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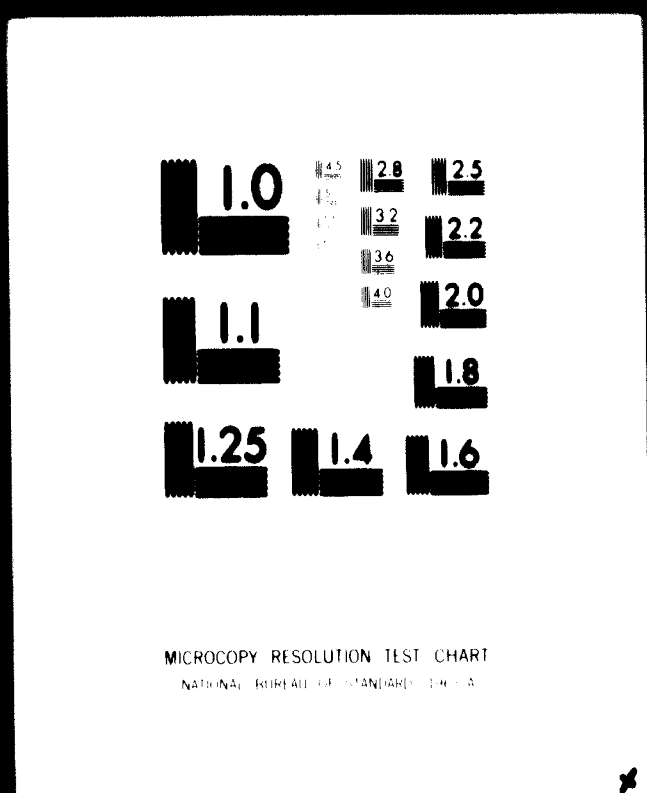

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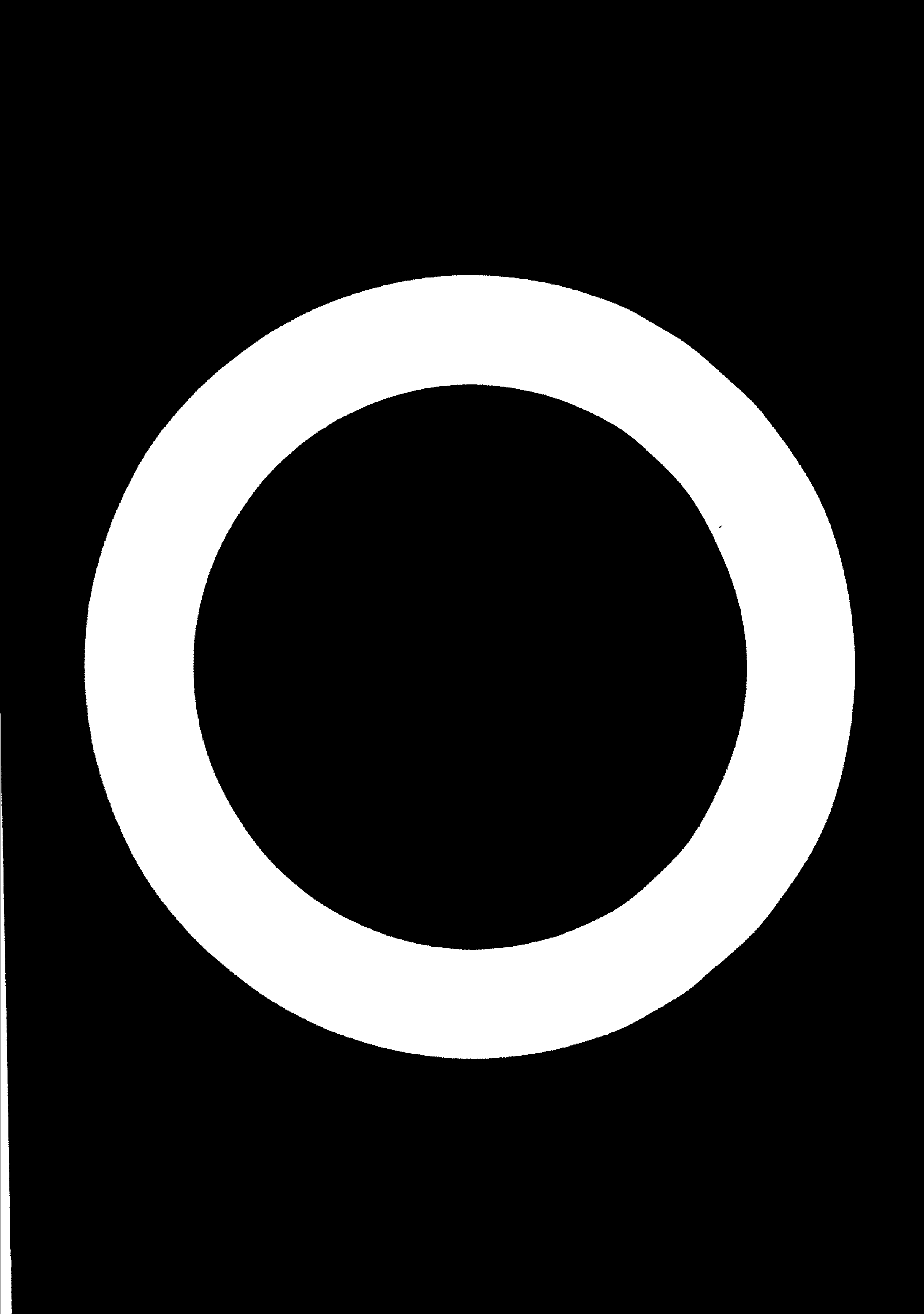
All these persons have made possible the preparation of this Report with their assistance and have helped to make it a pleasant and fruitful stay for Tecniberia's team in Turkey.



ABSTRACT OF THE REPORT

This reports covers the survey requested by the Government of Turkey to UNIDO on technical assistance in mercury extraction, being ETIBANK the specific Turkish Government agency concerned with the project.

The aim of the survey is to study the possibilities improvement of technology, plant and equipment maintenance and increasing the efficiency of mercury recovery by rotary kilns at the existing Haliköy and Konya smelting plants, together with the training of counterpart personnel in mercury extraction.

In order to fulfill the above objectives a thorough study of the present operating conditions of both metallurgical plants has been performed, together with a series of tests in order to assess present efficiency in mercury recovery.

The results of this study have revealed the following:

- The Konya plant although well projected from a metallurgical point of view, presents some deficiencies from mechanical, metallurgical or process control origin that have been analysed, and should be corrected in order to achieve the best results in accordance with the specific recommendations of this report.
- The Haliköy plant bases its operations in an orebody that seems to exhibit sufficiently good characteristics to advise the complete reconversion of the present Haliköy's metallurgical plant facilities, now out dated if we except the kilns and some auxiliary facilities.

The need for such a reconversion and the fact that the mine workings are affecting the surface of the present plant location by ground settlements would advise the construction of a

new plant at a different location, but this decision should be ac comodated to more favorable market circumstances.

The recommendations included in this study on the problems existing at Halikōy should be adopted taking into account the estimated future situation of the mercury market and the final criteria to be adopted by Etibank's Management on the implementa tion of the expansion programme for Halikōy mine and mercury re covery plant.

GENERAL INTRODUCTION

Background information

Turkey has been a significant source of mercury for over 50 years.

The importance of the industry lies in its export potential. The output and trade for mercury in 1.969 is reflected in the following figures (in metric tons).

Production	Consumption	Net trade	Value ($ million)
203	14	108	2,8

Etibank (the state mining enterprise) dominates the mineral industry of the country, for in 1.969 it produced all the blister copper, 56% of the chromite, 37% of the broon minerale and 63% of the mercury.

Etibank in one of Turksy's State-Owned Economic Enterprises which has a world-wide reputation in mining. Its capital of 500 million Turkish liras is fully paid up. Etibank's assets at the end of 1.969 stood above 5,4 billion Turkish lirae.

Etibank is a profit making State-Owned Economic Enterprise. It turns over to the State Treasury a considerable share of its yearly profits and reinvests the surplus profits provided by its own sources, thus ensuring to a major extent the development of Turkey's economy. In 1.969 it invested more than one billion Turkish liras in its power producing and mining installations.

Etibank produces chrome ore, blister copper, ferro chrome, prime mercury, refined sulphur, cupreous pyrites, lead and zinc concentrates, sulphuric acid, borax and boric acid and besides meeting the demand for domestic consumption, it exports chrome ore, blister copper, prime mercury, colemanite ore, cuprous pyrites, borax, boric acid and ferro chrome.

Etibank's activities include the building of power plants and the transmission lines and distribution of the power to various parts of the country.

Etibank also performs banking services since the year 1957. It has banking branches in Turkey's chief towns and commercial centers.

At present, Etibank has seven fully owned subsidiary mining establishments and operates four mining works, it has also seven hydraulic and four thermal power plants. At the end of 1969, the number of its banking branches has risen to thirty nine. Etibank is also a shareholder in twenty seven financial, commercial and industrial companies.

At the end of 1969, the total nominal capital of the companies of which Etibank possesses 15% or more shares amounted to 3.369.292.500 -Turkish liras. Etibank's share of this capital is -- 897.163.505 -Turkish liras.

Four of the companies of which Etibank is a partner have been founded with the participation of foreign capital.

The share of Etibank in the mercury industry has increased steadily since 1965, when total exports of 73.2 tons of mercury were performed 54% by private mines and 46% by Etibank. This proportion was reserved in 1969, 37% being performed by private enterprise and 63% by Etibank.

The total number of private mines working in 1969 were 11, while only 2 (Konya and Haliköy) were operated by Etibank. The high price of mercury over the 1965 - 1969 encouraged Etibank to increase capacity (including the reopening of the old Konya mine in 1969) and the State Planning Organization, SPO, to back a private mercury development company. However the price for mercury receding back continuously from a high of $ 536 in 1968, has made necessary the revision of the rather ambitious development plans established in 1970/1971.

These plans included expansion programmes for the aforementionned existing plants and studies for the establishment of a new mercury mining and smelting plant.

In the frame of this programme the Government of Turkey through the UNDP Resident Representative, submitted a request to UNIDO for technical assistance in mercury extraction, being Etibank the specific Government agency concerned with the project.

Objectives of the study

The primary objectives of the project, as described in the Substantive Terms of Reference attached to UNIDO'S invitation and on which Tecniberia's proposal was submitted were:

The improvement of the existing mining and smelting operations, at Haliköy and Konya as well as a suggesting an action related to the future development of the mercury industry of the country.

The specific tasks to be implemented were the following:

1. Regarding the existing facilities at Haliköy and Konya

- Study and assess the operating efficiency of the plants with particular regard to the recovery of mercury in rotary kilns. The assessment should include technical and economic aspects.
- Based on the assessment of the existing operations, elaborate specific recommendations leading to improvement of technology, plant and equipment maintenance, and to increased efficiency of recovery of mercury.

- Train counterpart personnel in mercury extraction, providing on the spot advice and guidance and/or short duration courses.

2. Regarding the development programme

- Based on the contractor's experience and taking into full consideration the existing local conditions, advise the Mining, Smelting and Associated Industries Project and Implementation Development on the expansion programme for the Hali Koy and Konya mercury mining and smelting plants. The advice should also include suggestions regarding the type of additional studies required.

- Review the available information and data on the establishment of a new mercury plant and advise on elaboration of additional studies, if any.

Nevertheless, and taking into account the present situation of the world mercury market the Goverment of Turkey decided to give up the expansions of the existing plants together with the development programme for the stablishment of a new mercury plant, for the time being.

Also it was considered by the Turkish authorities that the termsof reference for this survey should be purely on technical matters related to the mercury recovery and should not encompass any economical matters nor the assessment of technical end economical aspects of the mining methods and mercury potential of the mines.

Such amendments to the terms of reference were included in the letter sent by Etibank to UNDP, dated March 31, 1.972 and commenting on the draft contract between UNIDO and Tecniberia.

The proposed alterations to the contract were discussed and agreed at the arrival of the second mission of Tecniberia at Project area and the aims of the study were restricted to:

A) Improving technology, plant and equipment maintenance and increasing the efficiency of mercury recovery by rotary kiln at the existing Haliköy and Konya smelting plants;

B) Training counterpart personnel in mercury extraction.

Scope of the study

The scope of the study was based in the accomplishment of the project aims stated in the above paragraph. Therefore the following services have been provided:

A. Data Collection

Collect basic available data on the technical aspects of the existing mining and smelting plants of Haliköy and Konya. In order to collect such data, a team of two (2) Contractor experts visited the Project Area for one week. After completion of the data collection and its analysis in the Home Office, the Contractor's team returned to the Project Area to accomplish the tasks set forth in sub-paragraphs B through D inclusive, below.

B Mining Methods

Study ore breaking, ore loading and ore transportation and provide recommendations for their possible improvement.

C. Smelting

Analyze existing smelting operations and assess present efficiency in mercury recovery as follows:

1. <u>Technical Aspects</u>

a) Nature of Treated Ore

(1) Mineralogical Composition

(i) Determination of ore components.

(ii) Mercury content and ore type (cinnabar, metacinnabar, tetrahedrite, etc.).

(iii) Content in other minerals, especially the arsenic bearing ones (orpiment, mispickel, etc.) and pyrites, galena, blende and the like.

(iv) Type of gangue and related problems (abrasion).

(2) Granulometry

Maximum size of mine run.

(3) Humidity

Humidity content and related problems (fuel consumption, kiln feeding and material flow).

b) Operating Process

(1) Treated Ore and Gas Analysis and Sampling

Determine by analysis and sampling, the CO_2 content in kiln exit gases and the mercury content of the following:

(i) Treated Ore

Perform sampling in the kiln's feeding mechanism and/ /or in the conveyor to the kiln feeding bin utilizing the usual analysis methods of Eska or Colorimetric. Per several daily analyses if arsenic is present.

(ii) Calcined Ore

Since it must be controlled daily, take one sample each hour.

(iii) Stack Gases

Perform representative sampling correctly since this item is the main source of mercury losses. Analyze the mercury in stack gases in order to estimate the stack losses.

(iv) Dust

Analyze the cyclone dust to determine its mercury content. Determine the losses by the collected dust (tons/day) x ‰ Hg.

(v) Waste Water

Analyze the mercury lost in waste water.

(vi) CO_2 in Kiln Exit Gases

Analyze the CO_2 content in exit gases in order to control the combustion process. (Its content in exit gases should not be greater than 10-11% CO_2).

c) Temperature Control in Gas Circuits

Taking into account the difficulties presently experienced in measuring calcining temperatures, examine and assess temperatures at the following gas circuit points:

(1) At Kiln Exit (T_1)

(2) At Condensing System Inlet (T_2)

(3) At Condensing System Outlet (T_3)

(4) At Stack Inlet (T_4)

d) Pressure Control

Determine and assess pressures at the following points:

(1) Kiln Exit (P_1)

(2) Cyclone Gas Exit (P_2)

(3) Exhausting Fan-Section (P_3)

e) Laboratory Control Form

Record daily the main control figures shown above in order

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to maintain proper control of the mercury recovery process on a laboratory control form.

f) Materials Balance

After analysis of the above data, establish the material balance of the process in order to assess the technical efficiency of mercury recovery of the installation.

g) The above assessment shall be based upon measurements made with existing instrumentation on site.

2. Specific Recommendations

Taking into account the results of the above assessment of the plant's operating efficiencies in mercury recovery, provide specific recommendations on possible improvements in the following areas:

a) Technology and Mercury Recovery
Gas and fuel circuits and mercury recovery

b) Plant Overall Operation
Mechanization, automation, gas tightness and plant personnel protection against intoxication.

c) Equipment Maintenance
Floating movement of rotary kiln, refractories, mechanical maintenance and corrosion problems in the condensing system.

D. Personnel Training in Mercury Extraction

After the present technical efficiency of the mercury metallurgical plants is determined, give on the spot advice in order to improve the present situation, provide patterns of correct plant operation together with a short course to train counterpart personnel in mercury extraction. Place emphasis on control procedures for:

a) Treated Ore

b) Calcined Ore

c) CO_2 at Kiln Exit Gases

d) Temperatures

e) Pressures

f) Gas Circuits Tightness

Methodology of the study

Taking into account that two metallurgical plants, Konya and Haliköy had to be studied, the time factor was a basic element in this instance and thus it was necessary to give special attention to the methodology in the performance of the study.

The accomplishment of the project was divided into stages of specific nature, the first one of them being used as a basis for the two next ones. Detailed programming of each stage was carried out with great flexibility, establishing weekly programs, with a detailed schedule taking into account the number and diversity of actions simultaneously in process.

After a first stage of one week from 6 March through 12 March 1972 at the Project area with the purpose of collecting basic available data on the existing mining and smelting plants of Konya and Haliköy and familiarisation with the problems to be studied, questionnaires (see Annexes A and B) were submitted to the Turkish counterparts in order to be fully completed and the first mission returned to Spain to perform the analysis of the collected data in the home office and to prepare the working program for the next stages which were devoted in Turkey to:

a) Research

Stages at the Konya and Haliköy plants, performing adequate - tests to analyze existing smelting operations and assess present efficiency in mercury recovery.

b) Analysis of research

Study of the information gathered through research andestablishment of preliminary conclusions and recommendations.

General structure of the report

The general table of contents clearly shows the general - structure of this report very much in agreement with the contract, previously summarized in this Introduction.

Another factor which determines the order of exposition was the basic principles of report writing, specified in Annex B to contract, which we have tried to observe at all times.

Section I, Introduction, includes: Team Composition, Acknowledgements, Abstract of the Report and General Introduction.

Section II, includes Problems and Recommendations.

Sections III, IV and V are the body of the report.

Section III, includes a general description of the present installations of the Konya and Haliköy plants, starting with a summary of the mining operations, followed by the present characteristics of the metallurgical plants, including plant description with flow sheet of the processes, operating data, equipment data, present operating conditions and process control.

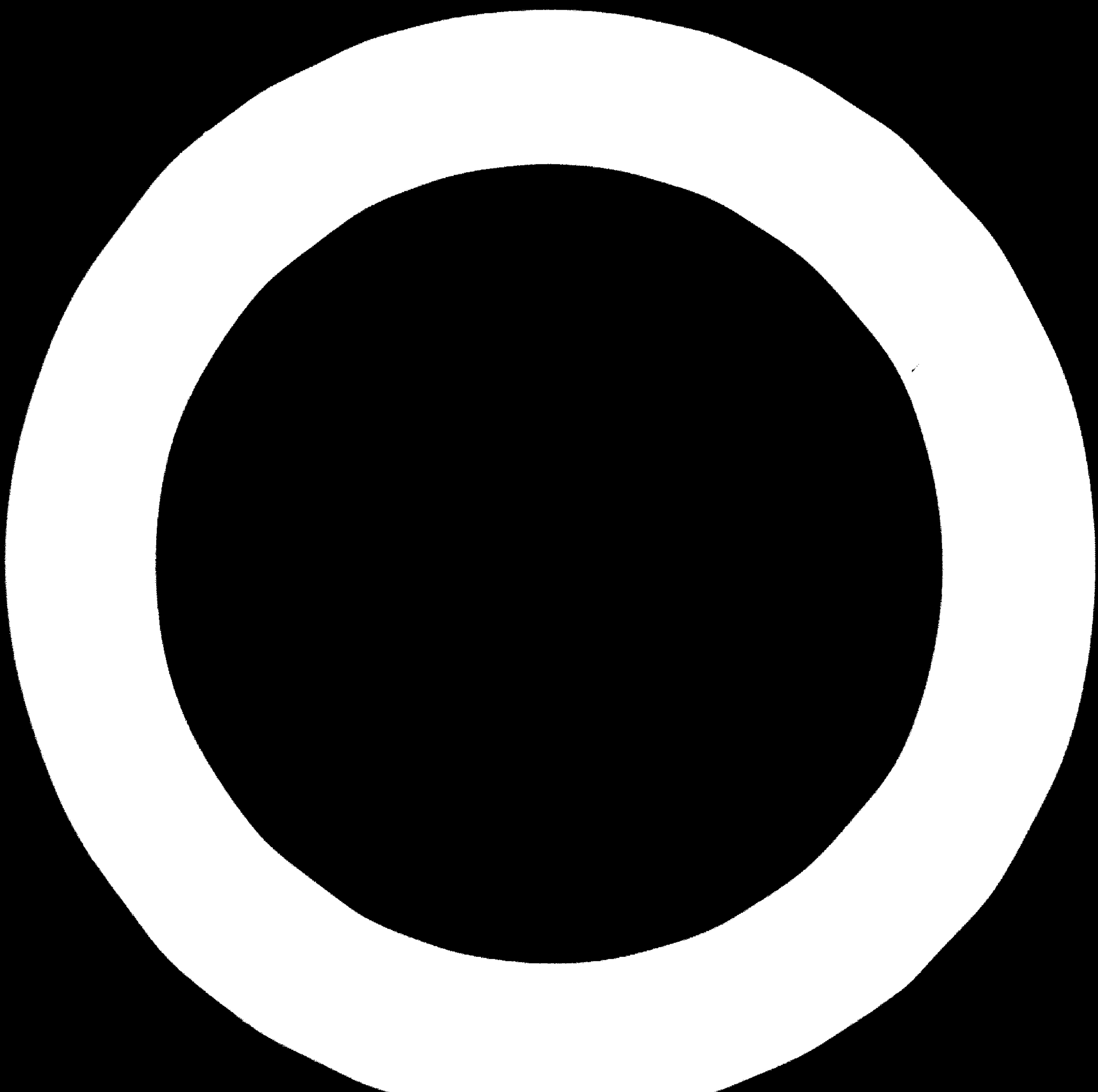
Section IV deals with the development and results of the performed tests at both metallurgical plants.

Section V summarizes the action followed for personnel training in mercury extraction practices.

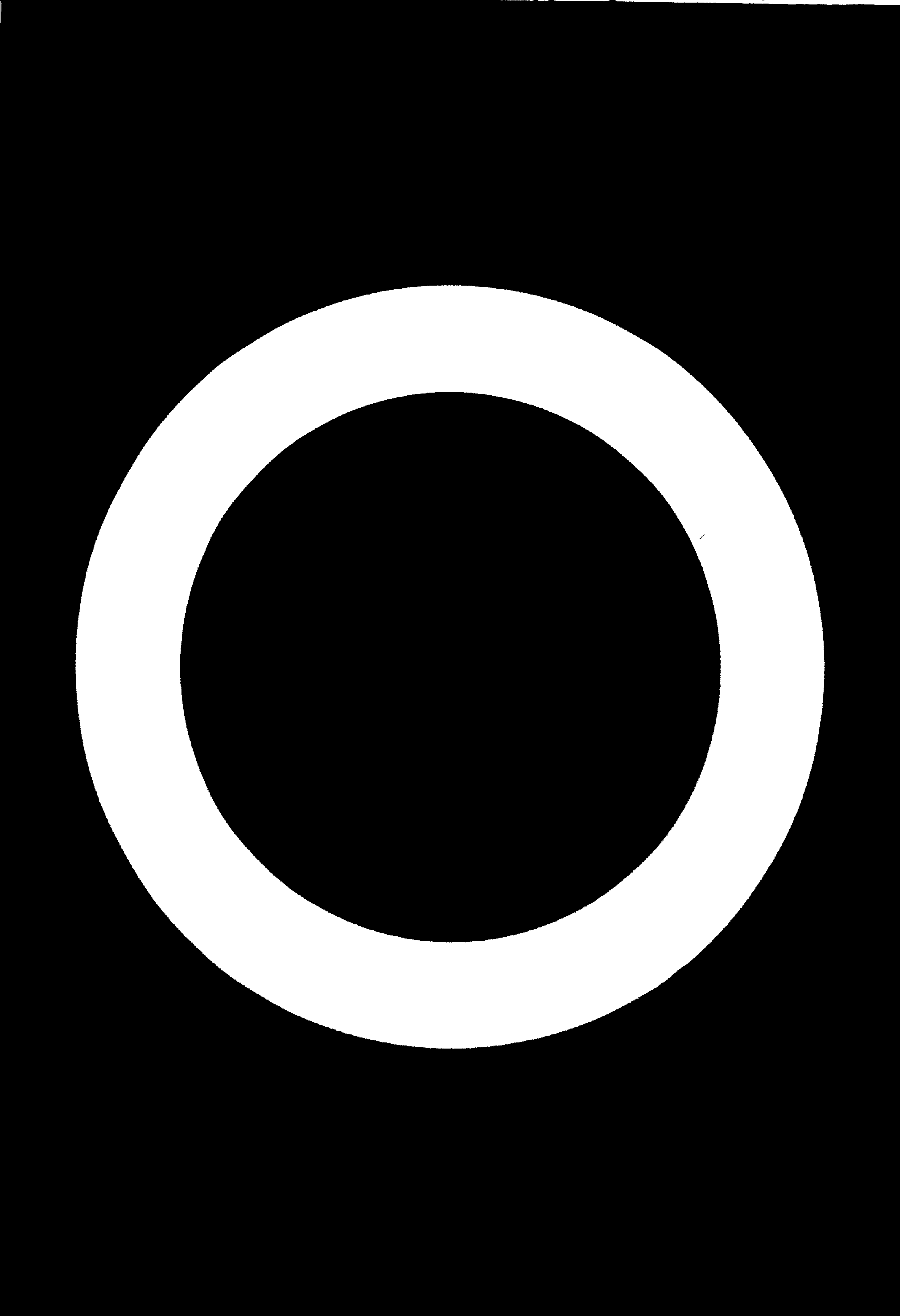
Taking into account the specifically practical nature of the objectives of this study, its body has been discharged, as much as possible, from complementary material which has been gathered in form of Annexes in Section VI of the report.

