in all sections of the operating plant will have to be achieved in order to improve copper recovery and reduce the operating costs. We developed in conjunction with Etibank, a number of suggestions in detail for achieving the short, intermediate and long-term goals, and these are presented below. In general, short-term objectives should be achieved before considering intermediate term objectives, and these before considering the long-term objectives. Achieving objectives in such a sequence is necessary because the results in the short term affect the work plan in the achieving the intermediate-term objectives, and similarly for achieving the long-term objectives.

A. SHORT-TERM TASKS

The short-term tasks comprise roaster and acid plant start-up.

1. Roaster

During our field work in January 1972, we assisted in altering details of the roaster installation as follows:

- Install U-tube manometers to verify pressure gauges and a differential pressure manometer to measure the differential pressure between two points below the level of the fluidized bed.

- Install purge lines on all pressure taps.

We believe that the following alterations in the installation are necessary:

- Install automatic mechanism to shut off main blower if the fluid bed temperature exceeds about 1000°C.

- Eliminate interconnections in the various sources of compressed air and assure adequate supply to all units.

- Relocate orifice plate to measure the actual air used for fluidization.

- Install a gate valve in series with the butterfly valve to obtain positive control of fluidizing air.

- Patch up holes in roaster brickwork.

- Centralize all information in the control room by installing channels for communication with various levels in the roaster installation.

- Install flow indicators on all the purge lines.

- Install vibrators on the calcine bin.

- Hardsurface the elbow in the Wagstaff gun.

On completion of these alterations, the roaster should be started up using the guidelines presented in Appendix E, and details that may be found in the literature*. No doubt new and unanticipated problems can arise in a new start-up trial and the operators should be prepared for these. We are confident that the plant engineers now have adequate technical background to identify them and to plan and execute corrective actions.

2. The Acid Plant

We understand from the UNIDO expert assisting Etibank on acid-plant operations, whom we met in January 1972, that the acid plant could be started up within a month after roaster start-up. Should successful roaster start-up be delayed, we would suggest that Etibank consider starting up the acid plant using only the converter off-gases to take advantage of the UNIDO expert's services in overcoming usual start-up problems.

Several acid plants around the world do operate only on converter off-gases; for example, the ASARCO smelters in Tacoma (Washington) and Hayden (Arizona). However, the acid plant at Ergani will differ in that it would then be operating on a smaller quantity of more dilute gas than it is designed for and under these conditions supplemental heat input might be necessary. The ability or inability to supply the supplemental

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*J.C. Blair, "Fluosolids Roasting of Copper Concentrates at Copperhill", J. Metals, March 1966.

heat might well decide whether or not the acid plant can be started up without utilizing the roaster gases. Another problem might arise if strong sulfuric acid purchase and storage is required for starting up the absorption section in the acid plant.

Installation of tight-fitting hoods or aprons on the new converters will be even more necessary in such an event in order to minimize dilution of the gases. Moreover, the converter operations would have to be carefully controlled and synchronized in order to produce (as far as practicable) a steady volume of gases of more or less constant SO_2 content.

3. Reverberatory Furnace

The following changes, suggested to Etibank during our first field trip in January 1971, have been implemented before the construction program was completed.

- Alter matte launders
- Install return line for burner fuel oil
- Install instrumentation for measuring furnace draft
- Provide compressed air for roof cleaning
- Provide holes in the roof for dipsticks

It remains only to provide instrumentation for sampling and analyzing the furnace off-gases, such as by the Orsat technique.

We would like at this point to reiterate our recommendation to alter the slag launder to provide a more convenient access by the operators during the tapping operations. Apparently, the plant management, after our visit in January 1971, decided against the alteration. There is a possibility that at some time a high slag or a high matte level condition can exist, and the flow through the slag taphole will be uncontrollable. High slag flow would be dangerous enough, but if matte flows into the granulating bosh, serious explosions could occur with severe injury to men and equipment.

4. Converter

- Install automatic emergency turnout equipment to raise the tuyeres above the level of the molten bath in case of power failure.

B. INTERMEDIATE-TERM TASKS

A substantial period should be allowed from the time that all equipment units of the smelter have been started up until the time they can be operated consistently and controllably at their design performance. The length of this period depends on the particular situation.

In a new plant such as Ergani, the operating staff does not have an experienced background in the key equipment items, i.e., the deep-bath reverberatory furnace, the fluid-bed roaster, and acid plant. Thus, the period of time required to achieve normalised operation will be a function of their ability to gain experience in the day to-day operations of the plant and to use such experience to correct indicated deficiencies in design. A period about three to six months from the time all individual units are in operation might be considered "normal" for the normalisation of smelter operations, but at Ergani, in view of the foregoing, at least six months after successful roaster start-up and perhaps as much as a year, is probably more realistic.

During the normalisation period, the plant management should continue to evaluate and measure the rate of improvement in the operation of individual units and determine the reasons for the deficiencies in equipment that may be indicated. Then, they would be in a position to anticipate major managerial problems in achieving normalisation and to act to solve them such as by procuring the necessary assistance, either within the Etibank organisation or from outside.

As already noted, normalisation problems are difficult to anticipate until the roaster and acid plant will have started up successfully. However, we can present below a list of possible problem areas that might be encountered during normalisation of production.

1. Concentrator

The problem here is to identify the reasons for poor plant performance. As already noted, the following phenomena might be significant in finding the proper solution.

- Frequent equipment failures in the crushing plant
- The presence of oxidised copper minerals in the ore requiring Na_2S additions
- The presence of slimes in the rougher flotation circuit requiring desliming cyclones.

- Undersized ball mills. The installation of instrumentation to measure the power drawn by a ball mill as a function of feed input might help to increase ball mill throughput. Also, changes in maximum ball size might be tried.

- The retention time in flotation might be insufficient.

- Changes in reagents, reagent dosages and pH regimes in flotation might be considered.

4. Roaster

The normalization problems here would involve changes in materials handling either on the upstream end or on the downstream end, rather than changes in the operation of the roaster itself. The following areas might be significant:

- Problems in slurry handling and pumping requiring changes such as in slurry composition, in slurry-pump location, and in nozzle location.

- Hang-up problems in cyclones requiring mechanical vibration or poke-holes to clear passages for proper functioning.

- High abrasion rates in the cyclones, requiring the installation of a ceramic abrasion resistant lining.

- Problems in operating the automatic pressure balancing system.

- Accumulation of coarse material in the fluoseals requiring dismantling for cleaning, excessive oxidation of the calcine from excess seal fluidizing air; need to change from fluoseals to a mechanical discharge seal.

- Improper diameter of fluid bed drain, hanging in the elbow of the drain requiring the installation of mechanical vibration or the use of air lances.

[illegible] Leaching

The possible problem areas are:

- Charging problems leading to foaming, to too-frequent opening of doors for corrective measures, and consequently, to excessive heat requirements.

- Slagging problems fouling the waste-heat boiler tubing.

- Refractory replacement problems.

- Magnetite build-up problems, requiring cast-iron additions to the bath, or air or steam lancing.

4. Converters

The problem areas here might be related to producing suitable feed gas for the acid plant, and would involve training the staff to control blowing rates in response to acid-plant requirements and to schedule the converter blows.

C. LONG-TERM TASKS

The major task for reducing operating costs is the installation of a metllaurgical and cost accounting system which is described in detail in Chapter IV. We strongly recommend the installation of such a system at the appropriate time.

Other tasks might be the addition to the physical plant of specific facilities for improving plant efficiency and reducing costs. Examples are

- The design and use of larger and deeper moulds for blister casting.

- The installation of instrumentation for automatic sampling in the concentrator.

- The installation of running-time meters on the new converters.

- The investigation of the possibility of doubling ladle size.

- Alteration of cementation procedures to reduce iron consumption.

- Consideration of the need for a pyrite roaster.

- Consideration of the economics of smelting high grade ores directly in the new plant (added to the reverb for fettling or added to the converter as flux) instead of utilizing the old plant. This would avoid maintenance costs on the water-jacketed furnaces.

VI. ADVICE AND GUIDANCE IN RELATED OPERATIONS

During our first visit to the field (January - February 1971), the plant management requested us to address ourselves to certain specific problems that were of immediate concern to them that bore some relation to the scope of our work. We complied with this request and as a result prepared a series of memoranda responsive to the specific problems identified. Most of these were submitted in draft form to the plant management while we were still in the field. Our formal response to these questions are included here as a series of appendices. Pertinent conclusions and recommendations are summarized below.

A. PROCEDURE FOR COMMISSIONING THE REVERBERATORY FURNACE

The detailed procedure for burning-in the bottom (or crucible) of the reverberatory furnace is presented in Appendix B.

B. PLANT CHIMNEY CAPACITY CALCULATIONS

The plant management was concerned that the existing chimney system would not permit simultaneous operation of the new plant and the old water-jacketed furnaces. Calculations presented in Appendix C indicate that when the reverberatory furnace is operating on green feed smelting, the controlling case, adequate draft exists for the simultaneous operation of one and perhaps both water-jacketed furnaces.

C. SULFURIC ACID PRODUCTION

The sulfuric acid plant is rated at 110,000 tons per year of acid. Acid plants are oversized when they are required to treat variable volumes of gases of variable composition, such as are produced during converting. The calculations in Appendix D show that there is insufficient sulfur in the plant feed materials for the full acid plant capacity to be utilized (assuming converting operations can be controlled to produce a high quality off-gas). If the national acid demand requires the operation of the acid plant at the rated capacity, a supplemental source of SO_2, such as an elemental sulfur burner or a pyrites roaster will have to be provided.

D. UTILIZATION OF COPPER-RICH DUST FROM WATER-JACKETED FURNACES

Approximately 50,000 tons of dust have been collected from the dust chambers of the water-jacketed furnaces and stockpiled. The calculations presented in Appendix F show that any reasonable amount of this dust (up to 60% of the total charge in green feed smelting and up to 85% of the total charge in calcine smelting) could be charged into the reverberatory furnace via the calcine bin and Wagstaff gun without a loss of blister production. The calculations in Appendix F, however, have no production cost inputs and it is possible that it might be less costly to concentrate the copper in the dust by flotation if this can be shown to be technically feasible.

E. TAP-HOLE REPAIRS

Appendix G presents procedures for repairing matte and slag tap-holes.

[illegible]

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APPENDIX A

THE TRAINING OF THE STAFF FOR THE NEW SMELTER

I. INTRODUCTION AND OBJECTIVES

The new copper smelting operations at the Ergani smelter involve technology and operating practices that will be new experiences for the operations staff. Accordingly, the program of training, formulated during the first field trip in January 1971, had the following objectives:

- to provide the opportunity for the staff to acquire an adequate understanding of the new technology, perceive the areas where the old plant practices either no longer apply or can be improved, and perceive the areas where efficiency can be increased;

- to provide an opportunity for the personnel directly concerned with operation to become familiar with new equipment items, i.e., those not previously used in the smelter; and

- to provide in outline form the elements of the operating practice for the new smelter.

On completion of the program, the staff could then be in a position to expand the outline of the operating practice developed during the program, to produce a comprehensive plant operating manual that ultimately could be used to assist in attaining high operating efficiencies.

II. METHOD FOR ACCOMPLISHMENT

The total effort was organized to train a number of staff categories. The composition of the class in each category was selected by the plant manager. The categories selected and the membership were the following:

- __Technological Principles__: The sessions in this category were attended by representatives of the supervisory staff only, all of whom were graduate engineers.

- __Operating Practices__: The sessions in this category were attended by the same representatives of the supervisory staff, and by the technicians and shift foremen.

- __Demonstrations__: The sessions in this category were attended by the same technicians and shift foremen, and by the shift workers and interested supervisory staff.

Because the plant at the time was under construction, the emphasis was placed on classroom discussions. The training program was conducted over a period of two weeks. Technological Principles were presented in ten morning sessions averaging two to three hours each. The participants were assigned homework problems in order to assess their understanding of the classroom presentations. Operating Practices required eight afternoon sessions of two to three hours each, and Demonstrations required two afternoon sessions of the same duration.

The Technological Principles emphasized the physico-chemical principles involved in the operation of the new smelter and were presented at the level of a graduate course in Extractive Metallurgy. Operating Practices were presented on a practical level in terms of:

- what is the best practice for accomplishing a particular operating goal
- why is such the best practice
- what problems can be encountered in applying such practice, and how can these problems be overcome.

III. TECHNICAL CONTENT

The technical content in each category was the following:

a. Technological Principles

1. Fluidization

The fluidized state; properties of fluid beds; their advantages and disadvantages; use of fluid beds as chemical reactors; basic fluid bed calculations; methods of feeding and removing solids.

2. Roasting of Sulfides

Basic methods: multiple-hearth; fluid-bed; flash (or pneumatic); integrated roaster-smelters; thermodynamic and kinetic principles; formulation of heat balances. Process control. Problem on design of pyrites roaster.

3. Smelting of Mattes

History of reverberatory smelting and converting applications in copper extraction; types of reverberatory furnaces; radiant heat transfer in a reverb.

Chemical reactions in the reverb; Cu-Fe-S equilibria, $FeS-Cu_2S$, $FeS-Cu_2S-FeO$ systems.

Matte-slag equilibria in the reverb--$FeO-SiO_2$, $CaO-FeO-SiO_2$, $FeO-Fe_3O_4-SiO_2$; importance of P_{O_2} and temperature in reduction of magnetite accretions. Copper losses in reverb slag.

Matte-slag equilibria in the converter--$FeO-Fe_3O_4-SiO_2$, Cu_2S-Cu, magnetite control.

Methods for calculating matte-fall and flux requirements. Problems on charge calculations.

4. Process Control

Sampling of process input and output streams; metallurgical and cost accounting; instrumentation for maintaining high efficiency; draft control.

b. Operating Practices

1. Fluo-Solids Roaster

Start-up procedure; normal operating practice; operating problems and remedies; instrumentation.

2. Reverberatory Furnace

Similarities and differences between Ergani and Murgul reverbs.

Start-up procedure--burning in of the bottom.

Normal operations -

Combustion control--burners, draft; charging method, green feed, calcine, regularity, speed, charge mixing, slag removal, care of granulating, sampling, care of bay and launder, manipulation to fit with converter operation.

Matte removal--tap hole care, plugging, opening, expeditious work, cooperation of crane and converters, lining of ladles.

Refractory maintenance -

Collection and disposition of dust; ancillary equipment: waste-heat boiler, precipitator, flues, scrubbers.

Preventive maintenance--operating problems and remedies; magnetite control; emergency and non-emergency shut-off procedures.

3. Converters

Charging; slag blow; silica addition; skimming; copper blow.

4. Casting

Moulds; trimming; sampling; handling.

c. Demonstrations

1. Roasting (Simulated)

Start-up; combustion and temperature control during operation.

2. Reverberatory Furnace (Simulated)

Charging methods; slag and matte removal; roof repair.

APPENDIX B

COMMISSIONING PROCEDURE FOR REVERBERATORY-FURNACE/WASTE-HEAT-BOILER/ ELECTROSTATIC-PRECIPITATOR UNIT

I. INTRODUCTION

The reverberatory equipment involved in the new smelter at Ergani represents a distinct departure from water-jacketed equipment used in the smelting practices followed previously. One should particularly note the following:

- The water-jacketed furnace depends on the coke in the charge for fuel which must be carefully prepared and proportioned in the charge. The reverberatory gets its fuel separately from the charge and no chemical reactions occur before the charge is melted. Instead, fuel is burned by itself, at as high an efficiency as possible, to melt the charge which then reacts chemically to form slag and matte. These are very much like the slag and matte formed in the water-jacketed furnace and are handled much the same way. Combustion control in the reverberatory furnace is very important for good results.

- Essentially, the reverb is a big box. The roof, walls and bottom are constructed of suitable refractory material in such a way as to receive the charge; to contain the oil fire; to contain the liquid material; and to conduct away the waste gases. Suitable arrangements are made to allow for the dimensional expansion of the construction materials, especially those subjected to high temperatures. The furnace construction provides for means of measuring and controlling draft, sampling waste gases, and controlling the flows of the slag and matte.

- The reverb furnace at Ergani is of modern construction. The inner walls have been built of sections of chrome-mag brick provided with about 4 mm. combustible material in the joints per 300 mm., as expansion allowance. The roof is the same except that there is also sheet iron in each joint. The iron melts and penetrates the refractory, and, as it oxidizes, makes a bond to hold the brick together and make the roof airtight.

The installation of the floor of the furnace is to be made and completed as part of the commissioning of the furnace. Previously, the bottom had been partially built up of materials that could be placed cold in layers.

The bottom layer consists of 2.26 meters (7.5 feet) of clay, placed in 15 cm. (6-inch) layers, each tamped firmly. A 30 cm. (12-inch) layer of crushed magnetite has been placed on top of the clay; and a 91 cm. (3.0 foot) layer of ground chromite ore placed on the magnetite. The upper two layers have been placed in a similar fashion as the clay.

The remaining topmost floor layer will be placed at high temperatures as a part of the commissioning operations. The proper placement of this remaining layer is essential, if the floor is to have a long and trouble-free life. It will comprise successive placements of slag and magnetite, built up to the level of the lowest tap hole.

II. COMMISSIONING PROCEDURE

Because of the urgency to resume blister copper production at the time we were asked to set down the procedure, the details depend on commissioning the reverberatory furnace before the flue bypassing the waste-heat boiler is completed. This lack does not affect the procedure for reverb start-up, but does introduce a risk should the waste-heat boiler fail for any reason before the bypass flue is installed.

A. Pre-Operation Inspection

1. Brickwork

Check all brickwork to assure proper installation especially brick adjacent to the skewback, closures around tap holes, doors and other openings; and to assure that no obvious leakage paths exist.

2. Tie-Rods

Plan the means whereby the tie-rods can be loosened (if necessary) to accommodate the expansion of the brickwork.

3. Fuel System

Check fuel-supply system, pipeline steam-tracing, purge lines, valves, heaters, and burners for proper operation in accordance with the manufacturer's instructions. Run-in all pumps to assure that they are mechanically sound and will operate trouble-free; and that the fuel pumps can deliver the quantities at the pressures required.