Section VII is the terminal section of the report, reenforcing the conclusions drawn and the recommendations made, supplemented by a selected bibliography on mercury technology.

Statistical data, graphs and tables are displayed for convenience and more accesibility, at the end of each corresponding Section.



II. PROBLEMS AND RECOMMENDATIONS



II.1. KONYA

A. Mining

A.1. Problems

During the First Mission to Project Area the mining operations of the Büyük Oçak mine of the Sizma dis trict were visited.

The main problems faced by mining operations in that orebody are:

- Very irregular mineralisation of the orebody, with small bonanza areas and large mass of waste.
- Few prepared reserves
- Lack of orebody investigation
- Absence of mechanization
- Excess of labor
- Excess of subordinate and administrative personnel

A.2. Recommendations

The first four aforementioned problems are clo sely tied together and their relationship it established by the nature of the orebody.

Its lack of regularity makes necessary the establishment of an accelerated rhythm in the preparation of reserves and investigation of new areas.

In order to maintain the labor force of the mine within reasonable limits it is compulsory to establish the maximum possible mechanisation of the operations to allow the winning of the necessary ore tonnage for the ope ration of the metallurgical plant.

Therefore, our recommendations are:

1. To increase the geological studies and number of drillings in order to perform only in waste areas such workings with possibilities of striking mineralizations.

2. To purchase modern drilling equipment suitable to be operated by a single worker.

3. To purchase small loading shovels, to get a reduction in excess labor presently working in the loading of ore and waste into wagons.

4. Transport problems are, in our opinion, much more difficult to solve, as this would need the modification of the mine's infrastructure.

5. The need for the introduction of deep changes in the present organization of the labor set up at the underground workings is evident, if Konya mine is compared with other Spanish mines of similar irregular mineralization, as shown below:

	Konya	Other mines
Tons of ore/day	200	200
Labor force	418	150
Hours per man and day	6	7
Working days per year	300	280
Total hours/year	752.400	294.000
Kg. of ore per man hour	79	190

The above differences in efficiency have been reached by an intensive mechanization, this being, on the other hand, the only solution that will allow the mines to bear the strong increases in the labor cost.

6. Concerning administrative and subordinate personnel their number is very high at Konya, as shown by the following figures:

	Konya	Other mines
Total payroll	694	187
Administrative	75	6
Total payr/admin. ratio	9	31

The 1/30 ratio is normal in Spanish mines. Therefore, at Konya it will be necessary:

- To eliminate excessive number of forms.
- To centralise at Ankara payroll, orders, accounting and other services.

It is important to realize that once the present mercury crisis is over the price curve will tend to some kind of stabilization, but the labor cost curve will be strongly growing, and therefore if the proper corrective measures are not taken, labor cost increments will have a great unfavorable repercussion on the mercury extraction industry.

The technical personnel at Konya is totally qualified to study specific solutions to the aforementioned problems, that should be carefully studied taking full account of local conditions, e.g., impact of such labor force reduction on employment situation in the area. However, it deserves close attention on the part of local management.

B. Metallurgical Plant

The problems of the Konya metallurgical plant

may be classified in three main groups: metallurgical, mechanical and process control problems.

B.1. Metallurgical problems

We will include in this group all those problems that may give place to a defective utilization or to a low efficiency of the plant.

The fundamental metallurgical problems arise from:

- Granulometry of feed ore
- Defective feeding and roasting of the ore
- Frequent slag formation
- Handling of cyclone dust
- Handling of stupps at the Exeli hoeing machine

In general the aforementioned problems are not of an independent nature. Therefore the recommendations proposed below are interrelated in some aspects and should be adopted taking into account their mutual existing relationship.

B.2. Recommendations on metallurgical problems

This is one of the factors that gives place to strong unbalance in the operating process.

During our stay at the plant we recommended to perform granulometric analysis of the feed ore, that had never been made before.

Reported results of dry screen analysis of the ore sample show that 36.6 per cent of the ore is below 4 mm.

A so high percentage of fines in the ore causes a defective feeding of the kilns, a reduction in the operating capacity of the same, frequent slag formation and a large drag of dust towards the cyclones and the condenser with the consequent leaning out of mercury in the stupps and defective mercury recovery in the hoeing machine.

All these aspects emphasize the interest in doing the best with the presently available facilities to palliate the problems proceeding from the granulometry of the ore. A final solution can only be found with a total change of the present ore treatment.

Therefore the objectives to be achieved with the presently available facilities are:

a) not increasing the percentage of fines in the feed ore by improper crushing
b) to feed the kilns with the largest possible ore size

The present crushing system does not meet the first objective, as can be seen in the flow-sheet of the process (see drawing III-1, Section III).

The system has the drawback of passing all run of mine ore though the primary crusher (4) and the 40 mm screen (5). Therefore, sizes over 40 mm are crushed again at the secondary crusher (6) with the result of a substantial increase in the total content of fines in the crushed ore.

We recommend in order not to increase abnormally the proportion of fines, to modify the present crushing system as follows:

- To remove or by-pass the primary crusher (4).
- To convey the run of mine ore directly to the screen (5), increasing the opening of this screen above 80 mm.

- To regulate the exit size of secondary crusher (6) in such a way that the coarse ore size will be above 80 mm.

The size of 80 mm for the coarse ore is merely indicative, as the second important objective to be met is to achieve the feeding of the kilns with the largest possible ore size. This optimum size can only be found by the trial and error method, and after a careful control during a proper length of time of the following factors:

a) Tonnage of calcined ore with different sizes
b) Content of mercury in the burnt ore or "calcine"
c) Stoppages by slag formation

If the behaviour of the feeding system and the kilns are satisfactory with a given size of coarse ore during a sufficient length of time and factors a), b) and c) are also favorable during the trial period, ore size should be gradually increased up to maximum possible limits -- through the proper action in screen opening and secondary crusher.

We anticipate that it will be possible to feed ore up to 120 mm.

To study a conclusive solution it is necessary to perform a systematic study of the correlation existing between ore size and mercury grade, during about a -- month's time and establishing the following tabulation:

	% Weight	% Hg
• Ore size > 2 mm.	A	H
• Ore size ≤ 2 mm.	A'	H'

The results of the above tests might lead to study a different process for the processing of the fines.

Summing up, we may state:

- There is a transitory solution which would partially attenuate the effects of the excess of fines, with slight modifications of the presently existing crushing circuit.

- It is convenient to make a systematic study of the correlation between granulometry and mercury grade of the ore. The results of such study may give place to the convenience of a new installation for the treatment of the fines, once the present mercury crisis will be over.

B.2.2. Defective feeding and roasting of the ore

The defective ore feeding to the kilns give place to a low capacity utilization of the plant. The difficulties found in the ore feeding are mainly due to a bad connection between the shaker feeder and the 50 t. surge bin. Nevertheless, when the feeding rate is increased the following problems have arisen:

- Rejects of ore through the entry side of the kilns take place.
- Calcines show an increase in mercury content.

Therefore the solutions recommended below, must take into account the above difficulties.

The proper feeding rate will be attained with the modifications in the connection between the surge bin and the feeder, shown in drawing E-1. The reject of ore will be totally overcome by an extension in length of 1.20 m. of the feeder and providing to such extension a small increase in diameter.

Therefore, with a more regular feeding of ore and avoiding ore rejects, we shall get a deeper bed of ore in to the kiln.

To achieve a good roasting is of paramount importance, as the result of the tests have shown that normally at Kenya, mercury losses in the calcines are much higher than it was suspected.

Nevertheless, to get satisfactory calcines is a compromise between technical and economical problems in order to reach a satisfactory balance among:

- Feed rate
- Rotating speed of the kiln
- Fuel-oil consumption
- Mercury content of calcines

It is of paramount importance to achieve a proper metallurgical process that the feed rate is normally performed. If such is not the case (or if the feeder gets empty), unbalance arises in the condensing process together with air inlets into the system.

The operating process, once introduced the aforementioned changes in the feeding system, will be conducted as follows:

- The kiln will start its operation at the present feeding conditions and fuel rate, reducing gradually the rotating speed of the kiln until rejects of ore take place, through the feeding end of the kiln. The minimum speed compatible with the absence of ore reject will be maintained.

- During three or four hours calcines will be controlled for mercury content.
- If such content is satisfactorily low, the feed rate and the rotating speed of the kiln will be both increased gra dually, without changing the fuel input, until getting a balanced operation (always with a proper Hg content in the calcines) by successive trials.
- If the mercury content of the calcines shows an increa se, fuel oil input would be increased (watching careful ly CO_2 content in exit gases and possible slag forma- tion). Should the mercury content in calcines not decrea se, then the feed rate and the kiln speed must be decrea sed.

  To get satisfactory results, with such tests, continuity is a must and, evidently, rapid results from analysis at the Laboratory.

B.2.3. <u>Slag Formation</u>

This is a rather common problem at Konya and the main reasons for its existence are:

- Low fusibility of the gangue
- Concentration of the high temperature zone in a very short length of the kiln
- High content of fines in the kiln

Slag formation will be reduced by feeding the kilns with a coarser size ore and by avoiding the feed-back of cyclone dust to the kiln

B.2.4. Cyclone dust

Handling of cyclone dust is done at Konya, by conveying the dust though a screw-conveyor to the calcines' bin, where the sensible heat of the calcines is used to distill the mercury content of the dust.

As an average of 1.5 tons of cyclone dust is recovered every day with a mercury content of 0.075%, the total mercury in the cyclone dust amounts to some 1,125 kg/day. With an average production of 7 flasks/day of mercury, the losses in mercury if the cyclone dust was taken to waste would amount to 0.46%. Even with higher mercury content in cyclone dust, the recovery of a small percentage of mercury in the cyclone dust is an advantage totally offset by the problems originated by the permanent recirculation of a substantial amount of dust into the system.

Such problems are:

- Clouding of the flame avoiding a proper visual control of calcines
- Difficult combustion of the fuel
- More frequent slag formation
- Creation of a toxic environment at the combustion end and at the discharge of calcines

Normal practice in mercury plants advices the disposal of cyclone dust to waste when its mercury content is equal to or below the grade of the ore.

In the case of Konya it is three times less. Therefore, we recommend to send cyclone dust to waste, although it is a product of difficult handling. The system shown in drawing II-2, where cyclone dust is mixed with water has given good results.

B.2.5. Handling of Stupps

Handling of stupps at the hoeing machine is giving poor results at Konya as the presence of bituminous shales in the ore and the unburnt residues of fuel-oil give place to olefines formation and to other organic substances which condense together with the mercury. These greasy substances hinder the very tiny droplets of mercury - (even treated with lime and mechanically hoed) to reach a sufficient degree of coalescence, lowering their specific weight and resulting in a poor efficiency of the hoeing machine.

The stupps collected at the hydraulic seal of the condensing pond are classified at Konya in two categories:

- Stupps with high mercury content
- Stupps with low mercury content

This classification is done by visual appreciation of the worker in charge of their distribution.

The cycle followed by the stupps is shown at the flow-sheet already discussed.

Rich stupps treated into the hoeing machine, have about 40% Hg. The efficiency of recovery is not greater than 60% in the hoeing process.

The poor stupps together with the used ones from the machine are fed back manually to the kiln.

The above system originates the following inconveniences:

- Bad efficiency at the hoeing machine
- Use of labor in a very toxic environment
- Unbalanced conditions and mercury losses in the condensing system

We recommend therefore to discontinue the recycling of stupps and to study the possibility of adopting the system used at the Spanish plant of Almaden described in Annex F.

Such possibility can only be evaluated after a test period at laboratory and at pilot plant scale, as Konya stupps may show a different response to the system than those of Almaden.

B. 3. Mechanical Problems

Problems of mechanical origin are found at the following sections of the plant:

- Output feeder of the run of mine ore bin
- Output feeder of the 1,000 t crushed ore bin
- Filling system of 50 t bins
- Vibrator feeder of calcines bin
- Conveying of calcines to dump
- Refractories life

B. 4. Recommendations on mechanical problems

We have observed that the problems of mechanical origin are often palliated at Konya by placing a worker at the points where difficulties arise. This solution must be avoided as rising costs in labor will have an ever growing influence on production results.

In the paragraphs below we analyse the main problems of mechanical origin and the recommended solutions to them.

B. 4. 1. Output feeder of the run of mine ore bin

This feeder, of the rocking tray type on rollers, closes the output opening of the bin. Its travel and tray are short, and therefore the closing of the bin is imperfect with the result of ore overflow. The presently adopted solution based in a hand operated gate, needs the presence of a worker per shift (4 men/day).

We recommend to adopt a suitable mechanical system, which with a small investment would allow to perform automatically the discharge of the bin without mannual help.

B. 4. 2. Output feeder of 1,000 t bin

This feeder is out of service for the same reasons as above. We recommend to adopt one of the many reliable existing mechanical feeders to correct the present situation.

B. 4. 3. Filling of 50 t bins

To avoid the overflow of ore from these bins, a man per shift is located at the discharge end of the conveyors to the bins.

The system could be made automatic, but we estimate that as a man must watch the crushing and conve

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ying installation, the automation of the 50 t. bin may be disregarded for the time being.

B 4. 4. <u>Vibrator output feeder of calcines bin</u>

The available height from the outlet of the calcines bin to the ground is very small, and therefore it is not possible to find a proper solution to the stoppages originated by the clogging of the bin outlet by the fall of slag pieces.

B. 4. 5. <u>Conveying of calcines</u>

The system used up to now, is very toxic for labor, needs a great amount of labor force and gives place to very frequent stoppages by wagon derailment and rail failures.

A new system is [illegible] ing skip hand-ling and wagon tipping [illegible] will solve the above inconveniences [illegible] recommend that the system should be completed by a water spray of the calcines to avoid dust [illegible]. This can be achieved by [illegible] of the bin and the skip track. The bin should be also provided with a cover and a stack taller than the highest level of the plant. Calcines should be thoroughly water sprayed while in the bins.

B. 4. 6. <u>Refractory Life</u>

As it can be seen in the study of kiln stoppages (see Annex G) the most frequent origin of those are refractory repairs and relining.

The amount of SiO_2 in the ore at Konya is modera te and in any case lower than that present in ores known by us and treated in rotary kilns with refractory life averaging two and half years.

Refractory life at Konya has never been longer than 14 months. The main causes for such a short life may be the poor quality of the refractory, bad setting of the bricks and abrupt temperature changes.

In Annex H, specifications for the recommended refractory brick are given.

B. 5. Process control problems

Process control is fundamental for an efficient plant operation, and all our recommendations can only be a guide to be followed in a systematic and patient way by the Konya experts to finally achieve operating results in accordance - with the frequent changes in the characteristics of the available ore.

We would like to emphasise that in our opinion, process control problems have not been given full consideration at Konya up to now. Present control instrumentation is inadequate and the very unhandy set up of the measuring points at the plant, involves great physical efforts for the operating personnel in charge of reading and recording, with the probable result of incomplete or wrong readings, and above all with the lack of taking proper and quick action to correct any deviation of the process.

This lack of attention and carelessness in corrective measures, have a very [illegible] effect on efficiency

Main problems [illegible] process control in Konya arise in the following areas:

- Feed ore weight control
- Feed ore sampling and analysis
- Calcine analysis
- Temperature control
- Pressure control
- Gas analysis

B.6. Recommendations on process [illegible]

B.6.1. Feed ore weight

The Konya plant [illegible] conveyor belt scale, but [illegible] not being used.

The scale is [illegible], and in our opinion it would be [illegible], but such location is [illegible] the present plant layout.

As feed ore weighing [illegible] properly performed to allow [illegible] efficiency, we can only insist [illegible] recommendations already provided to the Konya [illegible], in the [illegible] ed for the calibration [illegible] the scale by the qualified person [illegible] Ankara.

B.6.2. Feed ore sampling and [illegible]

Characteristics [illegible], where cinnabar is cure in form of rich concentrations inserted in totally - dead or waste zones, makes it very difficult to obtain a representative mercury grade of the ore, and carries - along with it a series of errors in sampling and analysis that may lead to abnormal results in efficiency calculations.

Sampling system at Konya may be qualified as generally correct but needs some improvements in order to minimise errors inherent to sampling process.

It is totally out of the scope of this report to - consider the theoretical grounds of sampling techniques, but nevertheless it is necessary to review in the most simplified possible manner some fundamental aspects of the source of sampling errors, to draw the practical conclusions to be applied to the case of Konya. This summary analysis is given in Annex I.

The sampler used at Konya plant has the following drawbacks:

a) The rectangular opening of the cutter is not wide enough.
b) The head of the ore fall is too high and bouncing of - large particles is common.
c) The system is submitted to frequent cloggings.

From the analysis of Annex I the following recommendations can be established:

1) The mechanical sampling device presently in operation at Konya must be modified according to the conditions given in Annex I.

2) Correlation must be established among mercury grade

of each sample and the corresponding amount of ore weighed though the automatic scale.

In that way, at the end of every day the following tabulation would be ready:

Weight (Kg)	% Hg	Kg x Hg %
P_1	G_1	K_1
P_2	G_2	K_2
.	.	.
.	.	.
P_n	G_n	K_n

$$\text{Average grade} = \frac{\Sigma K_n}{\Sigma P_n}$$

3) Number of samples should be figured following the rules of Annex I.

4) The assaying tempo of the laboratory must be sped up. Instead of 2 simultaneous assays, it is possible to perform 8 or 10, increasing only the number of burners, crucibles, gold plates and lids.

B.6.3. Temperature Control

This control was performed in the only kiln in operation (unit II) at three points of the circuit, using the following instruments:

- At kiln exit, (pyrometer and indicator on panel)
- At fan inlet (thermometer)
- At fan outlet (thermometer)

The pyrometer probe was damaged giving wrong indications, and the thermometers used at fan inlet and outlet had a too short stem, with the probable result of wrong readings. On the other hand, the location of the thermometers is very unhandy, making proper readings difficult and time consuming.

Therefore we recommend the installation of new - temperature measuring instruments, with centralised indication and recording on panel.

In Annex D , specifications of these instruments are provided.

D.6.4. Pressure Control

The pressure indicators on the gas circuit, show at every moment the operating anomalies in presence (clogging of ducts, abnormal air inlets, etc.).

The draft gauges used at Konya, although simple, are good enough for control purposes. They are centralised in a panel.

Nevertheless it is necessary to pay careful attention to their indications and to take quick action in case of anomalies, to correct them. The clogging for instance, in a condenser tube gives place to gas overloads in the remaining ones and to an increase in the speed of gases with the result of abnormal mercury and dust drags.

D.6.5. Gas Analysis and stack losses control

Assaying of furnace gases for CO_2 and O_2 determination had never been made at Konya. Therefore total gas volume, and the amount of excess air drawn through the furnace were unknown. Neither stack gas assaying for Hg determination is performed.

Abnormal excess air drawn to the furnace, occasions mercury losses by defective condensation, drag, and soot formation.

Too much excess air is frequent at Konya, due to uncontrolled air drawn though calcines bin, to bad combustion regulation and to emptying of kiln ore feeder. The location of the fan between cyclones and condensers makes the remaining part of the circuit work under pressure precluding air inlet into it.

We recommend therefore the installation of a recording instrument for O_2 determination at the kiln exit in order to know at every moment the amount of excess air in order to maintain the operation of the kiln within proper combustion conditions.

Specifications of O_2 analysers are given in Annex D.

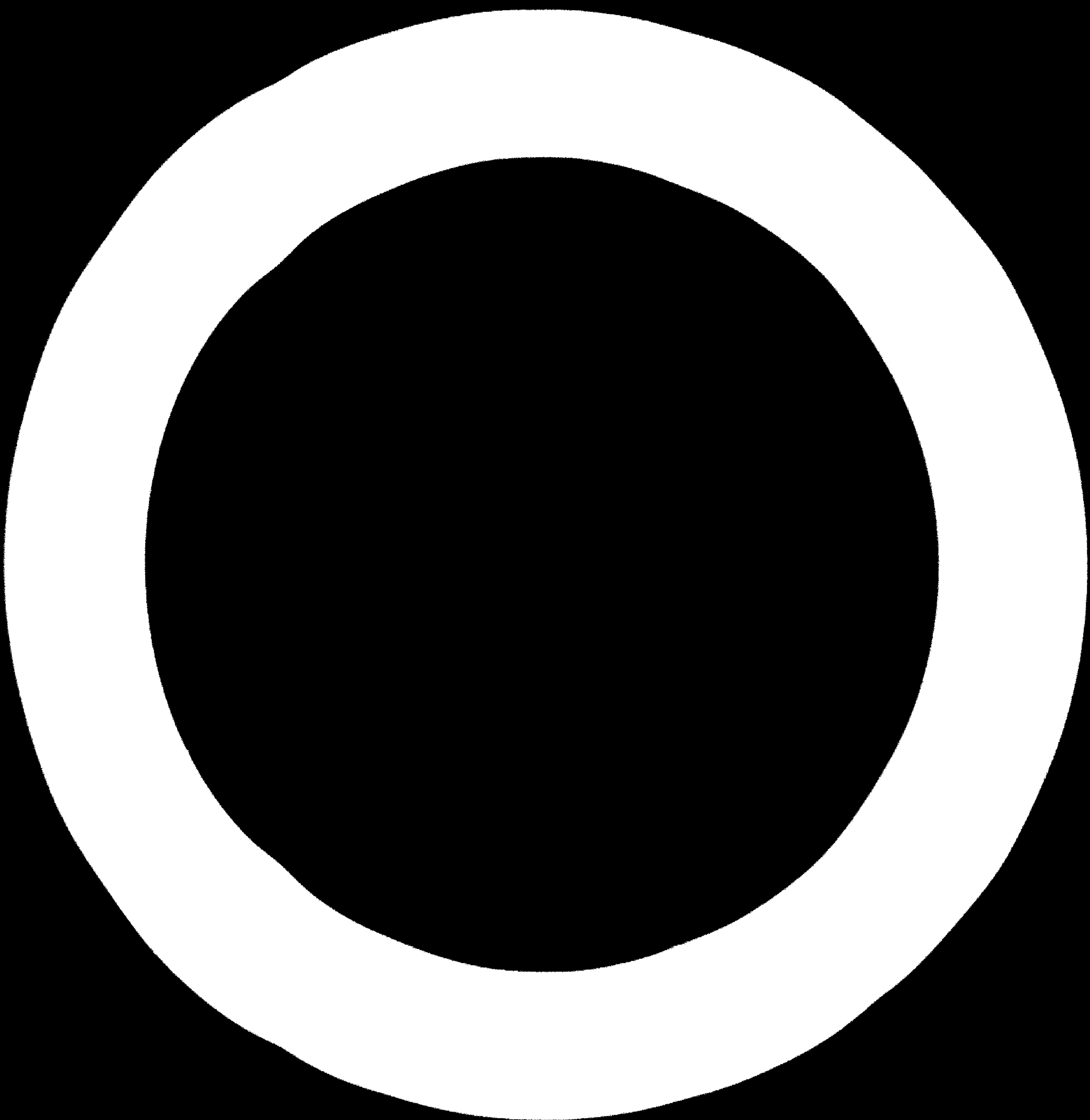
Stack losses through gas assaying for Hg determination are very difficult to assess, taking into account the difficulty of getting in the sampling duct similar aerodinamical conditions as those existing in the stack duct. Therefore at sampling only less dense mercury particles are recovered than those travelling though the stack duct, which results in inaccurate analysis.

The experience has proved in other plants that the mercury deposits in a given length of the stack duct are greater by far than the amounts corresponding to the alg ve routine analysis.

Even though it was not possible to measure mercury losses at scrubbers and stack dust, (such losses may be recovered by periodical cleaning, and therefore they are not losses in a proper sense) the scrubbers should be modified to allow their cleaning.

The construction of a parallel stack duct to the existing one, would allow the cleaning of one of the ducts while the second is in operation. This procedure would allow the assessment of stack losses with an small investment and the recovery of the mercury deposits.

The most efficient method to avoid stack losses although requiring a higher investment is the installation of a wet scrubber similar to those shown in Annex E, to reduce such losses to a minimum, contributing to extremely good efficiencies in total mercury recovery.



II.2. HALIKÖY

A. Mining

1. Problems

The mining operations of the Haliköy mercury orebody were visited during the First Mission to Project Area.

This orebody is well defined as it presents itself along the Bodyak fault and at the discordant contact of micaschists and gneiss. Given its importance, more attention should be paid in all phases of its mining operations.

The problems faced by the mining operations in the orebody derive fundamentally from the aging of its structures and from the lack of proper planning (due to an almost complete lack of investigation), and this has given place to a development of the underground workings almost simultaneous to the need of new exploitations.

As a consequence of this situation the known reserves of ore have been always insufficient, giving place to the performance of access workings in the most easy possible way, but disregarding their influence in a future mechanisation of the mine.

The accumulation of the above factors have originated effects of difficult solution in the present situation

It is necessary to emphasise that the exact definition of the mining problems at Haliköy needs several weeks of stay at the mine. Nevertheless, in a general manner the main problems to be solved could be listed as follows:

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a) General investigation program

b) Planning and performance of new access workings conditionned to the results of the investigation program.

c) Mechanisation of the outstope in galleries,

d) Transport's mechanisation

e) Work organisation

A.2. Recommendations

Lacking a deep knowledge of the orebody in run and depth it is difficult to establish recommendations to properly solve the aforementioned problems.

Nevertheless, our impression is that the Haliköy mine is "old", not for lack of reserves, but by structural defects. This opinion could be extended to the ensemble mine-metallurgical plant.

Therefore, and although the present situation of the mercury market may not encourage the starting of new projects, the Turkish experts available at the mining and metallurgical installations are fully qualified to develop a restructuration project, which should include the following phases:

a) General investigation program

The present investigation techniques of metallic orebodies allow the exact knowledge of the existing reserves, even in such cases as cinnabar orebodies, where the mineralisation is sparse and irregular.

The presente knowledge of the Haliköy orebody, and the new investigation studies recommended will give results that might decide the starting of a new mining and metallurgical unit at Haliköy.

This investigation program should include the following phases:

- Structural geology
- Geophysics
- Geochemistry
- Mechanical drilling.

The three first phases (structural Geology, Geophysics and Geochemistry) will be able to assess the mining potential of the area and their cost is much lower than mechanical drilling. The last phase of the investigation should not be performed until the results of the three first phases are known and analysed.

a. 1) Geology

Generally, the orebodies of the type of Halikôv where tectonics has played an important role, give place to the formation of a "puzzle" of mineralized masses, where it is necessary to know their original location. Structural and tectonics geological studies will clarify the existence of masses with mineralization possibilities, presently unknown.

a. 2) Geophysics

It is recommended the performance of geophysical investigations in such areas where the geological studies have determined the possibility of existence of mineralised masses and also in those one presently under exploitation in order to assess its development in direction and depth. Tests should be carried on with different systems in order to select the most convenient to the type of rocks and mineralization to be investigated.

a.3) Geochemistry

In order to collect as many data as possible to perform a successful drilliny campaign, geochemical methodes should be employed in order to delimit within an area of promising mineralisation, such zones of highest possibilities.

a.4) Drilling

The results of the above investigations should be confirmed by a drilling campaign with core extraction, as a final phase of the investigation program.

All geological, tectonic and geochemical studies can be performed by the existing technical personnel at Halikōy. The geophysical studies and the drilling campaign should be subcontracted to specialised Companies.

If the results of the investigation programs above outlined are positive they might induce ETIBANK to the development of a new mine. In case of negative results, costly and unnecessary investments would be avoided.

Finally, we would also point out that outside the present exploitation field of Halikōy, other mines are active, and this fact indicates the probable presence of an area with good possibilities of existence of workable cinnabar mineralisations. Therefore a preliminary study of the strike of the Bedyak fault seems advisable. Such study should be performed by pan prospection of the small water streams which cut the aforementioned fault. Our experience in similar mineral formations with such a simple method, shows that it is possible to delimit quickly and economically areas where detailed investigations should be performed.

b) New access workings

Present mine planning is done through access workings, consisting in butt entries to reach deeper zones. Nevertheless, such planning has the following inconveniences:

- A diminution of transport capacity
- An excess of labor force
- An excessive proportion of mine workings in waste in relation to the extracted ore
- An increase in the quantity of equipment (winches an pumps)
- A delay in the preparatory workings

The solution to be adapted in order to solve the above drawbacks, depends fundamentally on the results of the general investigation program outlined in the preceding paragraphs, which determine the decision to be taken on the different possible options about new access workings. To adapt a proper decision it is necessary to examine many technical and economical factors presently unknown until the proper investigations are performed.

We should emphasize that the recommended investigation program of the Habitby area must go before any decision about this problem. Also, the proper location of the new access workings to be projected must take into account the cost of the external transportation presently realized.

c) Mechanization of the outstopes in galleries

Although similar considerations to the above could be made about this problem, we estimate that the present layout of the mine allows a moderate mechanization of the different phases of the outstope in galleries.

The present dimensions of access workings allow the introduction into the mine of waste and ore loading machines of only small size. Nevertheless, for the present ore needs of the metallurgical plant we estimate that such problem could be properly solved with the purchasing and operation of small loader shovels (i.e. EIMCO 12-B or 6/8/RS) and pneumatic hammer-drills with pusher. Such equipment would greatly increase productivity in all opening and winning workings, as in this type of ore the percentage of workings in waste per ton of mined ore is very high.

d) Transport's mechanization

The solutions to this problem are very difficult given the present mine's layout. Both the mine's structure (working of small section, sharp curves, etc.) and the relatively low tonnage of ore to be mined, advise until a new project of the mine and metallurgical unit is not decided, and cost of labor is not excessive, to continue the present transport methods.

Nevertheless, we estimate as convenient to study the feasibility of installing small chain and budge conveyors in order to eliminate a great part of the present manual handling of mine cars. Such study can be perfectly performed by the existing mining technical personnel.

e) Work organization

In 1971, 60,9?1 tons of ore were treated at the mercury plant. Therefore, in the assumption of 300 working days, the comparative study with other Spanish mercury mines show the following results:

	Halikoy	Other mines
Ore tons/day	161	[illegible]
Labor force	[illegible]	1?0
Hours/man-day	6	7
Working days/year	300	300
Total hours/year	61?,?00	[illegible]
Kg. of ore/man-hour	76	1?0

The present development system of the mine will make heavier the present situation of excessive labor force.

We realize that an adequate organization of the mining operation with intensive mechanization would have a great impact on the labor force of Halikoy, as new and more productive methods would allow the reduction of the present number of workers. Nevertheless, the investigation and preparatory workings recommended in the above would absorb part of this labor force and would alleviate to some extent this important problem.

B. Metallurgical plant

The Haliköy metallurgical plant needs a total reconversion of its present production facilities in order to achieve completely satisfactory working results. This fact has been fully understood by Etibank's Management, who has studied several alternatives for the solution of the Haliköy situation including the construction of a totally new plant

Therefore, the recommendations provided in the following paragraphs on the problems existing at Haliköy should be adapted taking into account the estimated future situation of the mercury market and the final criteria to be adopted by Etibank's Management on the implementation of the expansion programme for Haliköy mine and mercury recovery plant.

As with Konya the problems of the Haliköy metallurgical plant have been clasified in three main groups: metallurgical, mechanical and process control problems.

D. 1. Metallurgical problems

In this group are included all those problems that give place to a defective utilization or to a low efficiency of the plant.

The fundamental problems arise from:

- Granulometry of fed ore
- Handling of stupps at the Exeli hoeing machine
- Uncontrollable air inlets into the system
- Defective design of the condensing system
- Defective design of settling tanks
- Defective fuel-oil feed to the burners

In general, as it has already been stated, the above problems are not of an independent nature and are interrelated in many aspects.

D. 2. Recommendations on metallurgical problems

D. 2. 1. Granulometry of feed ore

During our stay at the plant we recommended the performance of a granulometric analysis of the feed ore, never made before. Reported results show that 55.75% of the calcined ore is below 5 mm.

Such a high percentage of fines in the ore causes all the inconveniences that have been already discussed in the corresponding paragraphs for the Kenya plant.

In order not to increase unnecessarily the amount of fines already present in the run of mine ore, we recommend to crush only sizes above 80/85 mm. To this end, a screen should be located between the bottom of the run of mine ore bin and the crusher, conveying to this one, only sizes above 80/85 mm.

As for Konya, the size of 80/85 mm of the coarse ore is only indicative, and as our tests have shown, this size should be the starting point for extensive trials in order to find the optimum feed size ore following the indicated pattern already recommended at Konya, with a careful control of the affecting variables, i. e.:

a) Tonnage of calcined ore with different feed sizes.

b) Behaviour of calcines.

The above solution is a transitory one which would attenuate the excess of fines, with only slight changes in the presently existing crushing circuit, and which will, without any doubt increase present roasting capacity by 20% to 25% as our tests have shown. A systematic study of the correlation between granulometry and mercury grade distribution would also be extremely useful as a base for the proper design of the ore preparation system to be adopted in a future new mercury recovery plant at Haliköy.

B.2.2. Handling of stuppe to the hoeing machine

Present manual system of stuppe handling, from the condenser pond to the hoeing machine is very laborious, toxic and needs too much labor force. The proper solution of this problem would need the total modification of the present condenser system, placing the condensing pond at a higher level than the present one, allowing the feeding of the stuppe to the treating system by gravity.

The inconveniences of lime treatment of the stuppe at a hoeing machine and their feed back to the kilns, have been extensively discussed in Annex F of this report.

In the case of Haliköy, mercury losses in condensing originated by stuppe recirculation are even worse than in Konya, taking into account the defective characteristics of the condensers.

Therefore, and taking into account that the possible adoption of the Almadén-CENIM process for stuppe treatment should be contemplated more properly in the construction of a new plant, we recommend starting again the retort operation, even admitting its low thermal efficiency.

B.2.3. <u>Uncontrollable air inlets</u>

Air inlet into the system of impossible control takes place in the following sections of the system:

a) At the condenser pipes.

b) At the scrubbers

c) At the exhaust fan.

a) Readings of CO_2 content in gases with the Orsat analyzer of condensers have never been over 5%, while at the kilns' exit, recorded values have been systematically much higher (7 - 10%). These figures clearly show substantial air inlets into the condensers, through cracks and holes produced by corrosion.

Our experience in the roasting of ores with high sulphur content and in plants with similar corrosion problems as those existing at Haliköy advise the substitution of cast iron pipes as construction material of the condensers pipes by stainless steel.

AISI type 316, should be used at the intermediate section of the condensers (temperatures between 90º and 130º C) and AISI type 301 at the cold section (below 90º C). Cast

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iron pipes may be maintained at the hot section of the condenser (over 130º C).

Present life of condensing ducts at Halikoy is extremely low, especially at top and bottom connections where replacing must be done every 6 and even every 3 months.

Special care must be exerced at welding, and if cast iron is maintained at the hot section of the condensers, heat resisting plastic joints should be provided to avoid galvanic corrosion.

Costwise, and refering to Spanish conditions, 1 m of 301 stainless steel pipe, 2 mm gage has the same prices as the same length of cast iron pipe, 12 mm thick, for equivalent diameters.

b) To achieve a depression regulation in gas ducts at Halikoy, it is customary to leave partially open one of the top lids of the scrubbers. Such practice must be discontinued, as it affects condensing efficiency by dilution of mercury vapors and mechanical drag to the stack line. A regulating value should be placed at the exhaust fan inlet for depression regulation. Samples of deposits in stack line, our tests have shown mercury contents up to 57%. Such figures are an evidence of significant mercury losses through the stack.

c) Maintenance conditions of the exhaust fans are extremely unsatisfactory. Large openings in the fan's housing are perceptible, caused by corrosion of the steel plate. Here again, we recommend as construction material for the fan, stainless steel, AISI type 301.

B.2.4. Defective design of the condensing system

Temperatures at cold end of condensers are, as a rule, high, and were never below 31º C, even during the coldest month of the present year (January 1972).

Such high temperatures are produced by the inadequate design of condensing system.

Although there is a definite relation between area or length or volume contained in the condensing system, it has been many times illustrated that the relation is not in direct ratio to the tonnage being treated. This is probably, in part, due to variations in excess air required in the different sizes of mechanical furnaces, differences in rate of heat transfer caused by different gas velocities, or differences in composition of furnace gases.

As wide variations can exist owing to an extreme condition in one of the variable factors previously mentioned as affecting the design of a condensing system, and this matter is totally out of the scope of the present report, we have emphasized those factors that should be considered in the design of a new condensing system for Halikby, in connection with the construction of a new mercury recovery plant.

Nevertheless, a provisional solution to the problem would be the use of stainless steel as construction material for the condensing pipes, as this material would allow the water spraying of the condenser's surface, with the consequent lowering of the exit gases' temperatures and increase in condensing efficiency.

B.2.5. Defective design of settling tanks

Present design of settling tanks used for decanting waters from the washing of the condensing system is insufficient as mercury losses in overflow waste waters are significant, as our tests have shown.

Present dimensions and design of settling tanks be near increased in order that mercury lost, either mechanically or by solution in overflow water, should be near negligible. All overflow water should be collected in launders and run to settling boxes arranged with baffles to cause the current to underflow.

This will collect most of the mechanically transported metal, the metal in solution could not be economically recovered.

Nevertheless, the location of the present settling tanks, below ground level makes its cleaning very difficult, and requires excess labor force.

New design of settling tanks should foresee their location above ground level in order to expedite cleaning operations and deposit recovery.

B.2.6. Defective fuel-oil feed to the burners

Orsat readings at the exit of the kilns, show that burner of kiln I, does not work properly, as maximum values of CO_2 in exit gases are never above 7%. Burner of kiln II, has worked more satisfactorily with CO_2 values about 10%.

The bad regulation fuel/air in the burners causes a defective combustion process with the consequent soot formation and excess air in combustion gases.

Presently existing burners are obsolete and their fuel-oil supply is made by gravity which makes very difficult a proper combustion regulation.

Since 1970, a complete set of burner, gear pumps, air fans compressors and compressed air tank is available at Halikoy's storeroom. We strongly recommend the installation of such equipment in Kiln I, as soon as possible, in order to test its capacity and operating characteristics, in order to improve, as much as possible, the present combustion process, whose deficiencies derive more from the obsolence of the existing equipment, than from lack of attention by the personnel in charge of the kilns.

B.3. <u>Mechanical problems</u>

Problems of mechanical origin are found at the following sections of the plant:

- Shaker ore feeders to the kilns
- Calcines' removal
- Refractory lining
- Exhaust fan

B.4. <u>Recommendations on mechanical problems</u>

B.4.1. <u>Shaker ore feeders</u>

The available records for 1971 and 1972 and our observations during the stay at the plant, show that frequent stoppages arise at the kilns' ore feeders.

Reported cause for such failures is the inadequate steel quality of the mechanical gearing system.

On the other hand, the only possible way to regulate the incoming ore flow in the present system is to change the springs' tension of the feeders.

A simpler and more efficient feeder's system is that which incorporates a speed variator (manual or automatic) allowing the ore flow adjustment according to the size and characteristics of the incoming ore. They are rugged, inexpensive apparatus and subject to low maintenance costs.

B. 4. 2. <u>Calcine's removal</u>

Calcine's removal from the bins with the present system used to convey the calcines to the refuse dump - may affect the capacity of the kilns if not enough labor is available for the handling of the wagons.

On the other hand, this work is very toxic, as during the emptying operations of the calcines' bins and the handling and tipping of the wagons, the workers are subject to sulfidic gases and dust.

The operations should be mechanised, adopting a similar design to that used at Konya plant for calcine's disposal.

B. 4. 3. <u>Refractory lining</u>

73% of the stoppages' time during 1971, equivalent to more than 72 days of production losses, i.e. some - 670 flasks of mercury have been originated by the necessity of too frequent refractory relining.

The above figures depict clearly the importance of this problem at Haliköy.

We recommend testing the quality of refractory specified in Annex H of this report. Such refractory has given excellent results with ores of much more abrasive nature than those treated at Haliköy.

B.4.4. Exhaust fan

This problem has already been dealt with in above paragraphs from a metallurgical point of view. We would only note here, that stoppages by fan breakdowns were second in importance at Haliköy with 173 hours of lost production in 1971.

The recommendations already given about construction materials of the fan, and the installation of a water spraying system at the internal walls of the fan in order to wash out dust deposits would solve satisfactorily the present problems found at this section of the plant.

B.5. Process control problems

The analysis of control problems and the recommendations provided to improve the present situation in the paragraphs corresponding to the Konya plant are applicable for to the Haliköy plant. Nevertheless, we would emphasize that for the Haliköy case, feed ore control is even more important, as the characteristics of the ore roasted are more variable with respect to mercury grade than in the case of Konya, as can be easily understood from the figures and calculations of Annex I.

To achieve a proper metallurgical control it is necessary to assign to each analysis, the corresponding weight of ore, as it already has been explained for Konya. To this end the automatic sampler must work in coordination with the automatic scale, and should be located at the discharge end of the belt conveyor from the crusher. This location would also allow the installation of a similar sampling station to that existing at Kenya (small crusher and quartering).

The present system of samples preparation for analysis is manual and time consuming. Laboratory analysis procedures are slow and inaccurate as the available facilities and staff are not adequate. Especially, the balance is very old and is subject to strong vibrations by the hammering of the close by ore feeders to the kilns. These circumstances call in question the weighing accuracy.

Until proper values for ore grades and reliable weights for feed ore are attained, all calculated efficiencies for mercury recovery will be totally meaningless.

3 PRIORITY CRITERIA ON PROCESS CONTROL RECOMMENDATIONS

In sections II 1 and II 2 the present problems of Konya and Halıköy operations have been studied giving proper recommendations to their improvement and solution where possible

In our opinion, process control problems are of paramount importance and deserve close attention to allow better mercury recovery efficiency Therefore, in order to make our recommendations more useful, priority criteria of each of these process control problems are given below

Such criteria have been established from two different points of vue a) measurement systems allowing a proper materials balance and the assessment of representative mercury recovery efficiencies, and b) preferential locations points of other needed measures, such as temperature, gases, etc

a) Measuring systems

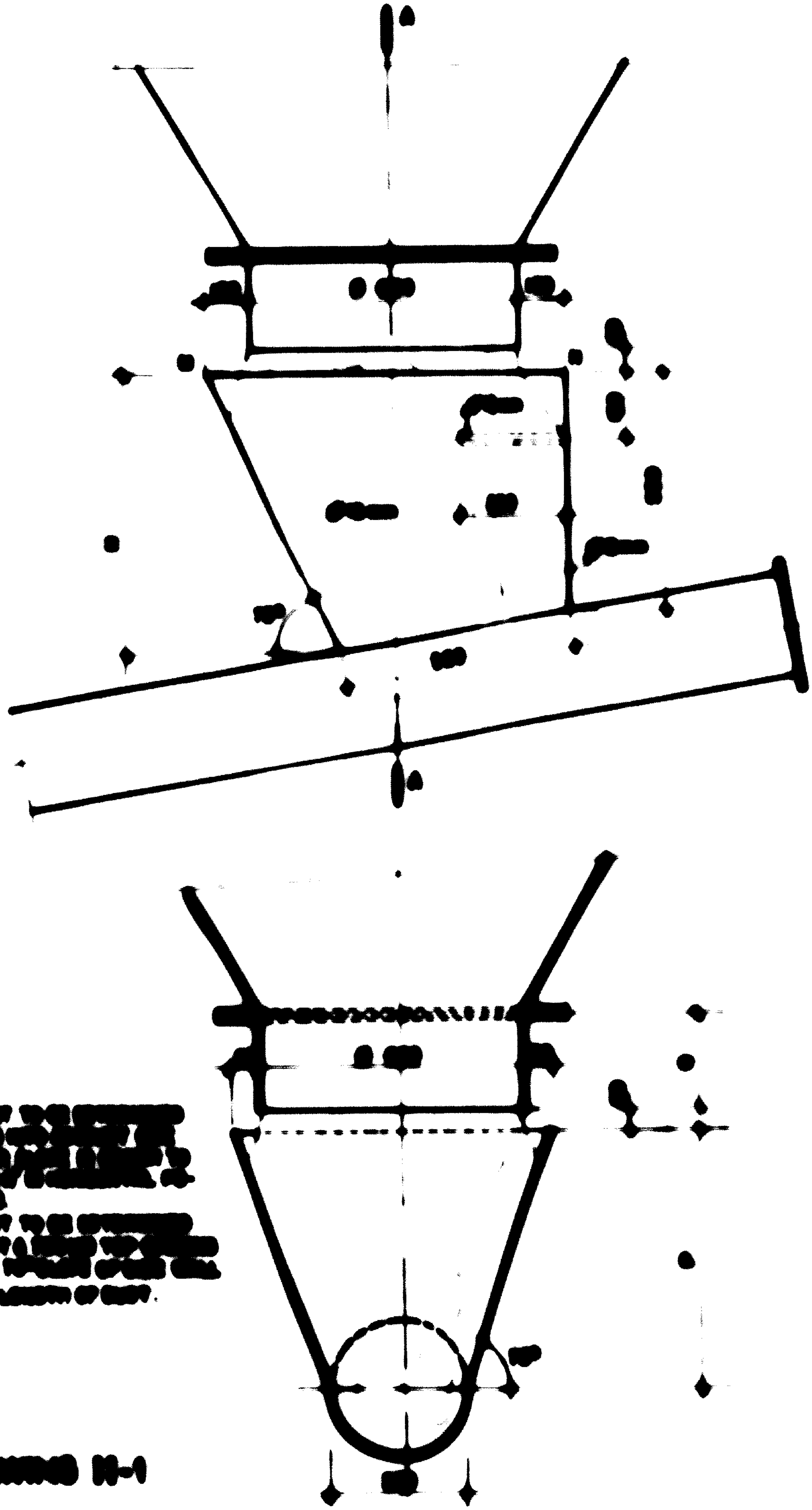
Indication of priority:

1st: Weight control of feed ore
2nd: Proper sampling techniques
3rd: Sufficient number of laboratory analysis of mercury grade of the ore
4th: Control of mercury losses (waste water, stupps, cyclone dust, etc)
5th: Proper calculation of recovered mercury
6th: Stack losses

b) Location of control points

Indications of priority is given in the following table:

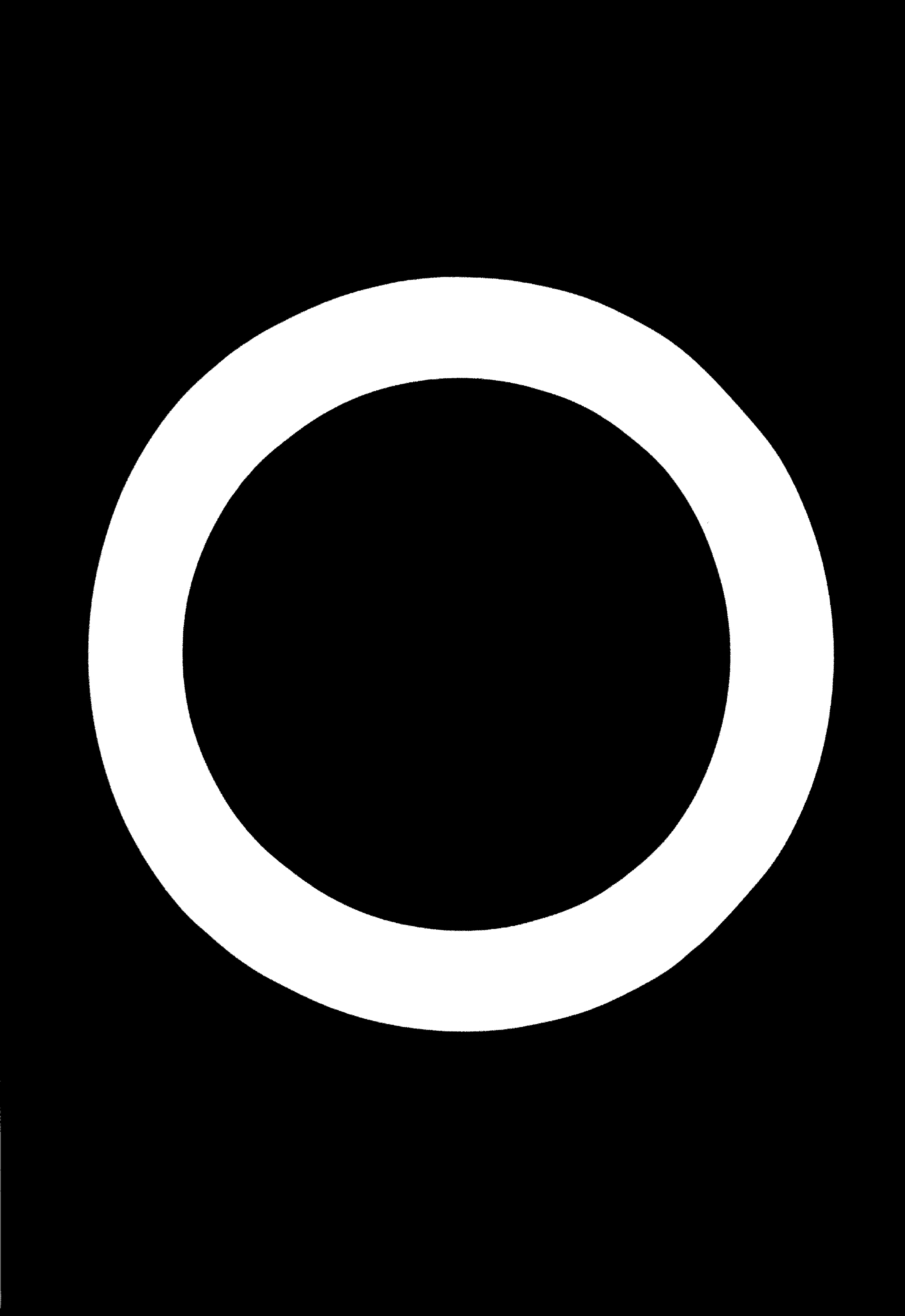
	Priority of location		
Sensors	1st	2nd	3rd
Temperatures	Kilns' exit	Condensers' exit	Fan's exit
Pressures	Same	Same	Same
CO_2 and O_2 gases	Same	Same	Same