4. Combustion Air Supply

Check duct system for proper installation; for freedom from leaks and foreign materials; for operable valves and dampers and for the installation of the filters, protective screens, gauges, and controls. Run-in all fans to assure that they are mechanically sound and will operate trouble-free; and that they can deliver the air quantities at the pressures required.

5. Furnace Water Cooling

Check all water jackets and water supply system for cleanliness, flow rates, and leakage.

6. Waste-Heat Boiler

Perform a complete pre-operation check of the waste-heat boiler in accordance with the manufacturer's instructions.

7. Electrostatic Precipitator

Perform a complete pre-operation check of the electrostatic precipitator in accordance with the manufacturer's instructions.

8. Downstream Flue System

Check induced-draft fan and associated ductwork, gauges, and dampers for proper installation. Run-in fan to assure that it is mechanically sound and will operate trouble-free; and that the water-cooled bearing system is adequately installed.

9. Furnace Start-Up Instrumentation

Have equipment available for measuring furnace temperature; a thermometer for low temperatures and a pyrometer for the higher temperatures.

10. Completion of Inspection

When satisfied that all of the above are in working order, the unit is ready for firing.

B. Firing Procedure

1. With the flue damper wide open start small fire on one or two burners, keeping others protected from heat in accordance with the manufacturer's instructions, or by continuous purging of the nozzle with steam and by maintaining a minimum flow of air through the burner register. After lighting up, the damper can be adjusted to give the desired draft.

2. Adjust fires so temperature in furnace will rise at the rate of about 6°C per hour. Initially, use a thermocouple well at the uptake end, mounted through the roof. Use a radiation pyrometer when the temperature becomes high enough

3. Examine tie rods frequently, not only in the steps so far but also in the steps to follow where furnace temperature changes occur. Slacken off nuts if they appear to be over loaded. A strain gauge is a good device for measuring the load. Keep record of amount slackened or tightened.

4. When temperature has reached about 1000°C, a layer of about 7 to 8 cm of slag-magnetite mix should be spread over the bottom by slinger or hand shoveling. When this material has melted, a second layer may be added, with a steadily increasing proportion of magnetite. This procedure is repeated until the level of the bottom is even with the bottom of the lowest matte tap hole. Initially, the magnetite content of the mixture should be 10-20%. At the end, the magnetite content should be no greater than 40-50%. During the build-up of the bottom, it is important to maximize magnetite addition consistent with maintaining fluidity in the mixture to assure that ultimately the floor will be level. The control of the rate of magnetite additions is a matter of judgment based on experience.

5. When the level of the floor has reached the level of the lowest tap hole, no further additions are made and the temperature should then be held steady for about eight hours. During the build-up of the floor, a convenient way of checking the level of the floor is by inserting a steel rod through the roof in one or more locations. The same technique can be used later during operations to measure the levels of slag and matte, and even the magnetite build-up on the floor. This period of steady temperature is considered as a "soaking period".

6. At the beginning of the soaking period, the fires are shut off and all furnace openings plugged as tightly as possible. Also, at the end of this "soaking" period, the uptake damper at the waste heat boiler inlet is closed, with one burner remaining in operation to minimize cooling the refractory while the damper is being inserted. The furnace is then allowed to cool, without fire, until the roof refractory temperature reaches about 430-500°C.

7. While the final layer of the furnace floor is being installed, the burners will deliver at least the rated gas volume and temperature to the waste heat boiler. The elapsed time until the installation of the final layer can be 9 to 10 days, at longest. Hence, the initial commissioning of the waste heat boiler can be carried out in accordance with the manufacturer's recommendations during the burning in of the floor. During the soaking period, after the uptake damper is closed, the boiler can be entered for inspection and the steam line inter connection can be installed.

8. At the end of the cooling period, the furnace bottom will have solidified, and some cracks may have opened in it. Relight fires and bring up the furnace temperature at a medium rate, maintaining proper tension in the tie rods. When cracks have closed, the furnace is ready to start smelting. Regular charge may be added at this stage.

9. Should the period of time to inspect the boiler and interconnect the steam line exceed the period of time to seal the furnace, undesirable further melting of the furnace will occur thereby decreasing the ultimate refractory life.

10. During the previous operations, all charging equipment should have been checked out, and charge material readied for use.

III. PRECAUTIONS

A number of precautions should be taken so that sufficient flexibility exists in adjusting the firing conditions or in meeting emergency situations.

A. In aiming for a furnace floor that remains stable over its entire area under the normal operating conditions, a high temperature throughout the furnace is necessary during the burning in of the floor. This may be difficult to achieve at the outlet end of the furnace. Should it prove that flame length and size cannot be manipulated to give a high temperature in this region, it would be desirable to have in reserve some auxiliary firing arrangement that can be introduced in the outlet end to boost the temperature in that area.

B. As a safety precaution, should fires for any reason go out, shut off all oil supply and purge the furnace with fresh air for fifteen minutes before attempting to relight the fires.

C. The regulating damper at the waste heat boiler gas outlet should be adjusted so that the furnace draft under the arch at the uptake end is maintained at 1.0 to 1.5 mm (about 0.05 inches) water gauge. This ensures a reasonably low air leakage into the furnace and the avoidance of a positive pressure anywhere in the furnace that could have potential to damage the structure. Draft adjustments should be attempted only when the furnace doors are closed, that is during periods between charging the furnace with concentrate.

D. Should a failure occur in the waste heat boiler during the burning in of the floor such that the boiler must be shut down immediately, the furnace should be shut down by cutting the fire immediately, by opening the boiler bypass dampers, and by closing the guillotine damper as expeditiously as possible. When the guillotine shut-off damper is secure, the fires may be restarted and set to accommodate the bypass flue capacity.

E. Should a failure occur in the induced draft fan, it most likely would be in the bearings or in the electrical drive. In such a case the furnace should be fired at a level that will not produce a positive pressure in the furnace, until repairs are completed.

F. The water supply to the cooling water jackets around the furnace should maintain a rate of flow such that the temperature of the overflow water at each jacket does not exceed approximately 60°C.

APPENDIX C

PLANT CHIMNEY CAPACITY CALCULATIONS

I. INTRODUCTION

The new smelter at Ergani will utilize the existing underground gallery and chimney installation for venting the reverberatory furnace gases to the atmosphere. When the acid plant is not in operation, the roaster and new converter offgases can also be vented via this flue system as shown in Figure 3.

II. PURPOSE

The plant management was concerned that the flue system might not be able to handle the gases from the old and the new plant simultaneously. The old plant may be operated several months in a year on high-grade lump ore. We were asked to calculate whether or not simultaneous operations of the two plants could be carried out using the existing flue and chimney system.

III. CONCLUSIONS

The calculation indicate that the flue system will not be overloaded under normal operations of maximum flue gas production through green-feed reverberatory smelting and simultaneous operation of one water-jacketed furnace.

IV. DETAILED CALCULATIONS

The details of the gallery-chimney system after the various gas streams have been combined (as shown in Figure 3) are shown in elevation in Figure C-1. Details for the dimensions of ductwork in the new facilities used in the calculations were taken from appropriate construction drawings as needed.

The description, data, and calculations to follow apply to estimating whether the existing chimney and gallery and new and old ductwork, can support the operations expected under the design conditions. Calculations are shown both in the metric and English system of measurements. Three cases for calculation of chimney-gallery capacity were considered as follows

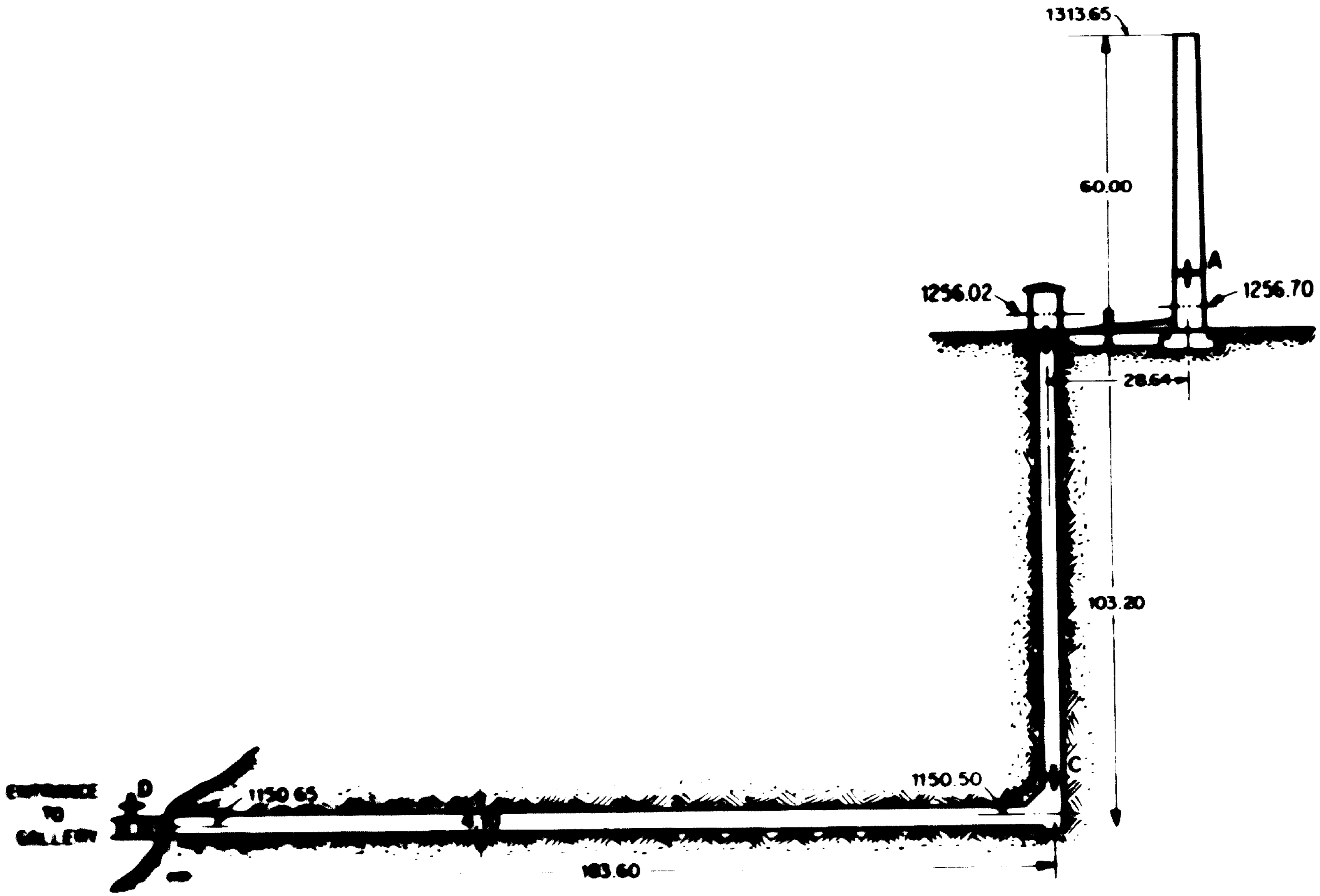

FIGURE C-1

DETAILS OF GALLERY-CHIMNEY SYSTEM AT ERGANI

Case	Description
1	Reverberatory furnace operates on green charge. The new converter gases vented to the flue. One old water-jacket furnace operates.
2	Reverberatory furnace operates on calcined charge. Converter gases sent to acid plant. One old water-jacket furnace operates.
3	Reverberatory furnace operates on green charge. Converter gases sent to acid plant. One old water jacket furnace operates.

Table C-1 contains a summary of the physical dimensions and derived cross-sectional areas of the Chimney-Gallery system.

Table C-2 contains the calculations of equivalent length of the various gas passages involved in the system. Part A contains the empirical formulae and the definitions of the terms. Part B are the actual calculations.

Table C-3 shows the calculation of ambient atmospheric pressure at Maden, Turkey, at the different elevations above sea level of the key locations in the chimney-gallery system.

The estimation of total gas flow to the entrance of the gallery is shown in Table C-4 for each of the three operating cases considered. Also, in Table C-5 the molecular weight of the gas is calculated for each of the three operating cases.

So far, the physical characteristics of the chimney gallery system and the flow characteristics of gases produced in three operating cases have been established. The next step becomes the estimation of the thermal characteristics.

In Table C-6 the specific heat is calculated for each of the gas compositions produced in the basic operating units, namely: the reverberatory furnace on green charge; the same on calcined charge, and the gases from the water-jacket furnaces. The specific heat of the new converter gas is estimated at the beginning of Table C-8, for Case 1, since this is the only operating case where new converter operation is considered.

TABLE _-1

PHYSICAL DIMENSIONS OF CHIMNEY/GALLERY SYSTEM

	Elevation Top (Meters / Feet)	Elevation Bottom (Meters / Feet)	Bends Angle (Degrees)	Bends Number	Length (Meters / Feet)	Cross-Section Area (Meters² / Feet²)	Cross-Section Perimeter (Meters / Feet)	Equivalent Length (Meters / Feet)	Source Drawing Number
1	[illegible]	[illegible]		[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
2	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	From Table [illegible]	[illegible]
3	[illegible]	[illegible]		[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
4	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]	[illegible]	From Table [illegible]	[illegible]

Notes: 1. Top of shaft ... on Figure ...
2. A to B
3. B to C
4. C to D

TABLE C-2 PART A

CALCULATION OF EQUIVALENT LENGTHS

Source: Peele, "Mining Engineers Handbook", 3rd Edition (Empirical Formulae in English Units)

1. Formula: L_e = [illegible] (Fig 22, 16-27) (Table 2, 16-20)

B = [illegible] (16-27) (all bends = 90°)

Combining: L_e = [illegible]

Where: L_e = equivalent length to be added to actual straight length in feet

A = average cross section area, ft^2

P = perimeter in feet

r = radius ratio = radius of bend / height of duct

a = aspect ratio = width of duct / height of duct

For abrupt turn (Figure 21, 16-27), assume "radius" = equal height, i.e., r = 1.0 and factor B is increased 10%

Then: B = [illegible]

and L_e = [illegible]

$K' \times 10^{10}$ = [illegible], take K as 20

For Case 1 (Table C-3), $K' \times 10^{10} = 22.0$

2 $K' \times 10^{10} = 22.1$ Use 22.1

3 $K' \times 10^{10} = 22.2$

TABLE C-2 - PART B

CALCULATION OF EQUIVALENT LENGTHS

Item	Entry 2 Table C-1	Entry 4 Table C-1
	Downstream (Use Upstream Section)	
1. A, ft^2	118	-
2. P, ft	47.5	-
3. a, 2.10/5.13	0.408	-
4. $K' \times 10^{10}$	22.1	-
5. $L_e = \frac{m}{ft}$	64.6 / 212	-
	Upstream (Use Upstream Section)	
6. A, ft^2	123	134
7. P, ft	28.5	48.0
8. a,	1.0	4.20/3.17 = 1.32
9. $K' \times 10^{10}$	22.1	22.1
10. $L_e = \frac{m}{ft}$	73.1 / 240	40.6 / 133
11. Straight Length $\frac{m}{ft}$	28.64 / 94	192.60 / 640
12. Equivalent Straight Length $\frac{m}{ft}$	166.3 / 546	233.2 / 773

TABLE C-3

CALCULATION OF AMBIENT ATMOSPHERIC PRESSURE

Elevation Above Sea Level		(1) Log $\frac{P}{P_o}$	$\frac{P}{P_o}$	Atmospheric Pressure	
Feet	Meters			psia	atm.
4310	1313.65	0.0711	0.849	12.5	0.849
4110	1256.70 1256.02	0.0679	0.859	12.7	0.859
3770	1152.50 1150.65	0.0622	0.867	12.8	0.867

(1) $\frac{P}{P_o} = (5.41 \times 10^{-5})\ h$

where: h in meters above sea level

P_o normal sea level pressure 14.7 psia

P pressure at elevation psia

TABLE C-4

ESTIMATION OF TOTAL GAS FLOW TO GALLERY

Total for Case $\frac{Nm^3/min}{SCFM}$	Component	Gas Flow $\frac{Nm^3/min}{SCFM}$ from Reverb.	Converter	Old Smelter	Total $\frac{Nm^3/min}{SCFM}$	Percent (Vol.) %
1.	O_2	11.3 / 399	32.2 / 1150	24.1 / 851	67.6 / 2400	4.4
	N_2	406 / 14336	225 / 8040	522 / 18250	1153 / 40686	75.2
1534.7 / 54,1[illegible]	CO_2	62.7 / 2220	-	47.1 / 1650	109.8 / 3870	7.1
	SO_2	34.3 / 1215	18.2 / 650	49.3 / 1725	101.8 / 3590	6.6
	(1) CO or (2) H_2O	(2) 78.4 / 2770	(2) 19.8 / 700	(1) 4.7 / 164	-	CO 0.3 H_2O 6.4
2.	O_2	7.9 / 280	-	24.1 / 851	32.0 / 1131	3.1
	N_2	[illegible]	-	[illegible] / 18250	797 / [illegible]	76.5
[illegible]	CO_2	[illegible] / 1520	-	47.1 / 1650	[illegible] / 3170	[illegible]
	SO_2	[illegible]	-	[illegible] / 1725	[illegible]	[illegible]
	(1) CO or (2) H_2O	(2) [illegible]	-	(1) [illegible] / 164	-	CO [illegible] H_2O [illegible]
3.	O_2	[illegible]	-	[illegible] / 851	[illegible]	[illegible]
	N_2	[illegible]	-	[illegible]	[illegible]	[illegible]

TABLE C-4 (Cont.)

Total for Case Nm^3/min / SCFM	Component	Gas Flow Nm^3/min / SCFM from Reverb.	Converter	Old Smelter	Total Nm^3/min / SCFM	Percent (Vol.) %
1238.6 / 43,640	CO_2	62.7 / 2220	-	47.1 / 1650	109.8 / 3870	8.9
	SO_2	34.3 / 1215	-	48.3 / 1725	83.6 / 2940	6.7
	(1) CO or (2) N_2O	(2) 78.6 / 2770	-	(1) 4.7 / 164	-	CO 0.4 N_2O 6.2

Note: Nm^3 measured at 0°C and 1 atm.
SCF measured at 32°F and 14.7 psia.