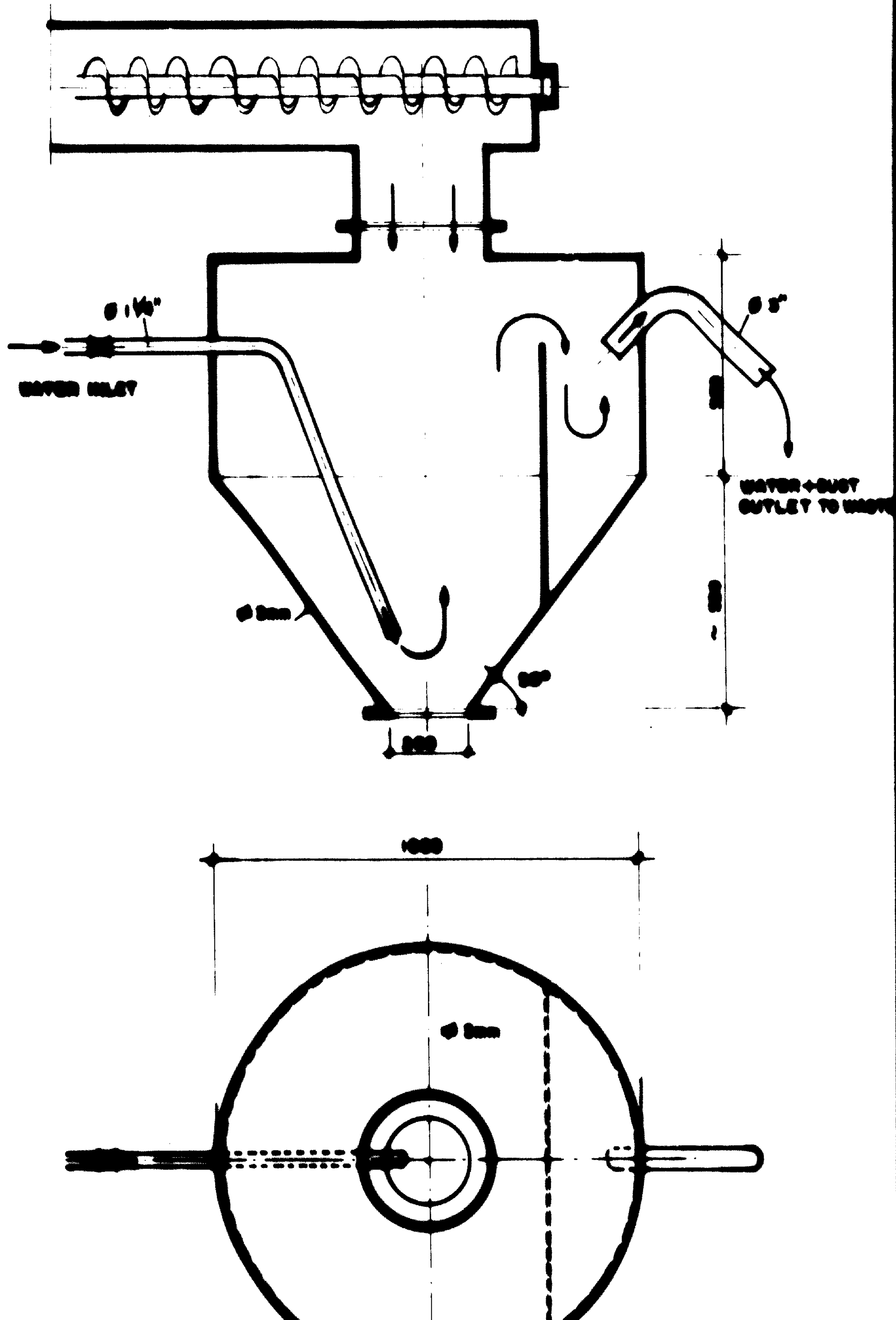


SECTION A-A

DRAWING H-1

I. INTRODUCTION





DRAWING II-2

III. DESCRIPTION OF PRESENT INSTALLATIONS AND OPERATING PROCESS

III. 1. **KONYA**

A. Collection of basic available data

The collection of basic available data on the technical aspects of the existing mining and smelting facili ties of Konya was started during the first mission visit to Konya area and was completed during the stay of the second mission to Project area and with the partial filling of the questionnaires by the counterpart experts. The information included in such questionnaires together with other data ga thered during the stay of the team in the Konya metallurgi cal plant are given below.

B. Mining

B. 1. Orebodies

The metallurgical plant of Konya receives mer- cury ores from two different orebodies, Sizma and Ladik. In each one of the orebodies several mines are worked, so me of them by open pit methods.

The ore from Sizma district has a higher mercu ry grade than that one from Ladik district, but taking into account that the mining concessions are not of the property of ETIBANK, royalties are to be paid according to ore pro duction and therefore such State body programs the mining operations in each district in order to get the most profi table conditions.

The mines are worked continuously, but the me tallurgical plant, processes separately the ores coming from each one of the two districts, in campaigns during several months for each type of ore. Therefore non processed ores are stored at the plant premises until they are reclaimed for metallurgical operation.

The program for 1972, taking into account the mercury production established for this year, foresees the processing of 80% of Sizma ore and 20% of Ladik ore during the year, with the following average mercury grades:

- Sizma: 0.290% Hg
- Ladik: 0.200% Hg

The programmed production for 1972 are 155 metric tons of mercury.

Sizma and Ladik ores are totally different as regards the nature of their mineralization characteristics. Sizma has a siliceous limestone gangue where mercury mineralizations appear very irregularly as cinnabar, while Ladik ore has a gangue of limestone with intrusions of bituminous shales and cinnabar appears in the contact zones of both rock formations.

B.2. <u>Data card and map</u>

In the following data card all basic available data on the Sizma and Ladik orebodies are put together. The corresponding map shows its location in Turkey.

MERCURY EXTRACTION (TURKEY)

DATA CARD

Name of mine - Çallca, Büyük Uçak, Medrese, Acikisla (Sizma district)
Cinarkan, Toprakli (Ladik district)

Owner

Data not available

Geographical Location

Etibank Konya Mercury plant is in the South of middle Anatolia, about 60 Km North-West of Konya and inside the bounderies of the town of Konya. The area is about 110 km² and consists of 4 main mercury deposits and other locations which are being explored. One part of the field is in the South near Sizma village and is called Sizma district; the other part in the North near Ladik village is called Ladik district.

Coordinates

X = n.a
Y = n.a
Z = 1750 m

Bibliography and References Up to date, no detailed research which covers the geological and mining aspects of the whole area has been made. The most important papers published by Etibank, M.T.A. and other organizations are as follows:

- KAADEN, G.V.d. (1964) Notes on the cinnabar occurences of the Mercur'lu-Ladik area. Vil Konya M.T.A. Ref. nº 3505 ANKARA
- SCHURNACHER, F. (1937) Bericht über das Quecksilber vorkommen Sizma M.T.A. Ref. nº 563 ANKARA
- SAMPLES, F.F. Mercury Mines at Konya, Asia Minor. Ref. nº 542 ANKARA
- WIESNER, K-LEMBERTC (1956) Sizma-Ladik Mercury Mines M.T.A Ref. nº 3729 ANKARA
- HALL R.B (1965) A field test of two methods of geochemical analysis for Mercury in soil U.S. AID, ANKARA
- BARNES, L-BROCKOWSKI, M. BAILEY, F. (1969) Geology and ore deposits of Sizma-mercury district. [illegible] Program in Geological Mapping Techniques Report
- [illegible], A-EREN, R. (1979) On the open pit mining over Büyük Uçak of Sizma district. Etibank Konya Civa Isletmesi ANKARA
- [illegible], I. (1980) Geochemical research on the mercury deposits at Konya Mercury Plant

Geological Description

The district consists of sequences of folded paleozoic in the E-W direction, old sericite-chlorite-biotite schists and limestone. The limestone is dolomitic in various places. This structure rises towards the South. Towards the North in the plains it has been hidden by conglomerate and alluvium covers. All the paleozoic rocks in the area had been subject to metamorphic changes. In the S-SE of the district, at Karatoro and Karadag, there are metamorphism of the green coloured basic extrusions. These series at low places are covered by a Quaternary alluvial conglomerate with residual regolith and alluvium.

Ore Body Geomorphological Description

The ore body formation is connected with the Quaternary volcanic action and the mineral deposits are epigenetic. The ore body contains, in an undifferentiated manner and in different grades, cinnabar, metacinnabar, antimony, pyrite, calcite, quartz, fluorite. Ore body starts from the limestone-schist contact and from there on moves down into the limestone. The worked ore in the district is cinnabar. The cinnabar ore bodies are small and discontinuous. In shape they show much variety, being tabular, pipelike or podlike. In the ore bodies cinnabar is usually erratically distributed. Some ore bodies are formed by disseminated aggregates of cinnabar locally accumulated in recemented limestone. Other prominent ore bodies consist of veinlets of cinnabar along fracture zones accompanied by cinnabar disseminated in the adjacent limestone wall rock.

Topographical and Climatical Description

The district has a hilly appearance in the E-W direction, which lies from N. to S. The mercury plant is about 1250 m high and the mines lie at an elevation of from 1300 m to 1750 m. The highest point in the district is Mahayani which is about 2161 m high. General drainage is from S. to N.

The climate of the region is continental, semi-arid and steppe like, with cold winters and hot summers. Precipitation is meager and mainly occurs during winter as snow and in early spring as rain. Vegetation is generally sparse, its local character and distribution being influenced by both rocks and topography.

INVESTIGATIONS

Chemical Analyses

Variable from mine to mine

The average analysis are for Hg (%)

- Sizma district: 0,313 (1970); 0,256 (1971)
- Ladik district: 0,241 (1970); 0,187 (1971)

Geological Studies

Regional geologic mapping on a scale of 1/25.000 has been made. Also geologic mapping of the mine district on a scale of 1/10.000 has been drawn, with the underground geologic mapping of Cirakman, Büyük and Calica mines, on a scale of 1/500.

Geophysical Survey

No geophysical survey of the district has been made. Only a resistivity research of a karstic cave in Calica mine has been carried out

Geochemical Survey

Geochemical surveys of the schists-limestone contacts between the regions Topcu Kayagi-Beypinari and Beypinari-Dumlu Tepe and around Calica Mine have been made. Generally ore body formations have been found at regions of anomalies.

Investigation of Bore Holes Pits and Galleries

At Toprakli Open-Pit, to determine the thickness of overburden and the ore body formation, 16 bore holes of about 800 m from surface have been opened and the distribution of the mercury deposits has been fixed. Investigations underground of bore holes have started in 1971 and up to date total length of bore holes drilled is about 1200 m. Especially at Calica region, more than 60 pits have been opened. In entire region, total lenght of the galleries is about 9,000 m.

Pilot Plant

No pilot plant had been established in the region before construction of the mercury plant.

Investigations and Studies Proposed

Out of the scope of this report.

MINERAL RESOURCES

Resources description

Data not available

Verified Tonnage

Probable Tonnage

Possible Tonnage

TECHNICAL TABLE

Description of Explotation Methods

20% of all ore produced is taken from open pit mining, the rest is produced underground. The method of production at open pits is called "grading method". As to underground mining, the production method changes according to type of formation of the mercury deposits.

These are shrinkage, caving and room pillar methods.

The grade of produced ore should be 0,2 - 0,3 % Hg. In some cases, the ore is hand - picked to improve the grade.

Mining Extraction Tonnage

	1969	1970	1971
Tonnage	24,567	[illegible]	[illegible]

Concentration or Process Method Description

The ore is not submitted to any concentration methods.

Tonnage, Concentrated or Processed

The following tonnage of ore have been processed at the metallurgical plan

	1969	1970	(Hg %)	1971	(Hg %)
- Sizma	-	[illegible]	[illegible]	[illegible]	[illegible]
- Ladik	-	7,[illegible]	[illegible]	[illegible]	0,[illegible]
	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]

Description of Transport Methods to Metallurgical Factory or Sales Destination

Sizma mines are in S. of and about 10 km from the Metallurgical Factory. [illegible] mines are in the [illegible] of and about 4 km from the factory. To both mines, there is a stabilized road which closes under heavy winter conditions only. The ore is carried to the factory by dumper trucks of 10-15 tons capacity.

MISCELLANEOUS

Personnel (Technicians and Workers)

It consists of director, technical vice-director, 5 mining, 3 chemical, 1 metallurgical, 1 mechanical engineers, 1 geologist, 4 technicians, 36 civil servants and 700 workers (mine and metallurgical plant).

Machinery

2 Excavators	14 Compressors
4 Bulldozers	6 Cranes (Electrical)
5 Loaders	2 Cranes (Hydraulic)
2 Wagon-Drills	5 Euclid Trucks (20 ton-capacity)
3 Underground boring machine	12 Trucks (8-12 ton-capacity)

Water Supply

The water needed in the plant is brought from a water source called Beypinari about 3 Km from the plant.

The mines are supplied by a near by water source.

Power Supply

Power is supplied by the State electricity network. In the event that this power is cut off, a 247 kw generating set is always ready to operate.

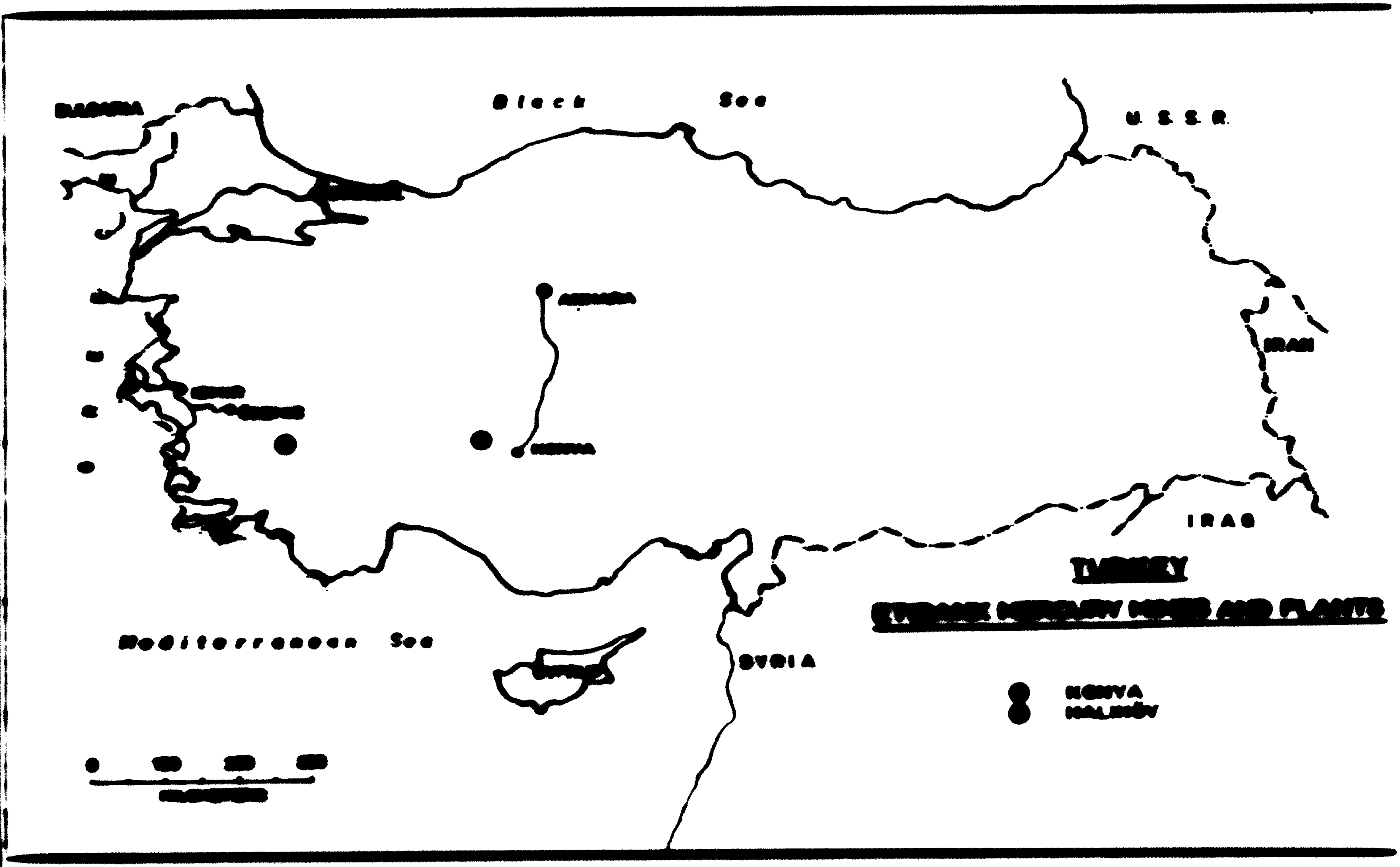
Other Supplies (Fuel, oil, explosives, wood)

- Explosives consumption: 0,807 kg/t of ore (1970); 0,985 kg/t of ore (1971)
- Compressed air consumption: 94,5 Nm^3/t of ore (1970); 86,8 Nm^3/t of ore (1971).

REMARKS

Only the Büyük Ocak mine has been visited

Client	Consultant firm	
U N I D O	TECNOGERMA	



Black Sea
U.S.S.R.
IRAN
IRAQ
SYRIA
Mediterranean Sea
TURKEY
KONYA

TEAM COMPOSITION

This report has been realized by a Tecniberia's team of experts, integrated by the following members (listed in alphabetical order):

- Lozano, Fernando	- Dr. Mining Engineer
- Mallol, Alberto	- Dr. Mining and Metallurgical Engineer
- Núñez, Adolfo	- Dr. Mining Engineer
- Núñez Javier	- B.S. Metallurgy

C. Metallurgical Plant

C. 1. Location

The Konya mercury plant is located in between the two mining districts of Sizma and Ladik, and its distances to the mining areas are 10 Km. and 3,5 Km. respectively.

C. 2. Plant Description

Flow sheet of ore preparation

The present flow-sheet of ore preparation installations up to ore storage bin is shown in drawing III. 1 enclosed.

Incoming ore loaded trucks dump on a small bin, covered by a fixed grid (1) with 300 mm openings. Under the bin a vibrating feeder (2) takes the ore to the belt conveyor (3) feeding a first jaw crusher (4) with the following operating characteristics:

- Capacity: 35 t/h
- Feed size: up to 300 mm
- Product size: 120 mm

The ore from crusher (4) is classified at the 40 mm screen (5). Coarse sizes (40-120 mm) are crushed in a second jaw crusher (6) giving ore sizes up to 40 mm. Conveyor belts (7) and (7') take the ore to an storage bin (7'') with a capacity of 1000 t of crushed ore.

Flow sheet of bin feeding system

From the 1000 t storage bin the ore is conv yed through a feeder (8) and a conveyor belt (9) to a 50 t surge bin (11). The conveyor belt (9) is equipped with an automatic scale (10), out of operation, and an automatic sampling device (37). (See drawing III. 1).

<u>Flow sheet of ore roasting, condensing and mercury recovery system</u>

(See drawing III. 1)

<u>Operating Data</u>

The plant works 24 hours a day in 4 shifts. The amount of ore roasted in each kiln per shift is 30-35 tons. The kiln entrance temperature is between 270-300º C, and ore is roasted up to 950-1100º C.

The kiln makes one revolution in 45 seconds although this speed may be changed by speed reducer and ore stays in the kiln for about 90 min. The burnt ore or "calcines" contains, on average 0.004% Hg

The kiln is heated with a burner which atomises fuel-oil by compressed air. The exhaust gases are suched by a 300 Nm^3/minute capacity fan, located into the system after the cyclones, and blown to the condensing system. The condensation pipes are washed with the water taken from the condensing pond every two days at the 12-18 hours shift. The pond is cleaned the next morning till 10 hours. The stupps are taken to the hoeing machine, in amounts of 100 to 200 kg. at every charge of the machine. The used stupps are sampled and given back to the kiln again.

The condensing pond is cleaned from muds and these muds, after sampling, are also fed to the kiln. However, the amounts are unknown.

From time to time the condensing ponds are cleaned and the recovered stupps are fed back to the kiln.

The average mercury production is about 7 flasks per day and the efficiency of the mercury recovery is below 85%. Mercury production in 1969, 1970 and 1971 was 47,400; 89,447 and 122,514 tons respectively.

Equipment Data

The plant has two roasting rotary kilns, whose operation started on March 1969 and December 1970 respectively. Both kilns have the same operating characteristics.

The project and engineering of the plant was performed by Del Monego Co., Italian engineering and construction firm.

The following description of equipment applies to each one of the kilns, unless otherwise stated.

Rotary Kiln Characteristics

Length:	37 m
Diameter:	2 m
Slope:	5 %

Driven by 20 CV motor and speed reducer unit.

Cyclones Type and Efficiency

The cyclones are of Sirocco type.

Their diameter is 500 mm. and there are 4 cyclones. Their efficiency is unknown.

Condensing System dimensions, and materials used in its construction

The condensing system is in 4 rows of pipes and consists of 3 pipes put on each other. The pipe length is 400 cm. and its diameter is 40 cm. In every row there are 65 pipes, 11 top joints and 11 bottom joints. The material used for its construction is cast iron.

Exhaust fan characteristics

- Speed of gas flow: Not known
- Capacity: 300 m^3/minute
- Depression: 200 mm/H_2O

Settling tank dimensions

5.90 m x 32.90 x 2,70. Its capacity is 122 m^3 of water.

Characteristics and dimensions of stack line and stack

Stack line: 2.5 x 2 x 160 m made of concrete

Stack: 1.90 x 15 m. made of concrete

Characteristics of hoeing machine (one for both kilns)

- Diameter: 1 m.
- Capacity: 150-200 Kgr. Stuppe in one charge. It has a water jacket around it with a temperature of 70° C. The arms turn at a speed of 15 r.p.m. The average time of treatment is about 1 to 1.5 hours.

Operating parameters

a) Operating efficiency of rotary kiln = $\frac{\text{Annual Working Hours}}{24 \times 365}$

1970 1. Unit, $\frac{6212}{8760}$ = 0,709

1971 I. Unit, $\frac{7305}{8760}$ = 0.834

II. Unit, $\frac{5385}{8760}$ = 0.615 Not in operation for 3 months

b) Capacity Efficiency = $\frac{\text{Actual Capacity (tons)}}{\text{Teoretical Capacity (tons)}}$

1970 I. Unit - Average = $\frac{132.5}{150}$ = 0.883

1971 I. Unit - Average = $\frac{114}{150}$ = 0.760

II. Unit - Average = $\frac{113}{150}$ = 0.753

c) Metallurgical Efficiency = $\frac{\text{Hg (Obtained)}}{\text{Hg (Content)}}$

1970 I. Unit = 81.85 %

1971 I. Unit = 81.89 %

1971 II. Unit = 84.86 %

Other Operating Data

Kiln exit temperature (estimated)	900° - 1100 °C
Kiln entrance temperature	270° - 300° C
Fan entrance temperature	270° - 290° C
Kiln exit vacuum	~ 0.8 cm. w.g.
Kiln entrance vacuum	~ 0.4 cm. w.g.
Cyclone depression	~ 6.0 cm. w.g.
Condensing system pressures	~ 2.2 cm. w.g.

Hg content in cyclone dust and tons/day of dust

Hg content in cyclone dust	0.079% Hg
Amount of dust (estimated):	1.5 ton/day (tons of treated ore x 0.012)

Gas flow (Nm^3/hour) in relation to weight of treated ore : Unknown.

Hg in gas (grs/Nm^3) at stack inlet-Unknown.

Specific consumptions

Fuel oil consumption per ton of treated ore

1969:	39 Kg/ton
1970:	41 Kg/ton
1971:	35 Kg/ton

Power consumption per ton of treated ore.