TABLE C-5

CALCULATION OF GAS MOLECULAR WEIGHT

Component	Case 1			Case 2			Case 3		
	Content (%)	Molecular Weight	Partial Molecular Weight	Content (%)	Molecular Weight	Partial Molecular Weight	Content (%)	Molecular Weight	Partial Molecular Weight
O_2	4.4	32	1.40	3.1	32	0.99	2.8	32	0.90
N_2	75.2	28	21.10	76.5	28	21.50	75.2	28	21.10
CO_2	[illegible]	44	3.12	8.7	44	3.85	8.7	44	3.85
SO_2	6.6	64	4.22	6.0	64	3.85	6.7	64	4.30
CO	0.3	28	0.08	0.5	28	0.14	0.4	28	0.11
H_2O	6.4	18	1.16	5.2	18	0.94	6.2	18	1.12
Total	100.0		31.80	100.0		31.27	100.0		31.30

TABLE C-6

CALCULATION OF AVERAGE GAS SPECIFIC HEAT

Component	Reverberatory Gas, Green Charge: Content	Av. Sp. Ht. ([illegible])	Partial	Reverberatory Gas, Calcined Charge: Content	Av. Sp. Ht. (0-[illegible]°C)	Partial	Old Smelting Gas: Content	Av. Sp. Ht. (0-150°C)	Partial
	%	cal/g mol °C, Btu/lb mol °F		%	cal/g mol °C, Btu/lb mol °F		%	cal/g mol °C, Btu/lb mol °F	
O_2	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	0.15	3.0	[illegible]	0.20
N_2	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	5.65
CO_2	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	9.2	0.67
SO_2	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	0.37	[illegible]	[illegible]	0.73
CO	-	[illegible]	-		[illegible]	-	[illegible]	[illegible]	0.05
H_2O	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	-	[illegible]	-
Totals	[illegible]		[illegible]	[illegible]		[illegible]	100.0		7.22

Heat Content: at [illegible]°C ([illegible]°F) above 25°C ([illegible]°F) — same as for Green Charge — at 150°C ([illegible]°F) above 25°C ([illegible]°F)

[illegible]

TABLE [illegible]

[illegible]

(Note: English units used for convenience because [illegible] data available in [illegible] theory [illegible] in [illegible] (Turbo[illegible]) in English units

[illegible]	Loose Charge	[illegible] Charge
[illegible]		
[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]
and [illegible]	Table [illegible]	Table [illegible]
[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]

[illegible]

TABLE C-[illegible] (continued)

Item	Steam Charge	Calcined Charge
[illegible] Calculate Heat Loss		
• Per Unit Pipe Area		
• Per Unit Length		
Heat flow across film at wall = heat flow to surroundings by radiation [illegible]	[illegible] = gas temperature; [illegible] = wall temperature [illegible]; [illegible] = wall temperature [illegible]; [illegible] = duct surface emissivity [illegible]; [illegible] = environment temperature [illegible]	
[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]
[illegible], Btu/hr ft²	[illegible]	[illegible]
Total Duct Length [illegible]	[illegible]	[illegible]
Surface per foot of length [illegible] ft²	[illegible]	[illegible]
[illegible] Flash Stack Heat Loss Calculation		
If heat loss per foot were constant then total loss [illegible] gallery entrance [illegible]	[illegible]	[illegible]
Gas temperature drop would be [illegible] °F	[illegible]	[illegible]
and gas temperature at gallery entrance would be °F	[illegible]	[illegible]

TABLE C-8

ESTIMATION OF TEMPERATURE AT [illegible] IN COMBINED [illegible]

Case 1

[illegible]

Component [illegible]

[illegible]

Average Specific Heat [illegible]

[illegible]

Gallery temperature [illegible]

TABLE C-8 (Cont'd)

Case [illegible]

Total Heat Content of Combined Stream [illegible]

[illegible]

Total Flow [illegible]

Average Specific Heat [illegible] Partial

[illegible]

[illegible] Temperature [illegible]

Case [illegible]

Total Heat Content of Combined Stream [illegible]

[illegible]

Total Flow [illegible]

Average Specific Heat [illegible] Partial

[illegible]

[illegible] Temperature [illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]	[illegible]
[illegible]	[illegible]
[illegible]	[illegible]
[illegible]	[illegible]
[illegible]	[illegible]

[illegible]

[illegible]

TABLE C-9

CALCULATION OF DRAFT AT LOCATIONS A, B, C, D
[illegible] Figure [illegible]

[illegible]	CASE [illegible]	CASE [illegible]	CASE [illegible]
1. [illegible]			
Temperature, °F			
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
Top of Stack	[illegible]	[illegible]	[illegible]
Flue Gas Rate [illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
Local Pressure, P, [illegible]			
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
Top of Stack	[illegible]	[illegible]	[illegible]
Ambient Temp [illegible] °F	[illegible]	[illegible]	[illegible]
°R	[illegible]	[illegible]	[illegible]

2. [illegible]

[illegible]

[illegible] - [illegible] draft [illegible]

[illegible] - height of [illegible]

[illegible] - average local pressure [illegible] height [illegible]

[illegible] - [illegible] weight of flowing gas

[illegible] - average temperature of flowing gas, °R

[illegible] - local ambient temperature, °R

TABLE [illegible] (cont'd)

ITEM	CASE 1	CASE 2	CASE 3

- [illegible] 0.192 [illegible]

  [illegible]

- [illegible]

  [illegible]

[illegible]	CASE 1	CASE 2	CASE 3
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]

TABLE C-9 (cont'd)

ITEM	CASE 1	CASE 2	CASE 3
Point A, cont'd			
d_i in wc			
K	[illegible] $\times 10^{-10}$	[illegible] $\times 10^{-10}$	[illegible]
T	[illegible]	[illegible]	[illegible]
L	[illegible]	[illegible]	[illegible]
A	[illegible]	[illegible]	[illegible]
V	[illegible]	[illegible]	[illegible]
SCEM P_n T_n			
[illegible] P f_n			
Subscript n normal			
SCEM (Table [illegible])			
d_i	[illegible]	[illegible]	[illegible]
Velocity change at top of chimney d_v			
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
d_v	[illegible]	[illegible]	[illegible]
Draft at A in wc	[illegible]	[illegible]	[illegible]
Point B			
Additional draft reduction	0	0	

TABLE [illegible] [illegible]

[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]			
[illegible]	[illegible]	[illegible]	
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]			
[illegible]			
[illegible]			
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]			
[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]

[illegible]

[illegible]

I. [illegible]

[illegible]

II. [illegible]

[illegible]

III. [illegible]

[illegible]

IV. [illegible]

[illegible]

[illegible] SO_2 [illegible]

TABLE [illegible]

[illegible]

[illegible] for [illegible]* [illegible]

[illegible]

1. New plant operating at [illegible]

[illegible]

2. [illegible] plant

[illegible]

[illegible]

[illegible]

1. [illegible]

2. [illegible]

3. Overall plant recovery [illegible]

4. As noted above, we assumed sulfur recovery ([illegible]) to be [illegible]% of [illegible] input of sulfur. This compares with recovery of [illegible] and [illegible] as derived by [illegible] in different sections of their February [illegible] calculation.

* [illegible] days/year

[illegible]

[illegible]

[illegible] INPUTS REPORTED BY [illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

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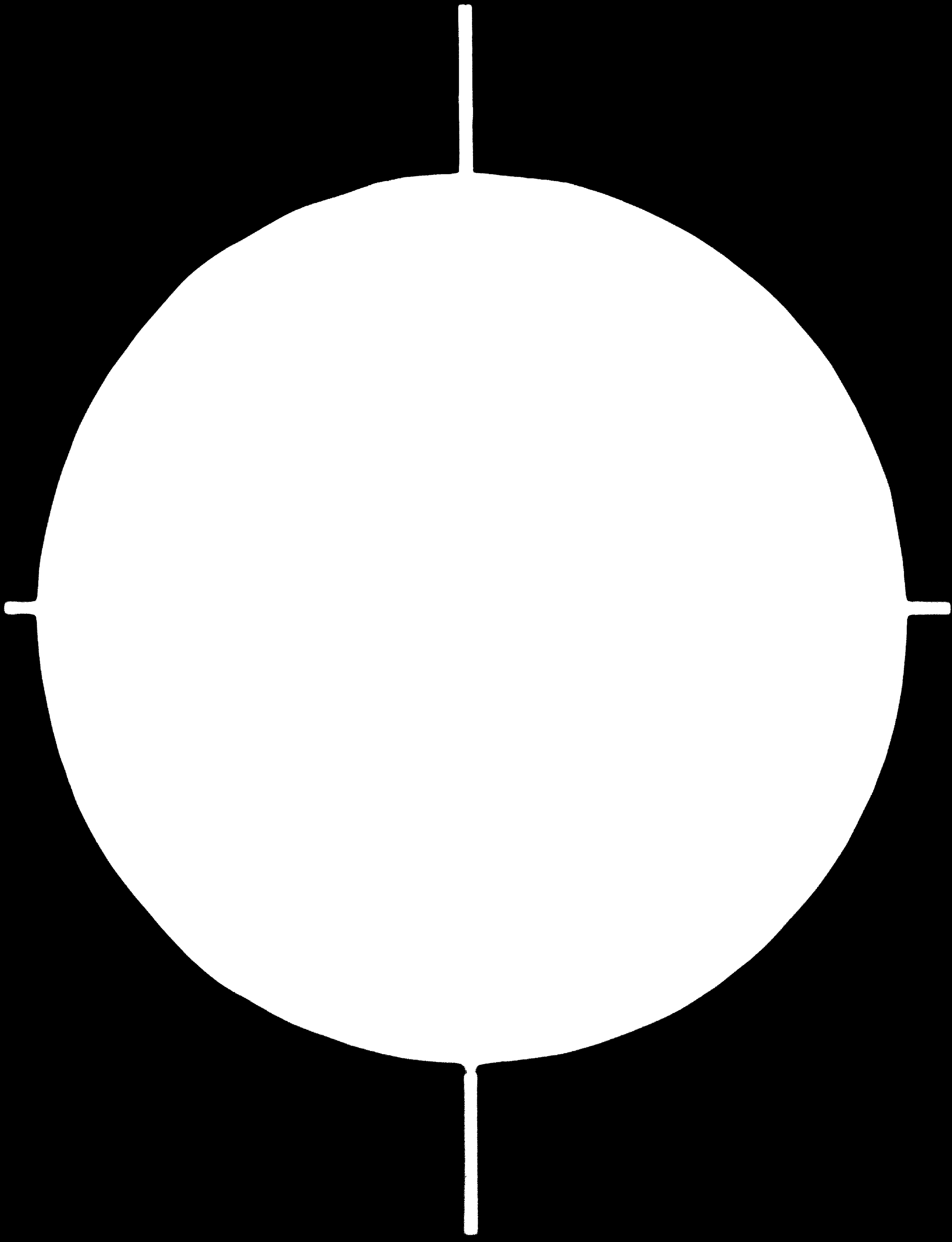
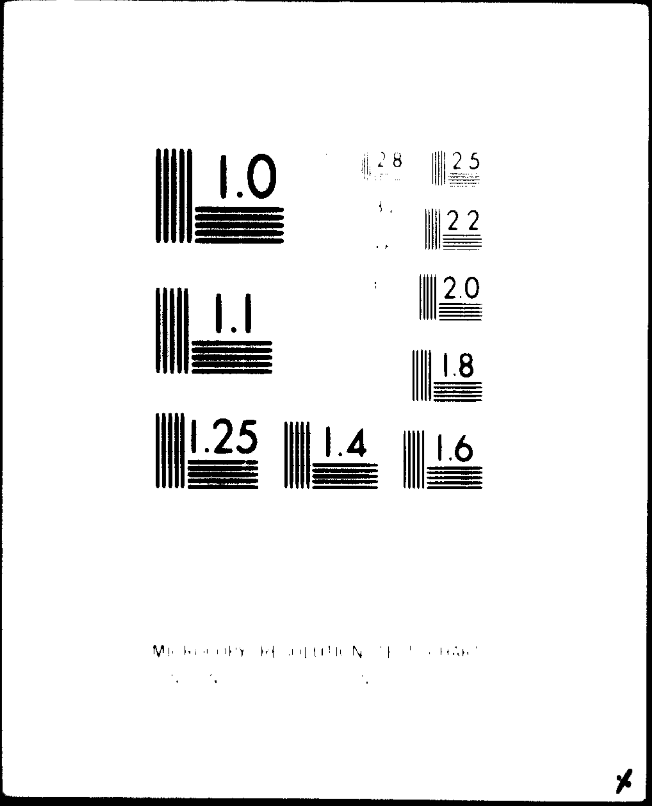




24x

E

D. **POSSIBLE SULFUR RECOVERY IF THE MATTE FROM THE WATER-JACKET FURNACE IS CONVERTED IN THE NEW CONVERTERS (assume 80% recovery):**

$$15{,}000\ \frac{\text{tons ore}}{\text{year}} \times 0.06\ \text{Copper} \times \frac{38.5\ \text{S in conc.}}{20.4\ \text{Cu in conc.}} \times 0.80 \times \frac{98}{32}$$

$= 4{,}160$ tons H_2SO_4 per year.

APPENDIX E

GUIDELINES IN STARTING UP THE FLUIDIZED BED ROASTER

I. INTRODUCTION

Since the supplier of the fluidized-bed roaster now installed at the Ergani smelter is no longer in business, the operating staff does not have access to assistance customarily supplied by equipment suppliers during start-up. Because the installation at Ergani is the first fluidized bed roaster in Turkey, the engineers and staff do not have alternative sources of background and experiences for a trouble-free start-up. On the other hand, we found the engineering staff to be quite capable, and accordingly we allocated a major portion of our effort during the second trip to the field to assisting and training them to develop an appropriate start-up procedure.

The procedure presented below is a condensed and abbreviated version of our detailed discussions with the engineers. Because of the imponderable factors involved in all plant start-ups, it should not be considered as a "cook-book" approach that can be followed without modification. Judgment based on experience should be applied to modify it as the need is identified.

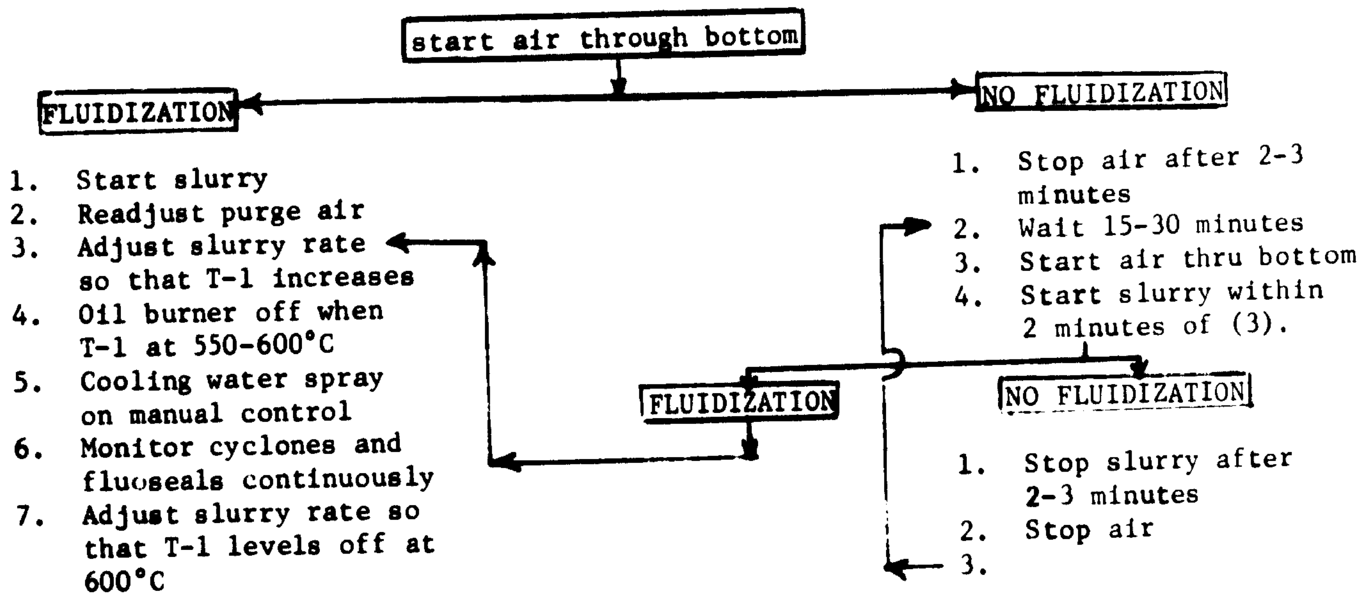
II. PROCEDURE

A. PRE-START-UP

1. Mix six to eight tons of dry 28-mesh silica flux with one to two tons of dry concentrate and charge into the roaster.

2. Inspect all instrumentation, pressure taps, adjust purge flows.

3. Start the oil-fired burner and heat the freeboard space in the roaster. Make sure that blower air does not enter the bottom of the reactor. Continue heating until the freeboard temperature is about 600°C.


start air through bottom
FLUIDIZATION
1. Start slurry
2. Readjust purge air
3. Adjust slurry rate so that T-1 increases
4. Oil burner off when T-1 at 550-600°C
5. Cooling water spray on manual control
6. Monitor cyclones and fluoseals continuously
7. Adjust slurry rate so that T-1 levels off at 600°C
NO FLUIDIZATION
1. Stop air after 2-3 minutes
2. Wait 15-30 minutes
3. Start air thru bottom
4. Start slurry within 2 minutes of (3).
FLUIDIZATION
NO FLUIDIZATION
1. Stop slurry after 2-3 minutes
2. Stop air
3.

APPENDIX F

UTILIZATION OF COPPER-RICH DUST FROM WATER-JACKETED FURNACES

I. PURPOSE

During the operation of the water-jacketed furnaces, the off-gases pass through a dust settling chamber before being vented to the atmosphere. Over the years, the dust deposited in the chambers has been stockpiled and this stockpile now amounts to about 50,000 tons of dust containing about 8% copper. The plant management is anxious to recover the copper content of this dust and requested ADL to assess the various methods for utilizing this dust.

II. APPROACH

We made approximate reverberatory furnace charge caculations on the basis that the dust would be charged to it in varying proportions during either green-charge smelting or calcine smelting.