1970	17 kwh/ton
1971	14.4 kwh ton

C.3 Present operating conditions

C.3.1 Characteristics of treated ore

C.3.1.1. Mineralogical components

The only available data on the ores are given below and correspond to average characteristics for 1971.

Other reported analysis for 1969 and 1970 are incomplete and therefore have been omitted from the present description.

	%		%
Hg	0.24	Zn	0.2
Fe	2.60	H_2O	4.0
SiO_2	42.0	Al_2O_3	1.24
CaO	11.0	Pb	0.3
MgO	3.5	Co_3Ca	34.52
Sb	0.4		

The above analysis is anomalous, as normally carbonate figures are given as CO_2 and oxides. As no percentage of CO_2 is given, it is estimated that no other carbonates are present and that the remaining oxides are a part of the alumina silicates.

Metals affine to mercury and iron have been assumed as forming sulphides. In the analysis, the percentage of S, should be given. Nevertheless, as this content is small, the error introduced by the assumption that all metals are in the form of sulphides is acceptable.

Therefore, according to the above assumptions the probable mineralogical analysis is:

SHg	0.2704%
S_2Fe	5.9271%
S Sb	0.[illegible]%
S Zn	0.1%
S Pb	0.[illegible]%

- Total sulphides ~ 7 %
- Total S ~ [illegible]%

Alumina silicates and humidity:

$(SiO_2)(Al_2O_3)_a(MgO)_b(CaO)_c(H_2O) = 61.74\%$

The above assumed mineralogical analysis has been confirmed by our observations at the plant, as estibine, blende and galena have been repeatedly observed in the ore.

Arsenic bearing minerals have not been traced. The nature of the gangue is of a carbonated nature partially silicified. Pirites are also present with cinnabar due to the contact with limestones and silicifications.

The proportion of SiO_2 in the gangue is not [illegible] enough to produce [illegible] effects on [illegible] metal parts (liners, feeders, etc.) [illegible] the [illegible]

The amount of silica shown in the [illegible] low the existing one in other cinnabar ores, and on the other hand, the high percentage of fines in presence will absorb the impacts or friction originated by coarse ore. Therefore, the abrasive nature of the gangue might be estimated as mild.

C.3.1.2 Granulometry

The run of mine ore is received at the plant in different sizes and upon the reception bin, a 340 mm grid is located. The ore with higher sizes is hand broken to allow it to pass through the grid, whose openings are fixed in accordance with the intake size of the primary crusher.

The amount of fines in the Diamo ore is very substantial. These fines produce difficulties in the metallurgical process.

No granulometric analysis of the ore was available and the laboratory has no facilities for its performance.

C.3.1.3. Humidity

Surface wetness amounts to 3-4%. This figure is not excessive for such types of ores as those of Stams and Leith, and might be considered as normal as regards other plants. At rainy seasons this amount grows substantially and in 1971 several stoppages were originated by this reason, as the ores became sticky or frozen in some points of the circuit. Nevertheless, this humidity content does not justify any special installation to reduce its percentage.

C.3.2. Process Control

C.3.2.1. Treated ore and calcines

The information gathered during our stay in the plant concerns the month of March and first part of April 1972, and are given in Tables III-2 and III-3.

During the above months, only one of the kilns (unit II) was in operation.

Collected samples are sent to the laboratory at the end of every shift which is six hours. Only one analysis is performed at the end of the shift, which amounts to four analysis/day for ore and calcines respectively.

C.3.2.2. Stack Gases

No control is performed on the stack gases. Therefore, mercury losses though the stack are unknown, but for efficiency calculations it is assumed that 1 kg. of mercury is lost for every 80 t. of calcines. The reason for such assumption has not been specified by the technical personnel of the plant.

C.3.2.3. <u>Cyclone Dust</u>

A sample is taken from the collected dust at the cyclones at every shift, for mercury analysis and average of the four shifts is given. (See Table III-2) Total amount of cyclone dust is not exactly known but as an estimated figure, the amount of cyclone dust is calculated as 1.8% of the treated ore tonnage. This calculation is based, apparently, on past experience when sporadical weighing of discarded cyclone dust was performed.

The average mercury content of cyclone dust has been during the month of March and first part of April substantially higher than the reported average of 0.07% for 1971.

The corresponding tonnage during the same period of time should be near 1.5 t/day, if the afore-mentioned calculation system is applied.

C.3.2.4. <u>Stupps</u>

Treated stupps at the hoeing machine are given back to the kilns. Reported mercury amounts in these stupps range up to 20% Hg. The amounts fed back to the kiln are not exactly measured, but some 15 to 20 wheelbarrow loads are taken every two days, half of their content being $Ca(OH)_2$

C.3.2.5. <u>Waste Waters</u>

Overflow water amounts are negligible in the operating system at Kenya. Therefore, it is estimated that mercury losses by that concept should be minimal.

C. 3.2.6. <u>Kiln Gases</u>

CO_2 and O_2 analysis of kiln exit gases had never been performed up to now. Therefore, excess air in combustion, duct velocities of the gases, etc., are unknown.

Fuel and air inputs were left at the judgment of the kiln operator.

C. 3.2.7. <u>Temperatures and pressures</u>

Information about these two factors of the process is included for the month of March in Table III-4. Apparently little or no importance was paid to these control figures. The very unhandy set up of reading points at the Kenya plant, gives place in our opinion, to unreliable readings.

As a matter of fact, the pyrometer placed at the exit of the kiln gave systematically wrong readings, but nevertheless, such figures were recorded on the shift control forms.

Our impression is that the presently available instruments for temperature and pressure control are far from being adequate, and if they are not greatly improved this aspect of the operation control will be very difficultly performed.

C. 3.2.8. <u>Laboratory Analysis</u>

The Kenya laboratory reports properly to the plant the different aspects of the process which are under its control.

Nevertheless, and even endowed with a properly trained staff, the laboratory does not work at full efficiency. The number of mercury analysis could be easily multiplied with very small investments.

Kochin method is currently used for mercury determination.

HALIKÖY

A. Collection of basic available data

The collection of basic available data on the technical aspects of the mining and smelting facilities of Haliköy, was started during the first mission visit to Haliköy area. However, questionnaires of Annexes A and B have not been replied yet by Haliköy experts. Therefore the following information includes only the data that our team has been able to collect during its stay at project area and are partially incomplete.

B. Mining

B.1. Orebodies

The plant of Haliköy receives ores from one orebody, where four mineralized pods exist. These pods are parallel to the faults of Bedyas and contain many small diaclases where mineralization appears. All mining workings are performed underground. Mineralization is fundamentally found in micaschists although enrichment zones are mainly located in clays at the contact zones of gneiss and micaschist (faults zone). Mercury mineral is normally cinnabar, with traces of metacinnabar. As associated minerals, pyrite and quartz are present with traces of magnetite and limonite. Traces of sulphur and arsenic are found.

B.2. Data card and map

In the following data card all basic available data of Haliköy orebody are put together. The corresponding map shows its location in Turkey.

A[illegible] [illegible]ce for Non-Ferrous Metals

MERC[illegible] EXTRACTION (TURKEY)

DATA CARD

Name of Mine

- Halıköy

Owner

Data not available

Geographical Location

Halıköy mercury plant is in the W. of Turkey, north of the Menderes [illegible] and in the boundaries of the town of Ödemis. Four mineralized rods are mined (Mine nº 2, Çayalti, Halıköy-Alt and Alakeçi). Each one of them has about 120 m lenght and from 8 to 25 m of thickness. All of them are parallel to the [illegible] fault.

Coordinates

N: [illegible]

W: [illegible]

Z: [illegible]

Bibliography and References

Data not available

Geological Description

The orebody is located in paleozoic grounds and mineralization is found in micaschists and contact clays with the [illegible] fault. Contact between gneiss and micaschists is discordant, producing the tilting of one of the contact [illegible].

The mineralization line, coincident with the fault is NW-SE with [illegible] NE dip.

Ore Body Geomorphological Description

The orebody formation is epithermal with the following [illegible]: quartz, pyrite, [illegible] and cinnabar. In some areas mineralization appears at clays on the contact of gneiss and micaschists (fault zone). Quartz and pyrite occur as hydrothermal intrusions into the orebody.

Topographical and Climatical Description

The district has a hilly appearance in the E-W direction and the mercury plant is about [illegible] m high. The climate of the region is mediterranean with mild winters and hot summers. Vegetation is abundant, its [illegible] character being influenced by topography.

INVESTIGATIONS

Chemical Analyses

Average [illegible] grade ([illegible]-1971) is [illegible] kg. The grade shows strong fluctuations.

Geological Studies

Data not available

Geophysical Survey

Data not available

Geochemical Survey

Data not available

Investigation of Bore Holes, Pits and Galleries

Data not available

Pilot Plant

No pilot plant had been established in the region before the construction of the [illegible] [illegible]

Investigations and Studies Proposed

Out of the scope of this report.

MINERAL RESOURCES

Resources description

Data not available.

Verified Tonnage

Probable Tonnage

Possible Tonnage

TECHNICAL TABLE

Description of Exploitation Methods

Sub-level stopping methods are performed, with [illegible] as the [illegible] of the [illegible] very low.

Mining Extraction Tonnage

	[illegible]	[illegible]	[illegible]
Tonnage:	[illegible]	[illegible]	[illegible]

Concentration or Process Method Description

The ore is not submitted to any concentration methods.

Tonnage Concentrated or Processed

The following tonnages of ore have been processed at the metallurgical plant.

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

Description of Transport Methods to Metallurgical Factory or Sale Destination

[illegible] is about [illegible] from the [illegible] factory. The ore is carried to the factory by [illegible] trucks of [illegible] tons capacity.

MISCELLANEOUS

Personnel (Technicians and Workers)

[illegible] personnel are [illegible] employees and workers. From [illegible] figure [illegible] are allocated to metallurgical [illegible] and [illegible] to general services.

Machinery

[illegible] available

Water Supply

The plants are supplied [illegible] water [illegible], [illegible]

Power Supply

Power is supplied by the State electricity network. The plant has available [illegible]

Other Supplies (Fuel oil, Compressed air)

[illegible] available

REMARKS

Client: UNIDO	Consultant: TECHNOERIA	

B-560

81.08.25

2 OF 3

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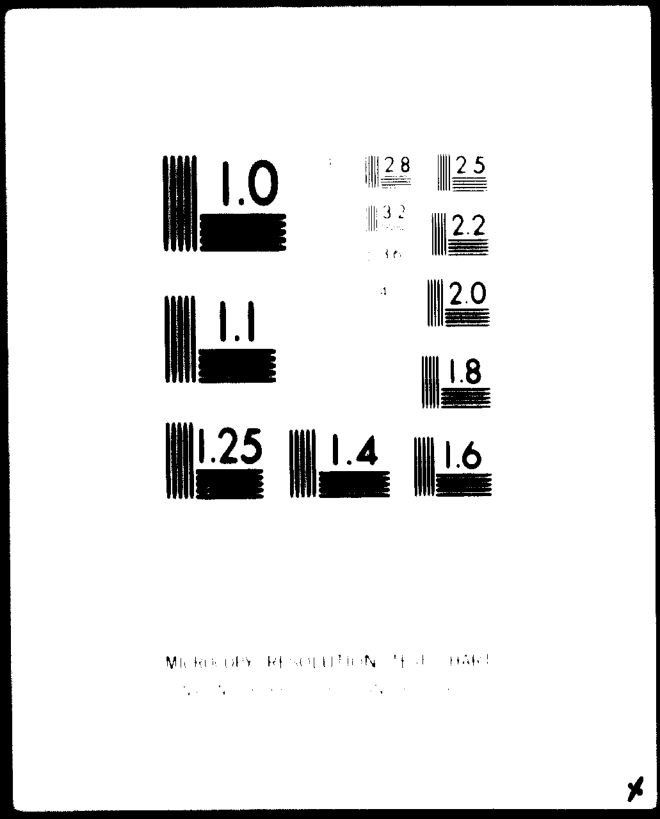

24x

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C. Metallurgical Plant

C.1. Location

The Haliköy mercury plant is located upon the orebody of its name. This location is the origin of some problems by buildings subsidence due to settlement of mining operations.

C.2. Plant Description

Flow sheet or ore preparation

The present flow-sheet of ore preparation installations up to the ore storage bin is shown in drawing III-5, enclosed.

Incoming ore loaded trucks dump on a 125 tons bin. (máx. capacity 150 tons) (1).

Under the bin a feeder (2) takes the ore to a - jaw crusher (3) with the following operating characteristics:

- Capacity 30 t/h
- Feed size: Up to 350 mm
- Product size: Below 50 mm

Flow sheet of kiln feeding system

A conveyor belt (4) of 41 m length, 600 mm width and 14º slope takes the crushed ore to a 100 tons bin (5) covered by a fixed grid.

From the bottom of the bin a feeder sends the ore to the conveyor belt (6) discharging at the kilns' feeding hoppers (8). An automatic sampling device (7) is located at the discharge of the conveyor belt.

Flow sheet of ore roasting, condensing and mercury recovery system

(See drawing III-5)

Operating Data

The plant works 24 hours a day in 4 shifts. The amount of ore roasted in each kiln per shift is 16 to 18 tons. The kiln entrance temperature is between 270-300º C and the ore is roasted up to 950-1,110º C. The kilns are driven by a 7.4 KV electric motors and speed reducer. Rotating speed of the kilns is constant. The calcines, contain only traces of mercury.

The kilns are heated with a burner which atomizes fueloil by compressed air. Fuel oil is fed to the burners by gravity. The exhaustgases are sucked by a 300 Nm^3/minute capacity fan, located at the end of the system, between the scrubbers and the stack duct. The condensing pipes are washed with clean water every day at the 6-12 shift. The pond is cleaned every day and the stupps taken to the hoeing machine. The used stupps were treated up to April 1972 in a fuel oil fired retort, together with the muds of lesser mercury content of the settling tanks. This procedure has been given up in April 1972, taking into account the great fuel consumption in the retort. Present flow sheet without retort use is shown at drawing III-5.

Tons of mercury production from 1969, 1970 and 1971 were 91,078; 100,050 and 100,240 respectively. Efficiency recovery is below 85%.

Equipment data

The plant has two roasting rotary kilns whose operation started in 1.961 and 1.963 respectively. Both kilns have the same operating characteristics.

The following description of equipment applies to each one of the kilns, unless otherwise stated.

Rotary kiln characteristics

Length	25,60 m.
Diameter:	1,50 m
Slope:	5%

Cyclones type and efficiency

Same as in Konya

Condensing system dimensions and materials used in its construction.

The condensing system is in 3 rows of pipes, with its diameter 40 cm. The total length is 430 m per kiln. The construction material is cast iron.

Exhaust fan characteristics

- **Speed** of gas flow:	Not known
- Capacity:	300 m³ minute
- **Depression**:	

Scrubbers

Kiln I, has a **scrubber on reinforced concrete sprayed with water. Dimensions of the scrubber are 7 x 3 x 5,20 m.**

Kiln II, has two scrubbers in series of wooden construction and cilindrical shape. Dimensions are 2,4 m

(Ø) x 6 m.

Stack line and stack

Stack line: 0,8 x 1,50 x 120 m, made of concrete

Stack: 0,8 m (Ø) x 12 m made of steel sheet

Characteristics of hoeing machine (one for both kilns)

Same as in Konya.

Operating parameters

a) Operating efficiency of rotary kilns =

$$= \frac{\text{Annual working hours}}{24 \times 365}$$

Average for both units: $\frac{7.608}{8.760} = 0,87$

b) Capacity efficiency = $\frac{\text{Actual capacity (tons)}}{\text{Theoretical capacity (tons)}}$

Average for both units = $\frac{75}{100} = 0,75$

c) Metallurgical efficiency = $\frac{\text{Hg (obtained)}}{\text{Hg (content)}}$

1.970 - (both units) = 82,10%

1.971 - (both units) = 82,90%

1.972 - (both units) = 86,90%

Other operating data

Kiln exit temperature (estimated):	950º-1.100º C
Kiln entrance temperature:	285ºC - 315ºC
Fan entrance temperature:	32 - 45ºC
Kiln exit vacuum:	Not available
Kiln entrance vacuum:	Not available
Cyclone depression:	6 cm. w.g.
Condensing system depressions:	6 to 8 cm. w.g.

Hg content in cyclone dust (estimated):	0.07%
Amount of dust (estimated):	0.6 - 1.0 ton day
Gas flow (Nm^3/hr) in relation to weight of treated ore:	Unknown

Specific Consumptions

Average fuel-oil consumption per ton of treated ore:

In kilns - 35 kg/ton

In retort - 500 kg/ton

Power consumption per ton of treated ore:

30 kwh/ton

C.3. Present operating conditions

C.3.1. Characteristics of treated ore

C.3.1.1. Mineralogical components

The only available data on the ore are given below and correspond to a chemical analysis (dry basis) made at Ankara on 30 December 1966.

	%		%
Hg	0.26	Na_2O	1.32
S	2.85	K_2O	2.02
SiO_2	59.56	CaO	0.43
Fe_2O_3	6.95	MgO	0.26
Al_2O_3 TiO_2	18.05	Fire losses	8.54
Na_2O	1.32		

On the same sample, X-Ray difraction analysis were performed, and quartz, sericite, crisolite, kaolinite and limonite have been found. Also, pyrite, magnetite, cinnabar and antimonite were present.

The gangue is of argilous nature, and its abrasive nature may be estimated as mild.

C.3.1.2. Granulometry

The run of mine ore is received at the plant in sizes up to 350 mm.

The amount of fines in the ore is very substantial, the 55.75 per cent of the calcined ore being finer than 5 mm.

Sizes of feed ore to the kilns was usually set at 50 mm.

C.3.1.3. Humidity

Surface wetness amounts to a minimum of 6% and a maximum of 12%.

No special difficulties in plant operations have been reported due to this fact.

C.3.2. Process Control

C.3.2.1. Treated ore and calcines

The data concerning the months of March and April are given in Tables III-6 and III-7, which show data on treated ore tonnage, mercury grade of the ore and mercury content of the calcines.

Collected samples are sent to the laboratory at the end of every shift. Only one analysis is performed at the end of the shift and the mean of the four shifts is given.

C.3.2.2. Stack Gases

No control is performed regularly on the stack gases. Reported values of 10-20 mg Hg m^3 are not significant, as they do not correspond to the true losses through the stack, taking into account the difficulties to achieve proper gas sampling.

C.3.2.3. Cyclone Dust

Total amount of cyclone dust is not known, and its measurement is very difficult, as dust is sprayed with water and sent to waste. Estimations of the amount of cyclone dust by plant personnel are in the range of 500 1000 kg. per day.

C.3.2.4. Stupps

No information is available on stupps fed back to the kiln nor mercury grade, taking into account that - stupps have been up to now treated in the retort

C.3.2.5. Waste Waters

Flow measurements and mercury content of waste waters had never been performed up to now. Nevertheless mercury losses by this concept may be important, as discussed in Section IV of this Report.

C.3.2.6. Kiln Gases

CO_2 and O_2 analysis of kiln exit gases had never been performed up to now. Therefore excess air in combustion, duct velocities of the gases, etc., are unknown.

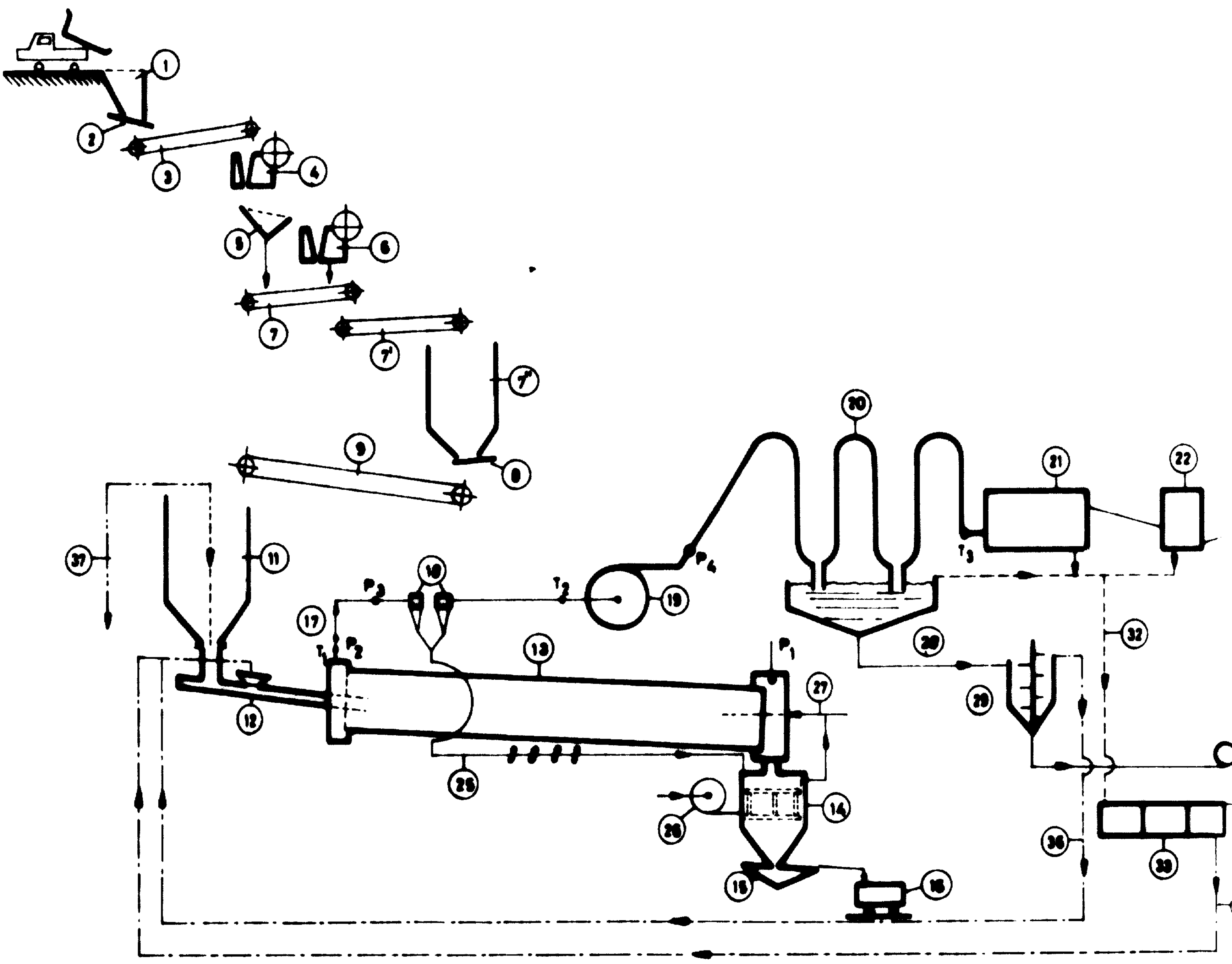
Fuel and air inputs were left at judgment of the kiln operator.

C.3.2.7. Temperatures and Pressures

Information about temperatures and pressures for the month of January is included in Table III-8. Apparently little or no importance was paid to these control figures.

C.3.2.8. Laboratory Analysis

The Haliköy laboratory is inadequately equipped and staffed for proper control purposes. Colorimetric method is currently used for mercury assaying of samples.



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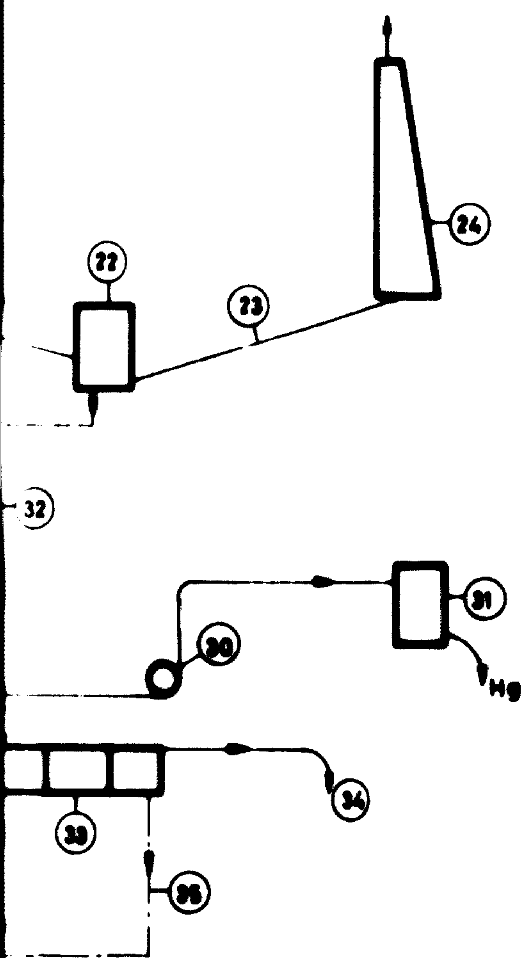
SECTION 1



A. Ore Feeding Circuit

1. Mine-run ore bin with fixed screen (350 mm
2. Ore feeder
3. Mine-run ore conveyor belt
4. Primary jaw-crusher (35 t/h)
5. Vibrating screen
6. Secondary jaw-crusher (35 t/h)
7. 7'. Conveyor belts
7''. 1000 t. ore bin
8. Ore feeder
9. Crushed ore conveyor belt
10. Automatic belt scale
11. 50 t. crushed ore bin
12. Kiln's vibrating feeder

B. Calcining Circuit

13. Rotary kiln. (175 t/day rated capacity)
14. Calcines' bin
15. Vibrating feeder
16. Calcines' wagon to refuse dump

C. Fuel Circuit

27. Fuel oil burner. Preheated air
26. Primary preheated combustion air

D Gas Circuit

17. Kiln's exit gases duct. (Ø 720 mm)
18. Cyclones (2 units in series)
19. Fan (Q, 300 Nm^3/min; H, 280 mm w.g.)
20. Condenser (4 series in parallel. Ø 403 mm &
297 m. length)
21. Water scrubler
22. Gas chamber
23. Stack duct
24. Stack

E. Mercury Circuit

28. Duct from pond to Exeli hoeing machine
29. Exeli hoeing machine
30. Mercury filling pump
31. Mercury container and flasks filling (by volu

F. Dust, Stupps and Refuse Waters Circuits

25. Cyclones' dust screw conveyor to calcines' b.
32. Refuse waters from stupps pond, scrubler, a
gas chambers to settling tanks
33. Settling tanks

SECTION 2

KEY

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on with fixed screen (350 mm)

onveyor belt
rusher (35 t/h)
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crusher (35 t/h)
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onveyor belt
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5 t/day rated capacity)

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to refuse dump

Preheated air
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es duct. (Ø 720 mm)
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to Exeli hoeing machine
achine
pump
ner and flasks filling (by volume)

Refuse Waters Circuits

screw conveyor to calcines' bin
from stupps pond, scrubler, and
to settling tanks

34. Refuse waters to waste or to condensers' washing
35. Stupps from settling tanks to kiln's feeder (manual handling)
36. Hoed stupps to kiln's feeder (mannual handling

G. Ore Sampling

37. Automatic sampler (to laboratory)

Konya Flow-sheet

DRAWING III-1

SECTION 3

TABLE III-2

Selected control parameters from Konya Plant

(March 1972)

Date	% Hg, ore	% H_2O, ore	Wet ore, tons	% Hg, Calcines	Calcines, tons	% Hg, cyclone dust
1	0.217	4.35	123,328	0.004	117,300	0.068
2	0.212	2.99	140,219	0.003	135,700	0.084
3	0.244	3.8	153,442	0.002	147,200	0.076
4	0.244	4.1	141,898	0.003	135,700	0.120
5	0.237	4.2	127,590	0.004	121,900	0.086
6	0.212	2.72	129,164	0.003	125,350	0.030
7	0.191	4	145,265	0.004	139,150	0.052
8	0.205	3.6	137,509	0.004	132,250	0.080
9	0.187	3.5	148,240	0.004	142,600	0.042
10	0.245	3.47	120,661	0.004	116,150	0.046
11	0.183	3.75	113,744	0.002	109,250	0.080
12	0.199	3.52	132,595	0.004	127,650	0.060
13	0.238	5.35	110,870	0.005	104,650	0.090
14	0.273	4.37	125,458	0.006	119,600	0.042
15	0.217	4.18	139,566	0.001	133,400	0.138
16	0.243	4.47	126,754	0.006	120,750	0.086
17	0.276	3.65	153,258	0.005	147,200	0.120
18	0.232	4.00	163,350	0.005	156,400	0.116
19	0.281	2.65	163,539	0.003	158,700	0.160
20	0.368	2.61	163,631	0.004	158,700	0.084
21	0.366	2.7	149,538	0.010	144,900	0.848
22	0.399	2.4	152,683	0.005	148,350	0.156
23	0.320	3.00	146,355	0.006	141,450	1.686
24	0.271	2.43	157,239	0.008	152,950	0.160
25	0.233	2.8	162,516	0.005	157,550	0.524
26	0.255	3.27	147,768	0.005	142,600	0.074
27	0.365	4.47	130,571	0.005	124,200	0.120
28	0.282	3.00	146,292	0.005	141,450	0.104
29	0.151	2.5	151,231	0.002	147,200	0.084
30	0.250	2.72	152,930	0.006	148,350	0.166
31	0.315	2.43	121,060	0.004	117,700	0.404
Total	(1) 0.256	(1) 3.20	4,183,328	(1) 0.004	4,038,100	(1) 0.193

- Source: Own elaboration, based on daily control forms
- Units: Percentages and metric tons
- Note: (1) Monthly average

TABLE III-3

Selected control parameters from Kenya Plant

(April 1972)

Date	% Hg, ore	% H_2O ore	Wet ore, tons	% Hg, calcines	Calcines, tons	% Hg, cyclone dust
1	0.311	2.36	133,404	0.005	129,800	0.164
2	0.319	2.3	150,282	0.004	146,300	0.066
3	0.278	3.1	134,375	0.002	129,800	0.050
4	0.309	4.37	133,905	0.006	127,600	0.172
5	0.353	2.83	148,891	0.004	144,100	0.076
6	0.246	2.70	147,377	0.006	143,000	0.096
7	0.311	1.81	150,641	0.003	147,400	0.082
8	0.184	1.9	133,710	0.002	130,900	0.154
9	0.255	2.11	149,882	0.006	146,300	0.190
10	0.244	2.4	145,789	0.003	141,900	0.140
11	0.217	3.00	137,553	0.004	133,100	0.158
12	0.172	3.3	122,939	0.003	118,659	0.478
13	0.264	2.3	135,818	0.003	132,300	0.114
14	0.265	1.8	135,125	0.005	132,300	0.458
15	0.370	2.58	128,797	0.008	124,950	0.178
16	0.273	1.67	140,314	0.008	137,550	0.206
17	0.369	2.27	134,857	0.009	131,250	0.116
18	0.311	2.30	-	0.003	-	0.256
19	0.266	-	-	-	-	-

- Source: Own elaboration, based on daily control forms
- Units: Percentages and metric tons

TABLE III-4

Daily averages temperatures and pressures at Konya metallurgical plant

(March 1972)

Date	Temperatures (ºC)			Pressures (+) or Depressions (-)			
	T_1	T_2	T_3	P_1 (-)	P_2 (-)	P_3 (-)	P_4 (+)
1	248	262	25	0.4	0.8	n. a.	3.65
2	251	266	21	0.4	0.8	"	3.57
3	246	273	23	0.4	0.8	"	4.10
4	255	273	23	0.4	0.8	"	3.15
5	267	280	26	0.4	0.8	"	3.32
6	285	294	20	0.4	0.8	"	3.05
7	282	292	13	0.4	0.8	"	3.72
8	268	285	24	0.4	0.8	"	3.35
9	270	282	28	0.4	0.8	"	3.95
10	271	291	26	0.4	0.8	"	3.57
11	271	279	26	0.4	0.8	"	3.70
12	267	283	17	0.4	0.8	"	3.22
13	269	275	12	0.4	0.8	"	4.38
14	284	298	9	0.4	0.8	"	4.33
15	269	285	17	0.4	0.8	"	3.65
16	279	294	28	0.4	0.8	"	3.80
17	264	282	22	0.4	0.8	"	3.85
18	266	283	20	0.4	0.8	"	2.40
19	270	281	18	0.4	0.8	"	2.67
20	277	280	19	0.4	0.8	"	3
21	265	284	19	0.4	0.8	"	3
22	268	276	21	0.4	0.8	"	3.2
23	269	275	22	0.4	0.8	"	3.98
24	254	267	20	0.4	0.8	"	2.90
25	249	260	7	0.4	0.8	"	3.32
26	262	271	21	0.4	0.8	"	3.61
27	261	264	24	0.4	0.8	"	2.75
28	256	266	23	0.4	0.8	"	2.63
29	243	253	25	0.4	0.8	"	3.36
30	236	253	27	0.4	0.8	"	3.30
31	250	274	24	0.4	0.8	"	3.70

- Source: Own elaboration based on shift control forms.
- Units: Degrees Centigrades and cm. w. g.
- Notes:

T_1, Gas temperatures at kiln's feed end
T_2, Gas temperatures at fan's inlet
T_3, Gas temperatures at cold end of condensers
P_1, Depressions at kiln's burner end
P_2, Depressions at kiln's feed end
P_3, Depressions at cyclones' inlet
P_4, Pressures at hot end of condensers

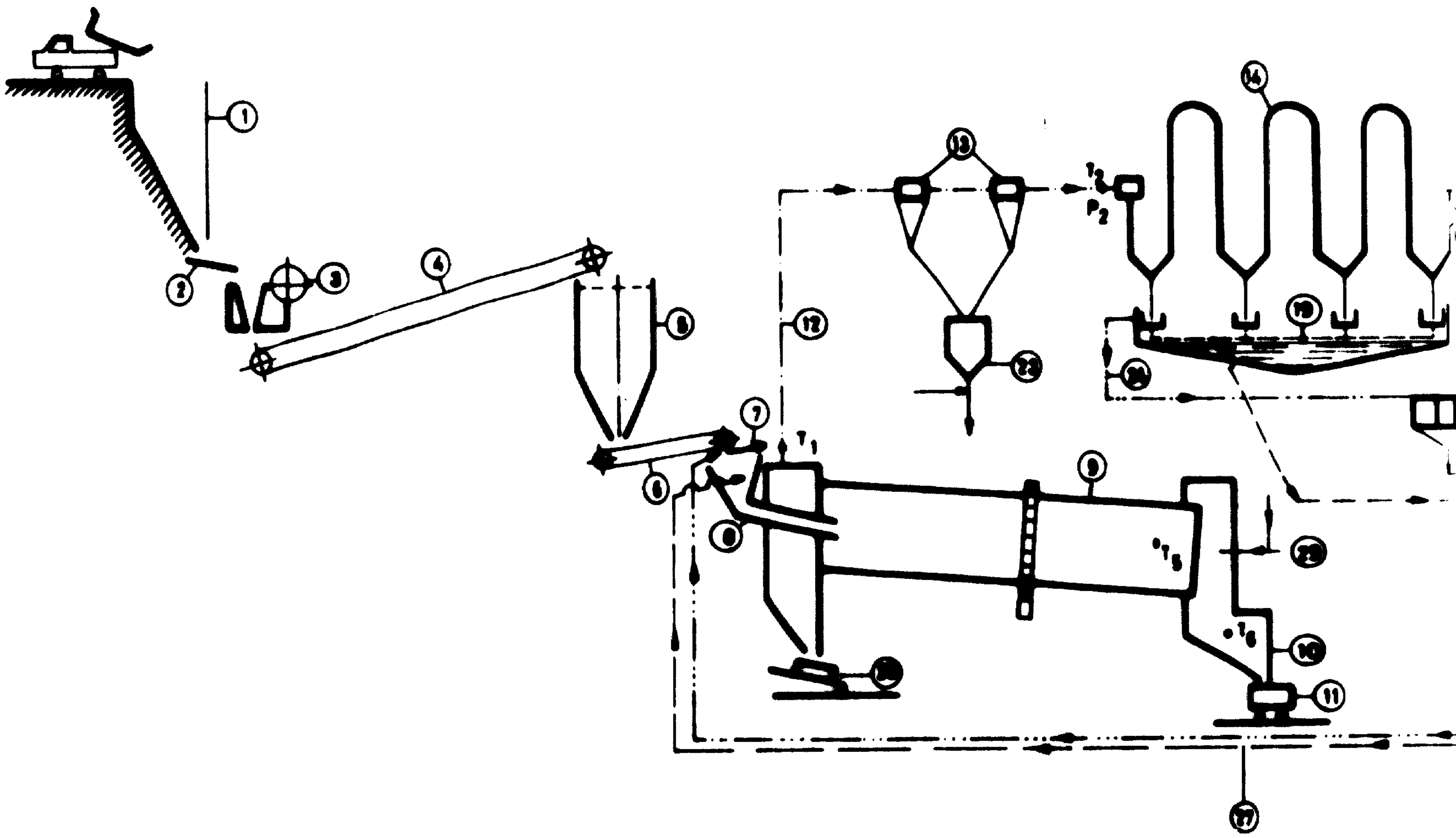
TABLE III- 6

Selected control parameters from Haliköy Plant

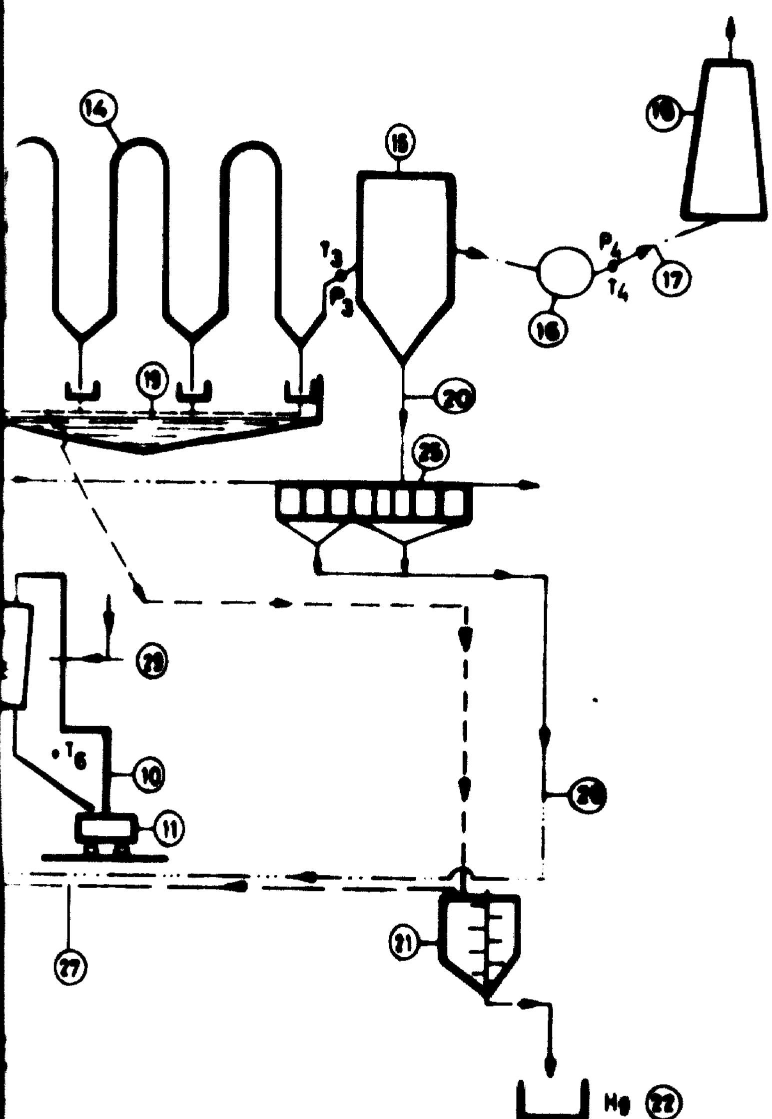
(March 1972)

Date	% Hg, ore	% H_2O ore	Feed-ore (dry) tons	% Hg in calcines	Calcines, tons
1	0.250	10	102		93
2	0.190	9	120		110
3	0.250	9	100		91
4	0.320	9	129		117
5	0.300	9	131		119
6	0.380	10.5	105		95
7	0.370	10	122		111
8	0.350	9	126		115
9	0.340	9	135		123
10	0.380	9	126		115
11	0.480	8.5	121		110
12	0.380	10	121		110
13	0.280	9	103	Traces	93
14	0.200	9	143		130
15	0.420	8.5	130		118
16	0.250	9	144		131
17	0.250	9.5	144		131
18	0.290	9.5	144		131
19	0.320	10	127		116
20	0.290	8.5	121		110
21	0.190	8.5	130		118
22	0.270	9	120		109
23	0.250	9.5	130		118
24	0.280	9	129		117
25	0.210	10	144		131
26	0.200	8	114		104
27	0.240	8	122		111
28	0.190	9	144		131
29	0.280	9	144		131
30	0.300	10	101		92
31	0.300	10	144		131
Total	0.345 (1)	9.19 (1)	3,922		3,569

- **Source: Own elaboration, based on daily control forms**
- **Units: Percentages and metric tons**
- **Note: (1) Monthly average**



SECTION 1



A. Ore Feeding Circuit

1. Mine-run ore bin (150 t)
2. Ore feeder
3. Jaw crusher (30 t/h)
4. Crushed ore conveyor belt
4'. Stand-by jaw crusher
5. 100 t Ore bin with fixed screen
6. Conveyor belt to kiln's feeder
8. Kiln's impact feeder

B. Calcining Circuit

9. Rotary kiln (100 t/day rated capa
10. Calcines' bin
11. Calcines' wagon to refuse dump

C. Fuel Circuit

29. Fuel oil burner (gravity fed). Am

D. Gas Circuit

12. Kiln's exit gases duct (Ø 400 mm
13. Cyclones (4 per kiln)
14. Condenser (3 series in parallel.
and 140 m. length)
15. Water scrubber
16. Fan (Q, 300 m^3/min; N, 30 KW)
17. Stack duct
18. Stack

E. Mercury Circuit

19. Condenser's water seal containers
20. Stupps from scrubber
21. Exeli hoeing machine
22. Flasks filling

SECTION 2

K E Y

cuit

e bin (150 t)

(30 t/h)
conveyor belt
crusher
in with fixed screen (50 mm)
lt to kiln's feeder
t feeder

t

(100 t/day rated capacity)
n
gon to refuse dump

ner (gravity fed). Ambient air

gases duct (Ø 400 mm)
per kiln)
3 series in parallel. Ø 400 mm.
length)
ber
) m^3/min; N, 30 KW)

t

water seal containers
scrubber
machine
g

F. Dust, Stupps and Refuse Water Circuits

23. Cyclone dust chamber (washed with water)
24. Overflow water from condenser washing
25. Settling tanks
26. Stupps from settling tanks to kiln's feeder (mannual handling)
27. Hoed stupps to kiln's feeder (mannual handling)
28. Kiln's overflow (to waste)

G. Ore Sampling

7. Automatic sampler (to laboratory)

Holikdy Flow-sheet

DRAWING III-5

SECTION 3

TABLE III-7

Selected control parameters from Haliköy Plant

(April 1972)

Date	% Hg, ore	% H_2O ore	Feed-ore (dry) tons	% Hg in calcines	Calcines, tons
1	0.200	9	122		111
2	0.210	8.5	126		115
3	0.240	9	125		114
4	0.360	9.5	126		115
5	0.220	9.5	99		90
6	0.420	10	62		56
7	0.570	9	64		58
8	0.520	10	58		53
9	0.480	9.5	69		63
10	0.330	9	66		60
11	0.580	9.5	62		56
12	0.450	10	67		61
13	0.400	9	66		61
14	0.340	10	60	Trace	55
15	0.380	9.5	66		61
16	0.330	9.5	88		80
17	0.140	9	142		130
18	0.160	9	135		123
19	0.240	10	119		109
20	0.310	10.5	129		118
21	0.260	10	131		120
22	0.240	9	112		102
23	0.260	9.5	130		11[illegible]
24	0.430	9.5	130		11[illegible]
25	0.350	10	134		12[illegible]
26	0.430	9.5	129		118
27	0.590	8.5	121		110
28	0.700	8.5	113		103
29	0.440	8	126		115
30	0.490	8	144		131
Total	0.283 (1)	9.2 (1)	3, 120		2, 839

- Source: Own elaboration, based on daily control forms
- Units: Percentages and metric tons
- Note: (1) Monthly average

TABLE III- 8

Daily averages temperatures and pressures

at Haliköy metallurgical Plant

(January 1972)

Date	Temperatures ºC			Depressions (-)	
	T_1	T_2	T_3	P_1 (-)	P_2 (-)
1	304	286	34	60	80
2	310	282	35	60	80
3	323	287	31	60	80
4	310	280	33	60	80
5	304	281	33	60	80
6	297	282	33	60	80
7	301	279	34	60	80
8	298	266	35	60	80
9	300	284	35	60	80
10	309	281	37	60	80
11	306	281	36	60	80
12	304	279	37	60	80
13	304	278	32	60	80
14	306	283	27	60	80
15	304	276	31	60	80
16	308	281	34	60	80
17	305	280	32	60	80
18	304	280	36	60	80
19	309	280	37	60	80
20	303	279	36	60	80
21	303	277	36	60	80
22	304	276	37	60	80
23	305	278	36	60	80
24	304	277	35	60	80
25	303	276	37	60	80
26	303	271	36	60	80
27	305	274	36	60	80
28	304	274	37	60	80
29	304	277	34	60	80
30	307	273	38	60	80
31	305	282	37	60	80

- Notes:

  T_1, Gas temperatures at kiln's feed end

  T_2, Gas temperatures at hot end of condenser

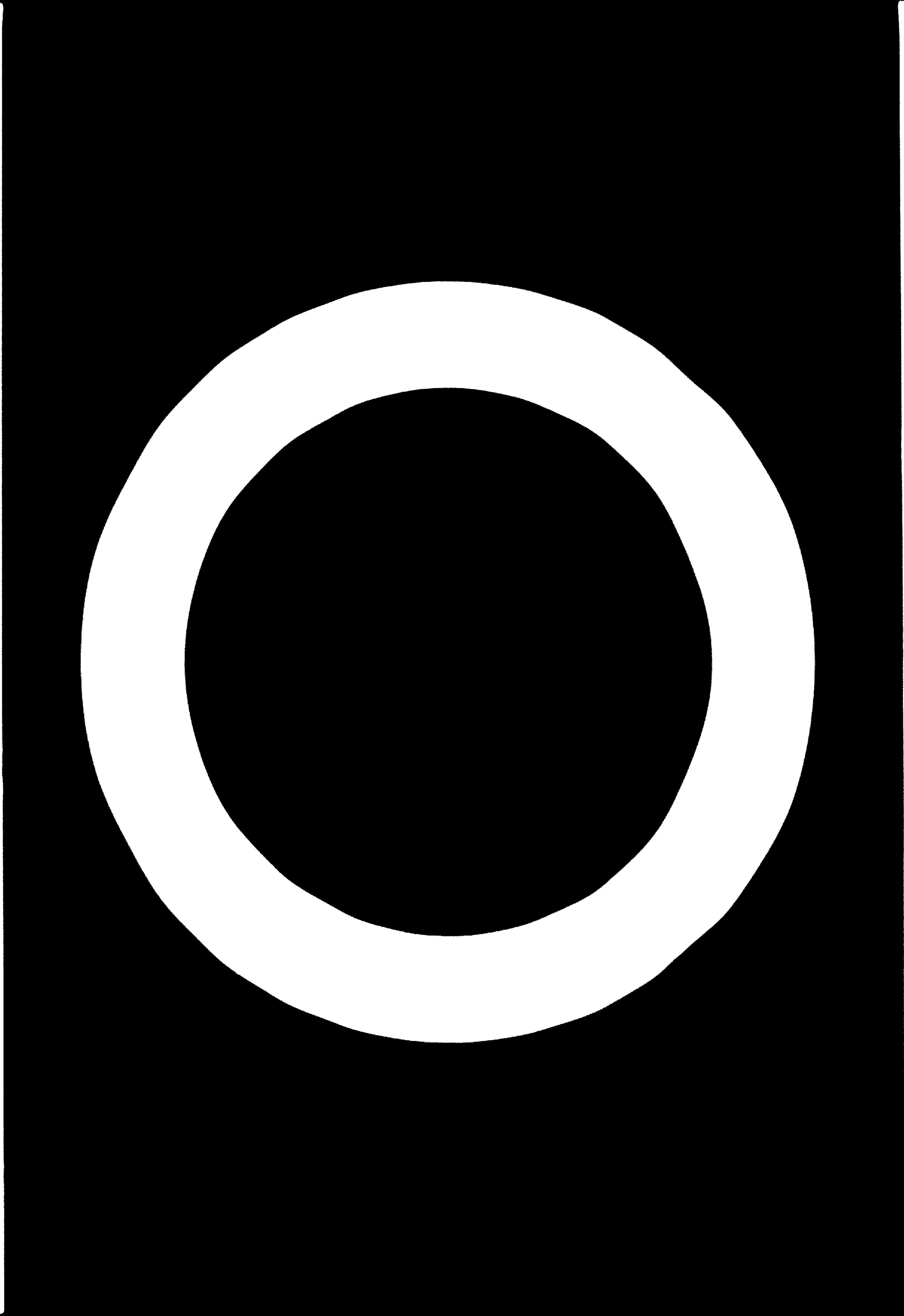
  T_3, Gas temperatures at cold end of condenser

  P_1, Depression at hot end of condenser

  P_2, Depression at cold end of condenser

- Source: Own elaboration based on shift control forms
- Units: Degrees Centigrades and mm. w. g.