III. SUMMARY AND CONCLUSIONS

A. CALCINE SMELTING

The calculations indicate:

- The grade of the matte will vary from about 20% Cu when using only dust to about 36% Cu when using only calcine.
- In order to maintain a daily blister production of about 60 tons/day, the amount of dust in a mixed dust-calcine feed could not exceed about 85%.

B. GREEN-CHARGE SMELTING

The calculations indicate:

- The grade of the matte will vary from about 27% Cu when using only a green charge to about 20% Cu when using 60% dust in a dust-green charge mixture.
- In order to maintain a daily blister production of about 60 tons/day, the amount of dust in a dust-green charge mixture could not exceed 60%.

C. CONCLUSIONS

When using either calcine or green-charge smelting, up to 85% or 60%, respectively, of dust could be added to the reverberatory furnace without decreasing blister output. In practical terms, this means that any reasonable amount of dust could be fed to the furnace within the limitations imposed by materials handling equipment. Because the dust is reportedly dry, the most appropriate way to introduce it into the furnace would be via the calcine bin and through the Wagstaff gun.

IV. DETAILED CALCULATIONS

The approximate analyses of the raw materials are:

	Dust	Calcine	Concentrate
Cu	8.0%	22.2%	21.0%
Fe	34.0%	36.4	31.6
S	12.8	18.0	37.6

Case A: Calcine Smelting

1. Grade of matte produced if only the dust is smelted:

   Basis: 100 kg. dust

   S to form Cu_2S: $8 \times \frac{32}{128} = 2$ kg.

   Cu_2S formed: $2 + 8$ $= 10$ kg.

   S remaining: $12.8 - 2 = 10.8$ kg.

   FeS formed: $10.8 \times \frac{88}{32}$ $= 29.7$ kg.

   Fe for FeS: $10.8 \times \frac{56}{32} = 18.9$

   excess Fe: $34 - 18.9 = 15.1$

   Total matte: $= \overline{39.7}$ kg.

   Matte tenor: $\frac{8}{39.7} = 20.1\%$ Cu

2. Copper production when dust is smelted at maximum smelting rate:

   Maximum fuel input into furnace is 105×10^6 Btu/hour.

   (Source: Parsons-Jurden Specifications 2.02)

   Assuming the fuel requirements are 3.5×10^6 Btu/ton of charge for smelting:

   Tons of dust smelted/day: $\frac{105 \times 10^6}{3.5 \times 10^6} \times 24 = \frac{720}{108}$ tons

   Less 10% $\simeq$ 612 tons

   Tons of 20% matte/day $= \frac{612 \times 0.08}{0.2} = 245$ tons

   Equivalent copper production: 49 tons copper

   Thus, the smelting of dry dust alone results in a decrease in blister production.

3. Grade of matte if only the calcine is smelted:

<u>Basis</u>: 100 kg. calcine

S to Cu_2S: $22.2 \times \frac{32}{128} = 5.5$ kg.

Cu_2S formed: 22.2 + 5.5 = 27.7 kg.

S remaining: 18 - 5.5 = 12.5 kg.

FeS formed: $12.5 \times \frac{88}{32}$ = 34.4 kg.

Fe for FeS: $12.5 \times \frac{56}{32} = 21.9$

Total matte: = 62.1 kg.

Matte tenor: $\frac{22.2}{62.1} = 35.8\%$ Cu

Because both the dust and calcine are deficient in sulfur, the matte grade could be obtained by linear interpolation when a mixture of dust and calcine is charged into the reverb as shown in Figure F-1.

4. Copper production when only calcine is smelted at the maximum smelting rate:

Assuming the fuel requirements are 3.5×10^6 Btu/ton of charge for calcine smelting; $\frac{105 \times 10^6}{3.5 \times 10^6} \times 24 = 720$ less 15% = 612 tons calcine per day.

Equivalent copper production = 135 tons/day.

The variation in blister production with varying ratios of dust and calcine in the charge is shown in Figure F-2.

<u>Case B</u>: <u>Green Charge Smelting</u>

1. Grade of matte produced when only green feed is smelted:

Basis: 100 kg. (dry basis) green feed

FIGURE F-1

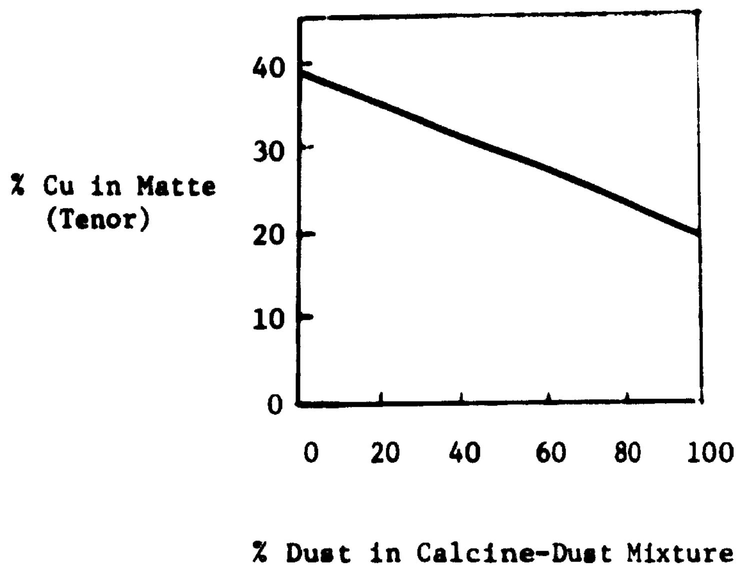
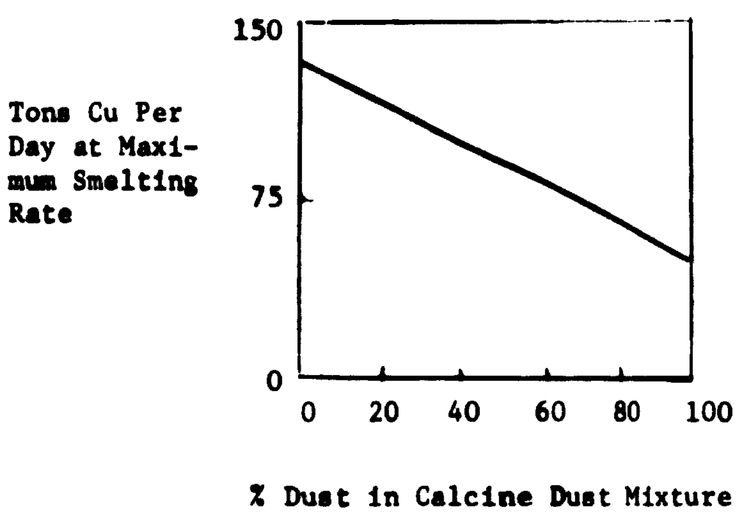


FIGURE F-2



Cu_2S: $21 \times \frac{160}{128}$ = 26.2 kg.

S to Cu_2S: $21 \times \frac{32}{128}$ = 5.2 kg.

FeS formed: $31.6 \times \frac{88}{56}$ = 49.7 kg.

S to FeS: $31.6 \times \frac{32}{56} = 18.1$

Total S in matte: 23.3

Excess S: 37.6 - 23.3 = 14.3 kg.

Total matte: = 75.9 kg.

Matte tenor: $\frac{21}{75.9}$ = 27.7% Cu

2. Grade of matte when smelting green charge and dust:

If 10% of excess S is lost and the rest combines with Fe in the dust: 14.3 - 1.4 = 12.9 kg. S $\times \frac{56}{32}$ = 22.6 kg. Fe required.

Each 100 kg. of dust has 15.1 kg. excess Fe (See paragraph A-1 above).

For the lowest possible matte grade, every 100 kg. of dry green feed will combine with:

$\frac{22.6}{15.1} \times 100$ = 150 kg. dust

or dust to green feed ratio is 150/100 = 1.5

or $\frac{150}{250}$ = 60% dust in a dust-green charge mixture.

3. Matte when 150 kg. dust and 100 kg. green feed is smelted:

Cu_2S formed: 1.5 x 10 + 26.2 = 41.2 kg.

Cu in Cu_2S: 1.5 x 8 + 21 = 33 kg.

FeS formed: 1.5 x 29.7 + 49.7 + (1.5 x 15.1 + 12.9)

= 44.5 + 49.7 + 22.6 + 12.9 = 129.7 kg.

Total matte: = 170.9 kg.

Matte tenor: $\frac{33}{170.9}$ = 19.3% Cu

This is sufficiently close to the matte grade (20.1% Cu) when only the dust is smelted (see paragraph A-1 above). Hence, increasing the amount of dust beyond 60% will not affect the matte grade. The variation in matte grade with different dust-green charge mixtures is shown in Figure F-3.

4. Copper production when only green feed is smelted at the maximum smelting rate:

   If the heat requirement in green feed smelting is 6×10^6 Btu/ton of charge, the amount of green feed smelted is:

   $$\frac{105 \times 10^6}{6 \times 10^6} \times 24 \times 0.75 = 315 \text{ tons dry concentrate per day.}$$

   Equivalent copper production = 66.1 tons/day.

5. Copper production when smelting dust-green charge mixtures as the maximum smelting rate:

   For a mixture of X tons of dust and Y tons of concentrate, the equation is:

   $$\frac{3.5 \times 10^6}{0.85} \times X + \frac{6 \times 10^6}{0.75} \times Y = 105 \times 10^6 \times 24$$

   when X = 1.5 Y, i.e. 60% dust

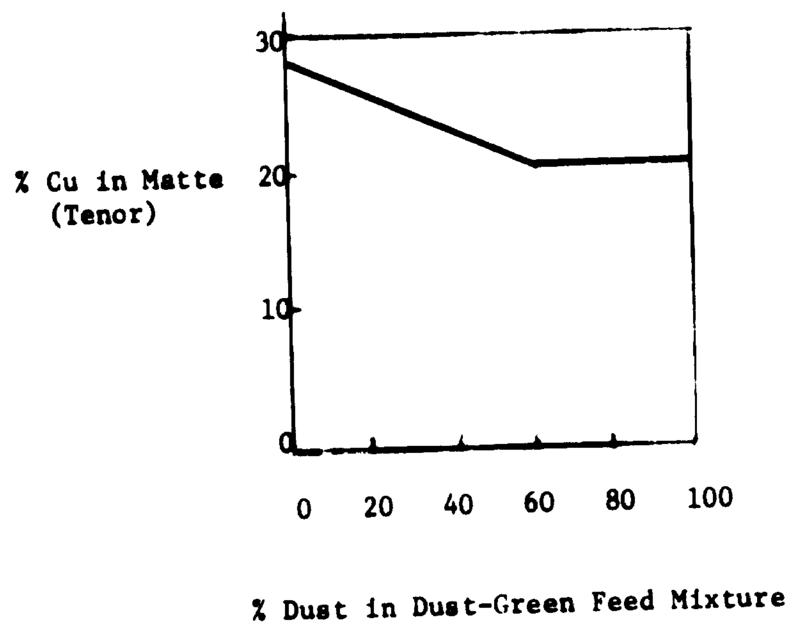
   $$Y \left(\frac{3.5 \times 1.5}{0.85} + \frac{6}{0.75}\right) = 105 \times 24$$

   $$Y = \frac{105 \times 24}{(6.19 + 8)} = \frac{105 \times 24}{14.2} = 178 \text{ tons}$$

   X = 267 tons

   Copper produced/day under these conditions (assuming converter capacity is adequate) = 178 x 0.21 + 267 x 0.08 = 38.4 + 21.4 = 59.8 tons

FIGURE F-3



% Cu in Matte
(Tenor)
30
20
10
0
0
20
40
60
80
100
% Dust in Dust-Green Feed Mixture

APPENDIX G

PROCEDURE FOR REPAIRING TAP HOLES

A. PROCEDURE FOR REPAIRING MATTE TAP HOLES

Matte can be tapped from the reverberatory furnace via one of three tap holes situated at different levels along the side of the furnace. The lowest tap hole is normally not used except when the complete draining of matte is required. The tap hole is lined with refractory brick along most of its length, except the outermost portion. The outermost section of the hole consists of a refractory block, embedded in a massive copper insert, which extends beyond and around the block so as to form a latch that prevents the refractory block from coming out, and at the same time is protection against chipping when bars are used. During operation, the tap hole is closed with moist clay to contain the matte. When matte is to be withdrawn from the furnace, the clay plug (which is mostly within the tap hole) is hammered out with a bar, or burned out with an oxygen lance. A new tapping block has a 35 mm diameter opening and is usually discarded when the opening reaches about 70 mm, or becomes difficult to close.

The tap block changing procedure is based on freezing the matte in its vicinity so that one can then remove the old block, repair the refractory, and place a new block while the furnace is still operating. Briefly, the procedure is as follows:

1. The old copper block is first cooled with compressed air or with an air/water mist. This can be applied either directly to the old block or by placing a circular copper form in contact with the block and then cooling the copper form. A fine air/water mist can be used for cooling but a direct water spray should never be used because of the danger of explosion. This procedure is continued for about 24 hours and longer if necessary, to freeze the matte in the tap hole. While the air-water mist is effective, it should be used carefully and experience concentrated on a few operators, because should steam penetrate the refractory, especially magnesite, the bond can be destroyed and the brick shapes disintegrated.

2. The block is removed slowly from the wall. This is a critical step, since there is danger of matte pouring out if the freezing has been inadequate. This can be gauged by the temperature of the metallic insert and by the appearance of redness in the adjoining refractory after the nozzle has been removed. If any redness shows, the refractory should be cooled with compressed air or air/water mist.

3. After the block is removed, the brick work can be inspected, repaired if necessary and smoothed in order to receive the new block. The new block may be wrapped in asbestos paper and pushed or hammered into place. Alternately, powdered magnetite can be used to seal the gaps between the new block and the brick work. See Figure G-1.

B. PROCEDURE FOR REPAIRING SLAG TAP HOLE

Because the slag notch is cooled with a water jacket, the water jacket and its adjoining area is cool and suffers minimum thermal damage. Usually, damage can occur to the refractory behind the slag notch which is exposed to the furnace heat and the hot slag. Briefly, the procedure for replacing this damaged refractory is as follows:

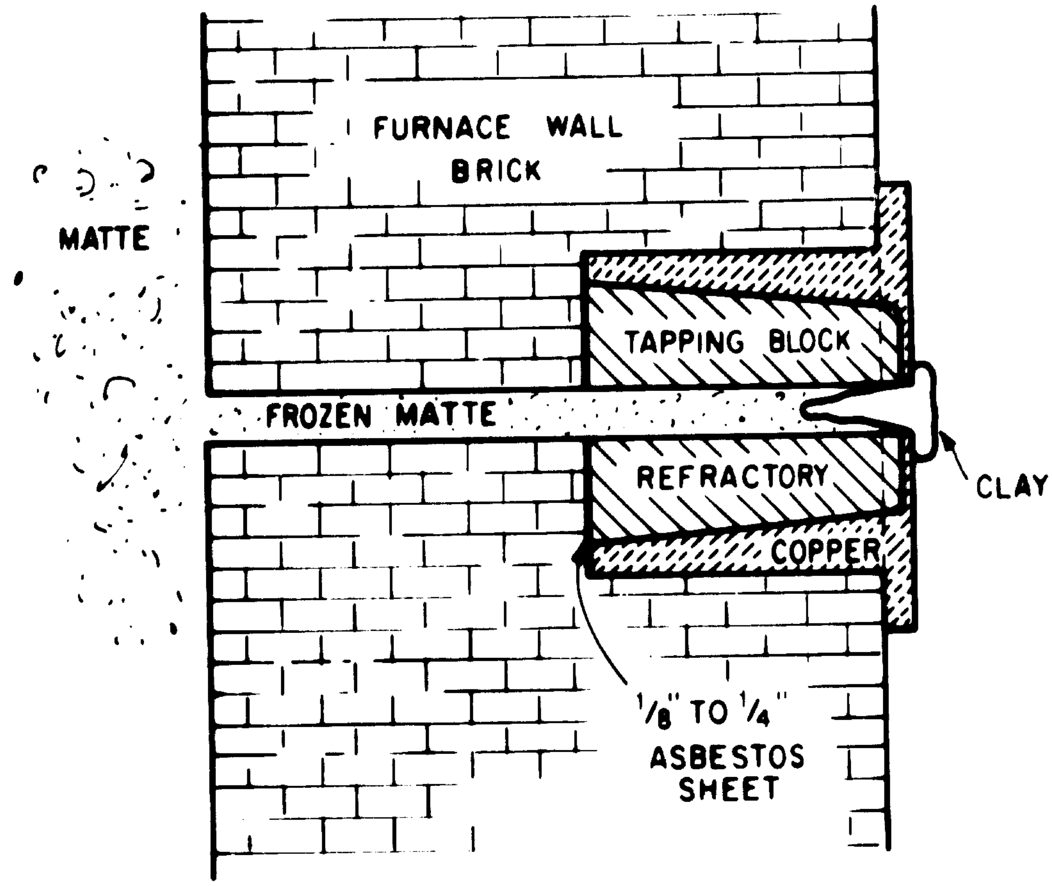
1. All the slag is tapped or skimmed from the furnace, after which most of the matte is tapped.

2. The burners are shut off. It is not necessary, however, to let the furance cool down before starting these repairs.

3. A protective wall of clay is built inside the furnace around the slag notch. This can be done by throwing blobs of clay from the roof. The wall has to be high enough for the bricklayer to work on the repairs without being exposed to excessive amounts of radiant heat from the furnace cavity.

4. Once the clay pile is complete, the old brick can be hammered out and replaced with new brick.

5. The temporary clay wall is demolished and the clay is removed via the slag notch before resuming normal furnace operations.



REFRACTORY GROUTING IS PLACED AROUND THE COPPER BLOCK AND WALL JOINT.

THE REFRACTORY BLOCK IS BEDDED BY CASTING THE COPPER AROUND IT.

CROSS SECTION OF A TYPICAL MATTE TAP

FIGURE G-1

CAMBRIDGE,
MASSACHUSETTS

CHICAGO
NEW YORK
SAN FRANCISCO
WASHINGTON
ATHENS
BRUSSELS
CARACAS
LONDON
MEXICO CITY
PARIS
RIO DE JANEIRO
TORONTO
ZURICH

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