IV. ANALYSIS OF OPERATIONS AND EFFICIENCY ASSESSMENTS



IV. 1 INTRODUCTION

This part of the report includes an analysis of the situation of the plants of Konya and Haliköy, such as it presented itself at the moment of the stay of the second consulting mission at the Project area.

To analyze existing smelting operations and assess present efficiency in mercury recovery and provide specific recommendations on their possible improvements, tests were performed at both plants with the following general programme:

- Processing of 1.000 tons aprox. of ore, controlling all the variables of the operation, i.e.:

1. Ore grade

   In order to determine the grade of the ore and its pattern of variability, samples will be taken every hour, assigning to each sample the corresponding weight of incoming ore.

   From each sample 3, analysis will be made at the laboratory

2. Calcines

   To determine mercury content of burnt ore or calcines, samples will be taken from every wagon sent to dump, soaking the samples in water and putting together all samples belonging to the same shift. Four analysis will be made from each composite sample giving the average mercury content in calcines at each shift.

3. Mercury production

   Daily mercury production will be recorded.

4. **Stupps**

Sporadic analysis of mercury content will be performed.

5. **Waste waters**

Periodical flow gaging of waste waters will be performed, in order to determine the average daily amount, together with mercury losses in such waters.

6. **Exit gases**

CO_2 and O_2 contents of kiln exist gases will be measured with the Orsat analyzer.

Where possible, scrubbers and gas chambers will be cleaned before the test, and at the end of the same deposits will be cleaned, weighing their amount and assaying their mercury content.

Similarly, a section of the stack duct will be cleaned in order to start the test with a clean section, Once the test is over, cleaning of the deposits will be made, weighing and analysing them. Such procedure will allow a fair idea of the importance of mercury losses though the stack.

7. **Temperatures and pressures**

Every hour pressures and temperatures will be read and recorded at the following points of the circuit:

- Exit of the kiln
- Exit of cyclones
- Hot end of condensers
- Cold end of condensers

IV. 2. KONYA

1. Tests performed and their results

The tests programmed in the above paragraph took place at Konya from the 20. 4 1972 up to the 28. 4. 72. With the exception of the mercury content of the stupps, and the cleaning of the scrubbers, gas chambers and stack duct given the great inconvenience that such cleaning supposed to the normal operation of the plant (as a minimum of 48 hours were estimated necessary by plant personnel for the cleaning, with the corresponding shutdown of operations and loss of - mercury recovery), all the remaining parts of the programme were carried through.

2. Development of the tests

While the tests were going on, we found some equipment difficulties .that were corrected, at least in part.

The most important were:

- Ore sampling device

It was observed that the cutter pick up a very small amount of ore and that, of course, it did not picked up coarse - sise particles, since its opening was very narrow. We had to enlarge the opening, although not in such a scale that was sufficient for the sise of ore entering into the kiln.

- Ore conveyors

They were frequently empty and this caused that the 50 t hopper to be empty, too and the feeding of the furnace was made in an irregular manner.

- Kiln feeding

While a man had no other task than to check the feeding of the furnace in a regular manner, the feeder was running frequently empty, especially at night shifts.

- Clogging of condensers and cyclones

Very frequently the pressure gauges showed strong variations in their readings, a sign of clogging at different points of the circuit. To settle the proper situation again, needed a lot of time.

- Disposal of calcines

Given the toxicity of this operation, workers did not always leave the discharge opening plugged with calcines and that gave rise to abnormal air intake into the kiln.

- Combustion system

Very often, the conditions in the system suffered a variation, above all in the moment of changing shifts. The blower injecting hot air to the burner was started or stopped, its flow altered, etc., without standard procedures for its operation.

- Slag formation

Slags were frequently formed, forcing us to stop the kiln without achieving a proper regularity in operation.

3. Results of the test

As a consequence of the alterations introduced in our program by the above said irregularities, the results we got were really abnormal.

If the Konya plant is to achieve normal results, a regular and continuous operation of the furnace and condensation system is a must. If this matter does not receive proper attention, the present situation will be never improved.

It is necessary that the technicians in the plant -- convey their concern to the men under them.

A large amount of analysis was made with the following results:

3.1 Feed Ore

Table IV.1, shows data on amounts of ore fed, average daily mercury grades, mercury content and mercury produced, together with the calculated efficiencies during the running of the tests.

The amounts of ore fed to the kiln have been calculated using the number of wagons of calcines obtained during the tests. 974 wagons were totalized, which assuming a unit weight of 1,050 kg of calcines per wagon gi a total of 1,022.70 tons of calcines.

Size of ore fed to the kiln was 40/45 mm during the first 5 days of the test. During the last four days, the size was increased up to 80/85 mm.

Although the results of the bigger size feed are not conclusive, taking into account that a much longer time of operation with the bigger size ore is necessary in order to draw proper conclusions, it can be seen from Table IV.1., that the capacity of the kiln showed a light increase at the end of the test.

Slag formation showed no tendency to increase with the bigger size of ore.

3.2. Calcines

Table IV.2., shows the amounts (1) of calcines produced during the tests, together with their percentages of mercury content, and corresponding mercury losses.

It can be observed from that table that the mercury content of calcines are substantially higher than the -- 0.004% reported as average for 1971 in Konya informations.

The reason for such difference, is based in that analysis of calcines were performed customarily at Konya on samples allowed to cool from exit temperature of the calcine bin to ambient. During cooling time distillation of mercury takes place and therefore the assayed samples at the laboratory shows systematically a much lower content of mercury than the true one.

Analysis shown in Table IV.2. have been performed, following our instructions, on samples that were soaked into water immediately after their exit from calcines bin, preventing then, mercury losses by distillation.

It can be seen, then, that the losses in the calcines represent a 3.33 percent of the mercury contained in the ore.

The necessity of achieving a good roasting of the ores is justified by the fact that if in the year 1971, an output of 122,514 Kg of Hg was obtained with a return or efficiency of 82.77 percent, the losses amount to 4,721.75 kgs (136.8 flasks), that, at the average price of 4,500 L.T/flask, represent 615,600 L.T/year.

(1) The exact amount of calcines is slightly below the treated ore figure, taking into account fire losses.

It is then indispensable to achieve a good roasting of the ores. In order to correct this deficiency in the process, the procedures recommended in Section II of this report must be carried out.

3. 3. Temperatures and pressures

Temperatures and pressures were controlled at different points of the circuit, finding the difficulties that are mentionned in Section II (Problems on Process control). Because of that we cannot rely on the accuracy of the recorded measures.

As far as it concerns the suitable ranges for temperatures, and pressures, they should be between the following values:

a) Temperatures

At the exit of the furnace they must range from 300 to 320º C.

At the exit of cyclons, from 280 to 300º C, in order to achieve that the dust collected in the cyclon may contain a minimum amount of mercury.

At the exit of the condenser, they must range from 20 to 30º C.

b) Pressures

Those measured in normal conditions are correct; the readings in the gauges placed at the hot end of the condenser show fluctuations between 29 and 90 m/m c.a. We repeat that the deficiencies causing that anomalism must be corrected inmediately. They normally begin with obstructions at the condenser.

3.4. Excess Air

The irregularities of operation in the process are denounced by the readings of the Orsat analyzer.

An idea of the said abnormal operation is given by the series of readings included in Table IV. 3. With the Orsat, and even more easily with the oxigen analyzer whose description is included in Annex D it is necessary to settle the points where an excessive entry of air is taking place, in order to get a proper operation since an excess of air gives rise to:

- Losses by mechanical drag, and
- Losses by mercury non susceptible of condensation.

The application of the percentages of CO_2 and O_2 shown in Table IV-3 to formulas (2) and (3) of Annex C must give identical values for x, i.e., number of moles of air in exit gases.

For CO_2 = 9%, we have

$$\% \; CO_2 = \frac{125}{1.47 + 4.76.x} = 9$$

the, x = 2.60.

Substituting in formula (3) for x, the above value of 2.60 we have:

$$\% \; O_2 = \frac{(x - 2) \; 100}{1.47 + 4.76 \; x} = \frac{57}{13.82} = 4.1\%$$

Although none of the analysis of Table IV 3, corresponds exactly to those theoretical conditions, we must assume that CO_2 readings are more reliable, taking into account their more exact determination with the Orsat analyzer.

Therefore for % CO_2 = 9, the excess air above the theoretically needed, (x = 2), will be:

$$\frac{0.60 \times 100}{2} = 30\%$$

For the type of fuel-oil used at Konya a 15% excess air is suitable, therefore it can be concluded that as a rule, at Konya too much excess air is used, with the consequent mercury losses by mechanical drag and unsuitable condensign conditions.

3.5. <u>Cyclone dust</u>

Reported mercury content for cyclone dust during 1971 has been 0.075%. However these averages have been consistently higher during the month of March and first half of April. (0.193% and 0.175% respectively).

No analysis of cyclone dust is available for the test period.

3.6. <u>Waste water</u>

The amounts of waste water from settling tanks have not been gaged nor analyzed for mercury content taking into account the very small amounts evolved.

3.7. <u>Laboratory analysis</u>

Mercury analysis were performed at Konya by the Eschka (or Whitton) method. In this method, the ore or calcine sample, together with fluxes is placed in a metal retort tube closed at one end. A clean, weighed gold foil is placed over the open end and a metal cup clamped down on it, sealing the retort. The cup is filled with water, and as the retort is heated, the quicksilver vapor rises in the retort to the foil, on which it condenses and amalgamates. The foil is again weighed, the increase showing the amount of quicksilver in the sample.

By this method during the running of the tests a total of 400 analysis of ore samples have been performed together with near 40 analysis of calcines.

4. Materials balance and operation efficiencies

Tables IV.1 and IV.2, show the general materials balance for ore, recovered mercury and calcines.

Taking into account the relative short duration of the tests and the unbalances introduced into the system during the test period by the irregular sequence of stupps charging, materials balance are more representative considering total yearly figures. We shall use available data for 1971, to establish the approximative materials balance of the metallurgical operation at Konya.

4.1. Materials balance

Using 1971 data we have:

- Treated ore (wet)	65,084 tons (4% H_2O)
- Treated ore (dry)	62,480 tons
- Calcines	62,480 " (1)
- Average grade of ore	0.236%
- Average mercury content of calcines ..	0.008-0.009% (2)
- Average mercury produced	122.5 tons

(1) The exact amount of calcines is slightly below the treated ore figure, taking into account fire losses, but for the purpose of this calculation both figures may be considered equal.

The amount of residual stupps after lime treatment is not exactly known, but it is estimated that some 300 tons of stupps were fed back to the kilns after lime treatment. We have estimated for those used stupps the following analysis:

- Mercury:	18%	54 tons
- Lime:	50%	150 "
- Water:	20%	60 "
- Dust:	12%	36 "
	100%	300 tons

From the above figures the most significant fact is that during 1971, some 54 tons of mercury have been recycled.

4.2. Mercury recovery efficiencies

4.2.1. Metallurgical efficiency

During 1971, the metallurgical efficiency was:

$$\frac{\text{Mercury (obtained)}}{\text{Mercury (content)}} = \frac{122.5 \text{ tons}}{62,480 \times 0.236} = \frac{122.5 \text{ tons}}{147.5 \text{ tons}} = 0.83$$

4.2.2. Condensing efficiency

To calculate this efficiency, it is necessary to consider that all mercury contained in the ore, and the recycled mercury from the stupps, passes into the condensing system, less the mercury lost in calcines and cyclone dust. The latter will be disregarded in this calculation taking into account its small value.

Then,

Mercury from ore, 62,480 x 0.00236	+ 147.45 tons
Mercury from calcines 62,480 x 0.00009	- 5.58 tons
	+ 141.87 tons
Recycled mercury (from hoed stupps) 300 x 0.18	+ 54.00 tons
- Total to condensing	195.87 tons

Then, the condensing efficiency has been:

$$\frac{122.5 + 54.0}{195.87} = 0.9011$$

Which gives 9.89% losses in condensing.

Therefore, the fact of recycling 300 tons/year of hoed stupps to the kilns, produces a yearly loss of mercury which amounts to

$$300 \times 0.18 \times 0.0989 = 5.34 \text{ tons.}$$

Therefore, if stupps are not recycled, the condensing efficiency would be increased by

$$\frac{5.34}{141.87} \times 100 = 3.76\%$$

And the resulting condensign efficiency,

$$90,11\% + 3.76\% = 93.87\%$$

4.3. Operating efficiency of the kilns

This efficiency has been defined as the ratio:

$$\frac{\text{annual working hours}}{24 \times 365}$$

For 1971, this ratio reached a value of 0.834 for Kiln I, and 0.615 for Kiln II, which was not in operation for 5 months.

During the tests, the operating efficiency had the following value (Unit II only in operation).

Theoretical working hours (from 9 A.m. of 20-4-72 to 12 hours of 28-4-72) = 195 hours.

Hours of stoppages during the tests = 8.33 hours

$$\text{op. eff.} = \frac{186.67}{195} = 96\%$$

4.4. Capacity efficiency

This value is given by the ratio:

$$\frac{\text{Actual capacity (tons)}}{\text{Theoretical capacity (tons)}}$$

During 1971 this ratio reached a value of 0.760 for Unit I and 0.753 for Unit II.

During the performance of the tests (Unit II):

$$\frac{\text{Actual capacity}}{\text{Theoretical capacity}} = \frac{1,022.70}{150 \times 8.125} = 0.84$$

5. Specific recommendations

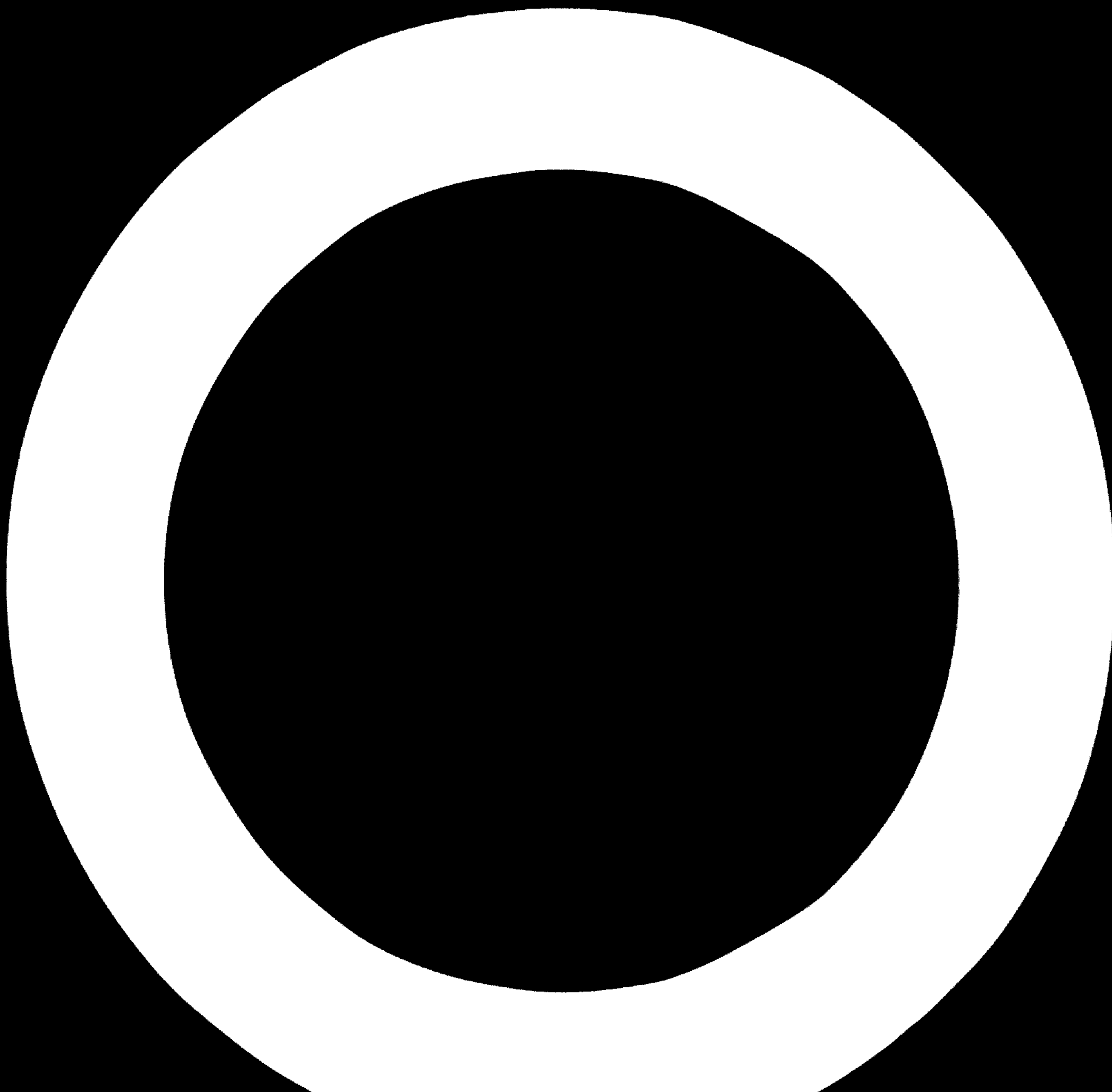
The results of the tests, together with the analysis of the problems encountered during our stay at Konya have allowed to provide specific recommendations on possible improvements in the following areas:

a) Mercury recovery and technology

b) Plant overall operation

c) Equipment maintenance and operations control

Such recommendations have been included in Section II of this Report.



IV. 3. HALIKOY

1. Tests performed and their results

Tests programmed in paragraph IV. 1. took place at Haliköy from the 2. 5. 1972 up to the 12. 5. 1972.

All the parts of the programme were carried through, with the exceptions already noted in the case of Konya, i. e. the mercury content assaying of stupps and the cleaning of scrubbers, gas chambers and stack duct.

2. Analysis of existing operating conditions

Before starting the tests and modifying the operating conditions, a careful analysis of the existing situation at the time was performed together with the measurement of various parameters in order to assess present operating conditions.

The most important results of such analysis are listed below:

- Only one mercury analysis of the ore is performed at every shift.
- Only one mercury analysis of calcines is performed per day.
- Excess air is not controlled by lack of proper instrumentation (Orsat analyser)
- Kiln exit temperatures are adequate.
- Temperatures at cold end of condensers are too high and probably at summer time will be even higher.
- Depression at cold and hot end of condensers are too high.
- Granulometry of feed ore was set at 40 mm. size.

- Traces of mercury were apparent in mud deposits of was te water.

During the development of the tests, we found several equipment difficulties of very difficult correction, given the design characteristics of the equipment of the plant.

3. Results of the tests

Tests were started by increasing the number of samples and corresponding analysis of feed ore and calcines. Taking into account the defective conditions of the laboratory equipment and the limited personnel available it was not possible to perform more than four ore and one calcines analysis per shift.

Obtained results were the following:

3.1. Feed ore

Table IV-4, shows the amounts of ore fed, ave rage daily mercury grades, mercury content in the ore and mercury produced together with the calculated metallurgical efficiencies during the running of the teste.

The amounts of ore fed to the kilns have been calculated using the number of wagons of calcines obtained. During the days of the test a total of 2.126 wagons were obtained, which assuming an average unit weight of 530 kg of calcines per wagon gives a total of 1.126 tons of calcines, corresponding to some 1.208 tons of dry feed ore, using

Haliköy's rule of thumb calculation procedure (1).

Size of ore fed to the kilns was set at 40/45 mm during the first days of the tests, increasing the ore size to 80/85 mm the 8 th of May up to the end of the tests.

Although the results of the bigger size ore cannot be conclusive, taking into account the equipment short comings and the frequent interruption of the tests for several reasons, they are very promising about the possibility of increasing kilns capacities, provided the necessary fuel input is given to the burners in order to obtain proper calcines.

The total number of calcines' wagons obtained during the feeding period with 80/85 mm size ore, amounted to 459, in 7 shifts. Such rate would increase the capacity to 167 tons/day for both kilns, which compared with the 142,5 tons day of 1971 average, means an increase of -- 24,5 tons/day, or, in other words, a 17,6% increase in capacity.

3.2. Calcines

Calcines show normally at Haliköy only traces of mercury. During the tests and even with the bigger size ore the amount of mercury in the calcines showed no increase whatsoever.

3.3. Temperatures and depressions

Temperatures at the cold end of condenser reached

(1) Divider 0.91 is used as a rule at Haliköy to convert weight of calcines into weight of dry feed ore.

average values from 38° C to 40° C

It was not possible to decrease temperatures - with a proper regulation of depressions in the condenser, as no regulating valve nor speed variator was available at the suction fan. On the other hand, the bad state of the condenser pipes badly corroded and the abundant air inlets through the scrubbers made unsuccessful any intent of depression regulation.

As far as it concerns the suitable ranges for temperatures and depressions, they should be between the following values:

a) <u>Temperatures</u>

At the exist of the kilns, they must range from 300 to 320° C

At the exit of cyclons, from 280 to 300° C in order to achieve that the dust collected in the cyclon contains always a minimal amount of mercury

At the cold end of condensers, they must range from - 20 to 30° C.

b) <u>Depressions</u>

- At the cold end of condensers, 55-60 mm w g
- At the kiln's exit, 2-3 mm. w g

3.4. Excess air

At the beginning of the tests readings of the Orsat analyser showed great excess air at kiln's exit, (5% CO_2, readings were frequent) Corrective measures limited by the gravity feeding system of fuel to the burners, and some air inlet though the ore feeder, gave better results, shown at Table IV-5 appended.

Nevertheless, at condensers' exit amounts of excess air were extremely high All efforts to improve such conditions were fruitless by the already mentioned circumstance of the abundant leakages of condensers' pipes caused by corrosion Heat values for CO_2 have never been higher than 5% during the tests

- 5 Cyclone dust

Mercury content in cyclone dust has been very moderate during the tests Analysis performed have shown an average mercury content of 0 07%, an amount which may be qualified as satisfactorily low

3 6 Waste water

Visual observation of residual muds of waste water at the beginning of the test indicated an apparent mercury content

Two tests have been performed in order to measure the amounts of water sent daily to waste and its mercury content

To this end flow gaging and sampling were made at two different periods

a) Before water became muddy by the washing of the condenser

b) During the period of muddy water effluence by effect of condenser washing

Flow gaging was performed every 15 minutes - using a 40 l. drum and samples for mercury analysis were taken every 5 minutes.

Tables IV-6 and IV-7, show the recorded data - from both tests during period b) of the same. During period a) an average filling time of the drum of 55 seconds was re corded.

During period b) of the first test 185 grams of dust were collected in the samples with a mercury content of 3.80%. For the second test corresponding figures were 228 grams and 2.8% Hg. respectively.

For period a) 7.7 grams of dust for every 4.75 l of water and 0.8% Hg. were measured.

Therefore, the total daily mercury losses were:

1st. Test

Period a):	0.69 kg	of mercury/day
Period b):	4.76 kg	" " "
Total:	5.45 kg.	" "

2nd. Test

Period a):	0.69 kg.	of mercury/day
Period b):	6.18 kg.	" " "
Total:	6.87 kg.	" " "

Therefore, yearly losses according to the above figures would be:

1st. test:	5.45 x 365 = 1,989.25 kg/year
2nd. test:	6.87 x 365 = 2,507.55 kg/year

Mercury losses by this concept may be, then, estimated from 2 tons to 2.5 tons/year, corresponding to 2% and 2.5%, respectively, of the yearly average mercury production of Haliköy for 1969, 1970 and 1971.

3.7. Stackline deposits

Four samples of stackline deposits have been taken and analyzed. Two of them were taken through the upper manhole located at 12 m. of the foot of the stack and the other two through the first manhole after the exhaust fan. In both cases samples from the bottom and walls of the stackline were taken.

The mercury analysis from such samples gave the following results:

Upper manhole .	Sample nº 1 (bottom):	57.46% Hg
	Sample nº 2 (wall):	33.93% Hg
Lower manhole	Sample nº 3 (bottom):	6.10% Hg
	Sample nº 4 (wall):	27.96% Hg

Such figures show clearly a very poor efficiency of the condensing system and might give total losses in stackline and stack near 12-15%.

3.8. Laboratory Analysis

Mercury analysis is performed at Haliköy by the calorimetric method, with ferric sulphate as an indicator. The method is used for checking feed ore samples and also calcines samples.

By this method, during the running of the tests 137 samples of feed ore have been analysed, together with calcines samples (one per shift) and other miscellaneous analysis (cyclone dust, stackline deposits, etc).

4. Materials balance and operation efficiencies

Table IV-4, shows the general materials balance for ore and recovered mercury during the tests.

Calcines have been omitted, taking into account that they only showed traces of mercury.

For the same reasons stated in the case of Konya we shall use available data for 1971 in order to establish more representative materials balances and operating efficiencies.

4.1. Materials balance

Using 1971 data we have:

Treated ore (wet)	48,921 tons (8% H_2O)
Treated ore (dry)	45,476 "
Calcines	41,383 "
Average grade of ore	0.254%
Average mercury content of calcines.	Traces
Mercury produced	100,240 tons

During 1971, no stupps have been recirculated to the kilns, and all stupps produced were treated in a fuel-oil fired retort.

4.2. Mercury recovery efficiencies

4.2.1. Metallurgical efficiency

During 1971, this efficiency was:

$$\frac{\text{mercury (obtained)}}{\text{mercury (content)}} = \frac{100,240 \text{ tons}}{45,476 \times 0.254} = \frac{100,240}{115,509} = 0.867$$

4.2.2. Condensing efficiency

As no stupps were fed back to the kilns, and mercury losses in calcines may be estimated as negligible, condensing efficiency coincides with metallurgical or global efficiency, therefore its value was 0.867 or 86.7% during 1971.

Condensing efficiency is substantially poorer than in Konya, by the reasons already stated about condensing temperatures and air intake into the condenser ducts.

4.3. Operating efficiency of the kilns

For 1971, the operating efficiency, i.e.:

$$\frac{\text{annual working hours}}{24 \times 365}$$

reached the value: (see Annex G, Stoppages)

$$\text{Unit I} = \frac{7,987}{8,760} = 0.91$$

$$\text{Unit II} = \frac{5,492}{8,760} = 0.63$$

During the tests, working time for both kilns totallised 218 hours. Therefore the average operating efficiency of the plant was:

$$\frac{218 \text{ hrs}}{234 \text{ hrs}} = 0.93,$$

as 16 hours of stoppages were recorded during the tests by crusher breakdown and miscellaneous causes.

4.4. Capacity efficiency

This value, given by the ratio:

$$\frac{\text{Actual capacity (tons)}}{\text{Theoretical capacity (tons)}}$$

reached a global value for both kilns in 1971 of

$$\frac{48,921 \text{ tons } (1)}{2 \times 365 \times 100 \text{ tons}} = \frac{48,921}{73,000} = 0.67$$

During the test, and for an average humidity of the ore of 7.4%, the corresponding figure has been:

$$\frac{1,305 \text{ tons } (1)}{2 \times \frac{100}{24} \times 234 \text{ tons}} = \frac{1,305}{1,950} = 0.67$$

5. Specific recommendations

The problems found and analysed during our stay at Halikőy have allowed to establish the specific recommendations included in Section II of this Report.

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(1) Wet ore

TABLE IV-1

CONTROL OF ORE FED AND MERCURY PRODUCED DURING KONYA TESTS

Day	% Hg	Ore, (tons)	Hg, content, (Kg)	Hg, flasked, (Kg)	Effic. (%)	Remarks
20-IV (1)	0,262	52,5	137,55			Stupps are not fed
21-IV	0,282	137,5	387,75			
		190,0	525,30	345,0	65,6	
22-IV	0,319	141,50	451,38			Stuppsare not fed
23-IV	0,218	123,90	270,10			
		265,40	721,48	241,5	33,47	
24-IV	0,245	132,30				Stuppsare fed
25-IV	0,142	155,06				
		241,50	479,17	724,5	151,20	
26-IV	0,208	135,40	281,60			
27-IV	0,189	120,70	228,12			Stuppsare fed
28-IV (2)	0,196	69,30	135,80			
		325,40	645,52	462,3	71,6	
Total	0,230	1.022,70	2.351,65	1.773,3	75,4	

(1) Two and half shifts only
(2) Two shifts only

TABLE IV-2

ORE AND CALCINES CONTROL DURING KONYA TESTS

Day	Ore			Calcines	
	Tons	% Hg	Hg, content (Kg)	% Hg	Hg, losses (Kg)
20	52,5	0,262	137,55	0,015	7,87
21	137,5	0,282	387,75	0,009	12,37
22	141,5	0,319	451,38	0,004	5,66
23	123,9	0,218	270,10	0,008	9,91
24	132,3	0,230	324,10	0,012	15,87
25	109,2	0,142	155,06	0,010	10,92
26	135,4	0,208	281,60	0,080	10,83
27	120,7	0,189	228,12	0,004	4,83
28	169,3	0,196	135,80	0,002	1,39
Total	1.022,7	0,230	2.351,65	0,008	79,65

TABLE IV.3.

Orsat analyser readings at Konya Plant

% CO_2	O_2 %	% CO_2	O_2 %
8	12	6	6
7	3.5	9	3
9	5	8	4.4
8	4	8	6
8	3	8	4
7.2	5.8	6.7	5.3
8.5	4	8.5	2.5
8.5	2.5	8	2
8.5	2.5	9	2
9	2.5	9/9.5	3/2.5
9	2.5	8	4
8.5	3.5	9.5	4
9	2.8	6	4
7	4.5	7	5
7	3	6	3
6	3	5	1
8	2	7	2.5
10	3	9.6	2
9	3	9.5	3
8	2	6	2
6.5	1	7.5	2.5
8	2.5	8	2
5	3	8	2
8	1.5	5	2.5
7	2	7	2.4
9	2		

TABLE IV-4

CONTROL OF ORE FED AND MERCURY PRODUCED DURING HALIKOY TESTS

Day	% Hg	Ore, tons	Hg, content (Kg)	Hg, flasked (Kg)	Effic. (%)	Remarks
2-V (1)	0,498	73	363	218	60,0	
3-V	0,265	113	299	460	153,8	
4-V	0,309	118	364	238	65,4	
5-V	0,279	108	301	133	44,2	
6-V	0,310	105	325	148	45,5	Stupps were fed during all the time of the tests.
7-V	0,284	124	352	245	69,6	
8-V	0,265	114	302	482	159,6	
9-V	0,246	120	295	231	78,3	
10-V	0,180	140	252	160	63,5	
11-V	0,180	153	275	170	61,8	
12-V (2)	0,221	40	88	85	96,6	
Total	0,266	1.208	3.216	2.570	79,0	

(1) Two shifts only
(2) One shift only

TABLE IV-5.

Orsat analyzer readings at Haliköy Plant

Kiln I		Kiln II	
% CO_2	% O_2	% CO_2	% O_2
7	1	12	2
6	3	12	2
6.5	-	11	3.2
8	2	11	3.2
10	3	10.3	3
10	2	11	3
9	1.5	10	3.5
9	2	10	3.2
8.8	1.4	10	3
8	1.2	11	2

TABLE IV-6

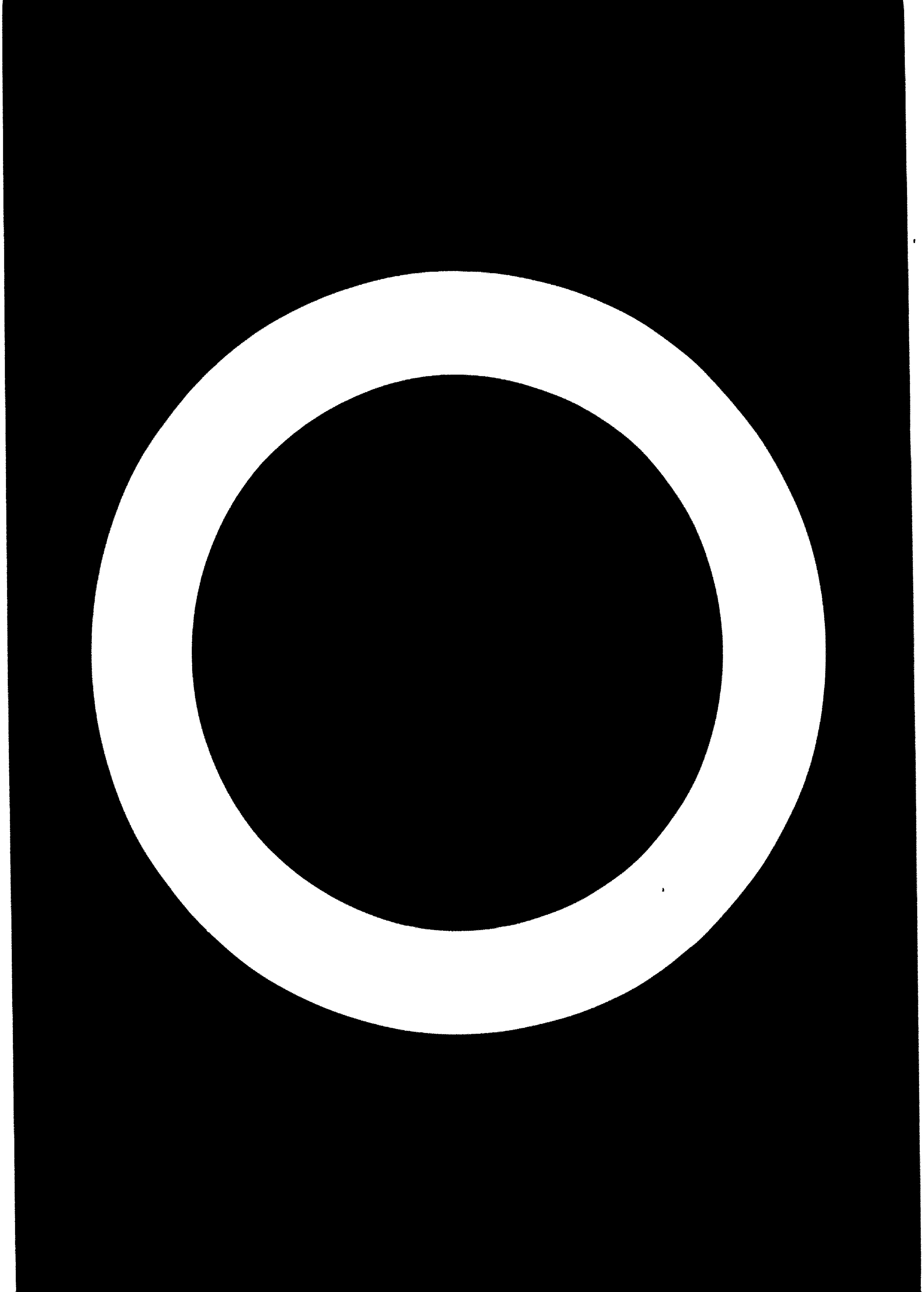
FLOW MEASUREMENTS OF WASTE WATER AT HALIKOY PLANT (4 MAY 1972)

Hour	Filling time of 40 l. drum (seconds)	Remarks
7 h	14	Water begins to become muddy
7 h 15 m	15	Water very dirty
7 h 30 m	13	"
7 h 45 m	12	"
8 h	15	"
8 h 15 m	14	"
8 h 30 m	14	"
8 h 45 m	13	"
9 h	17	"
9 h 15 m	12	"
9 h 30 m	24	"
9 h 45 m	22	"
10 h	30	"
10 h 15 m	35	"
10 h 30 m	30	Water almost clean
10 h 45 m	55	Water clean. End of test
Duration of test: 3 h 45 m	Average filling time: 21 seconds	Total flow during the 3 hours and 45 minutes: 25.714 liters

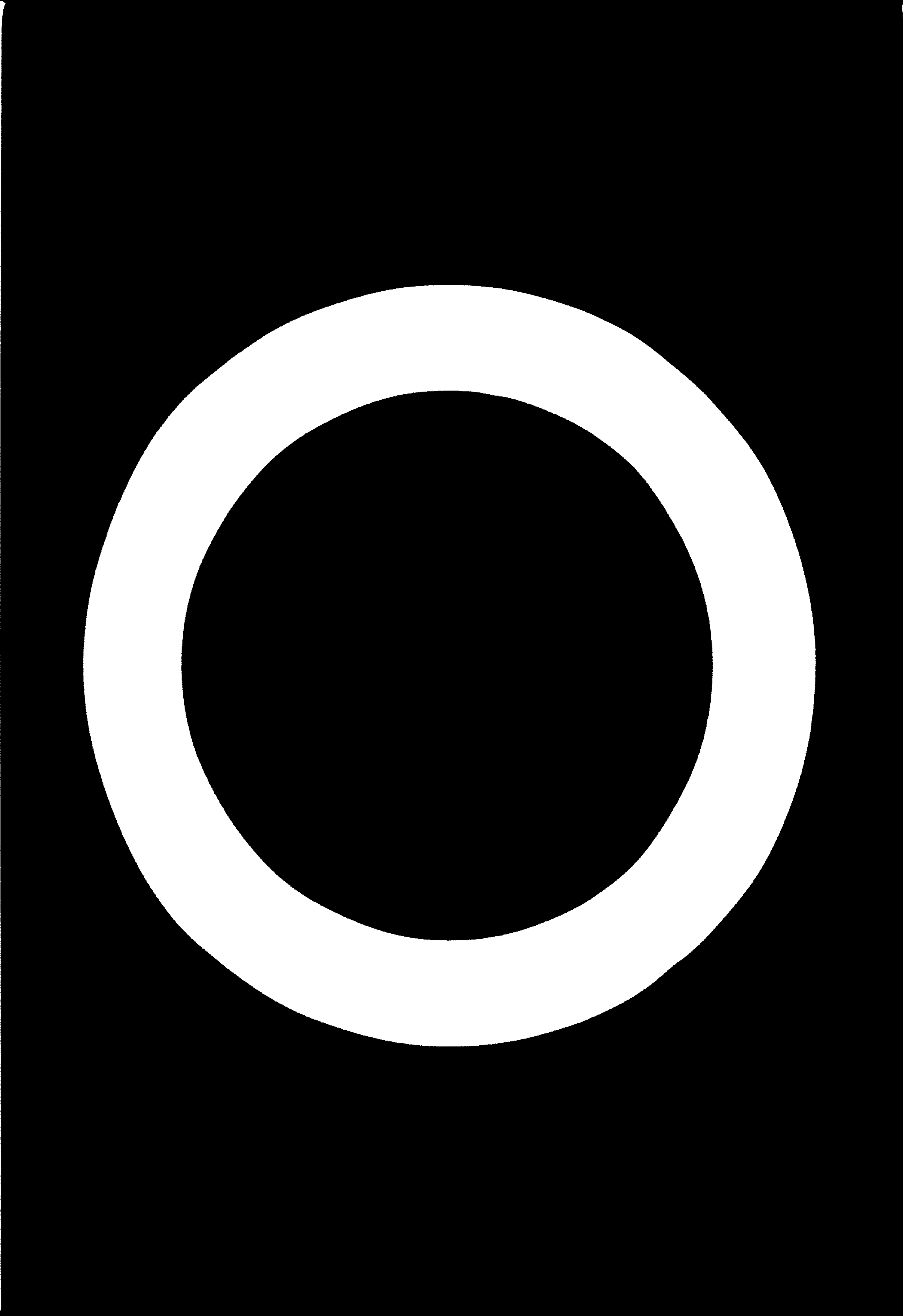
TABLE IV-7

FLOW MEASUREMENTS OF WASTE WATER AT HALIKOY PLANT (10 MAY 1972)

Hour	Filling time of 40 l. drum (seconds)	Remarks
7 h 30 m	22	Water begins to become muddy
7 h 45 m	18	Water very dirty
8 h	17	"
8 h 15 m	30	"
8 h 30 m	32	"
8 h 45 m	18	"
9 h	22	"
9 h 15 m	10	"
9 h 30 m		"
9 h 45 m	15	"
10 h	35	"
10 h 15 m	17	"
10 h 30 m	12	Water almost clean
10 h 45 m	15	Water clean. End of test
Duration of test: 3 h 15 m	Average filling time: 20,23 seconds	Total flow during the 3 hours end 15 minutes: 23.134 liters



V PERSONNEL TRAINING IN MERCURY EXTRACTION



1. <u>General</u>

During our stay at the Konya and Haliköy plants, patterns of correct plant operation have been given to counterpart personnel in mercury extraction.

In order to get maximum effectiveness of such training and taking profit that both Superintendents of Konya and Haliköy plants have a good knowledge of English, training of shift foreman has been made through them, especially on control procedures, giving on the spot advice to improve all difficulties that were encountered during the development of the tests.

Besides that, daily meetings have been held with plant superintendents, with the frequent assistance of plant Management, to discuss incidents of daily work and to prepare tests procedures for the following day.

Special emphasis has been placed on control procedures for the following matters:

a) <u>Treated ore</u>

Correct sampling procedures on feed ore, weighing of samples, laboratory analysis and recording of daily results have been established.

b) <u>Calcines</u>

Shift foreman and workers at the calcines bin have been instructed on proper calcines sampling, soaking samples in water after their extraction from the calcines wagons in order to avoid mercury distillation by hot temperature of the calcines, with the consequence of wrong analysis results.

c) CO_2 at kiln exit gases

Plant superintendents have been instructed in the correct use of the Orsat analyser for CO_2 and O_2 control in exit gases. At the end of our stay they were performing current analysis on both gases with the above instrument.

d) Temperatures and pressures

Importance of proper control of both variables has been stressed, and also the need of taking quick and proper action in case of anomalous readings.

f) Gas circuit tightness

The detrimental effects of excess air in the circuit and its repercussion on mercury recovery have been discussed in detail with plant personnel.

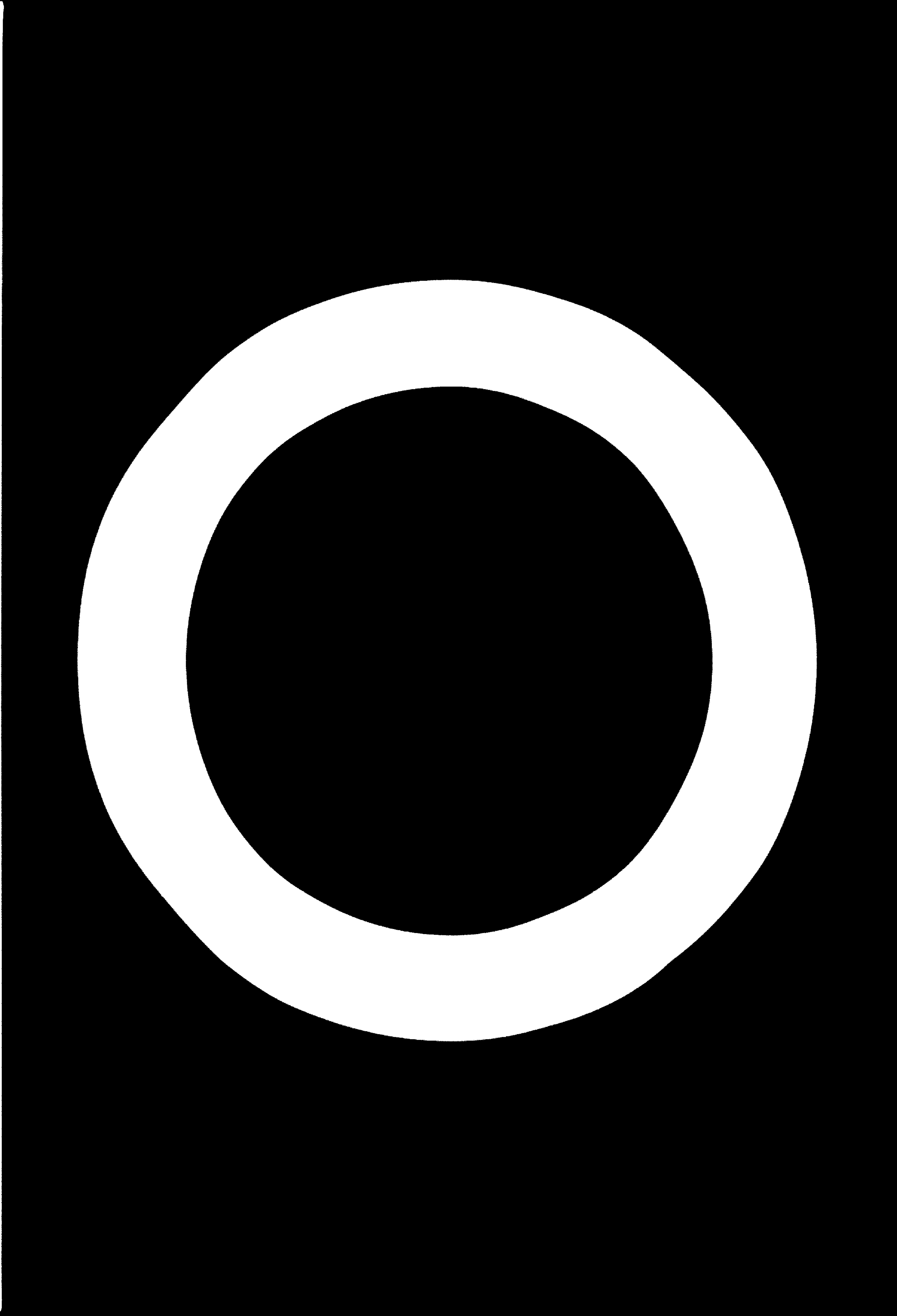
2. Recommendations on personnel training

We estimate that at the end of our stay in Konya and Haliköy the need for a careful control of plant operations is fully understood by Turkish counterparts, and that several of the provided patterns of correct plant operation will be incorporated to daily routine work, with the limitations presented by existing instrumentation on site.

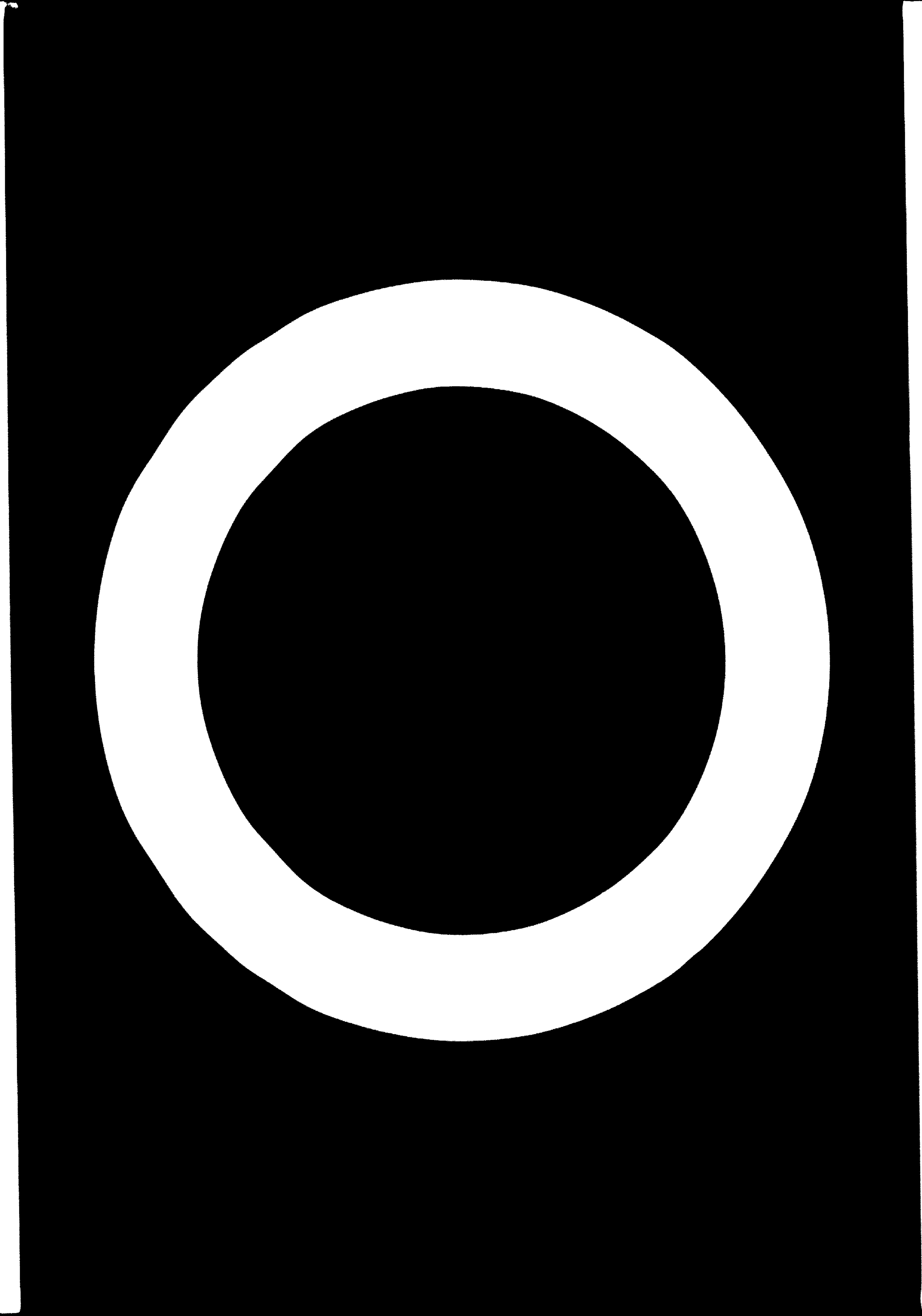
Nevertheless we have observed an almost total lack of information interchange about problems and operating procedures between the Konya and Haliköy plants, and therefore we would recommend periodical meetings of engineers of both plants in order to discuss current problems and make possible exchange of opinions to solve them.

Such meetings, must have in order to be fruitful a ve ry specific discussion subject, and should be held, in alternate sequence, in each one of the two plants.

Finally, we would like to recommend the convenience of training stages in foreign mercury plants for the engineers of Konya and Haliköy.

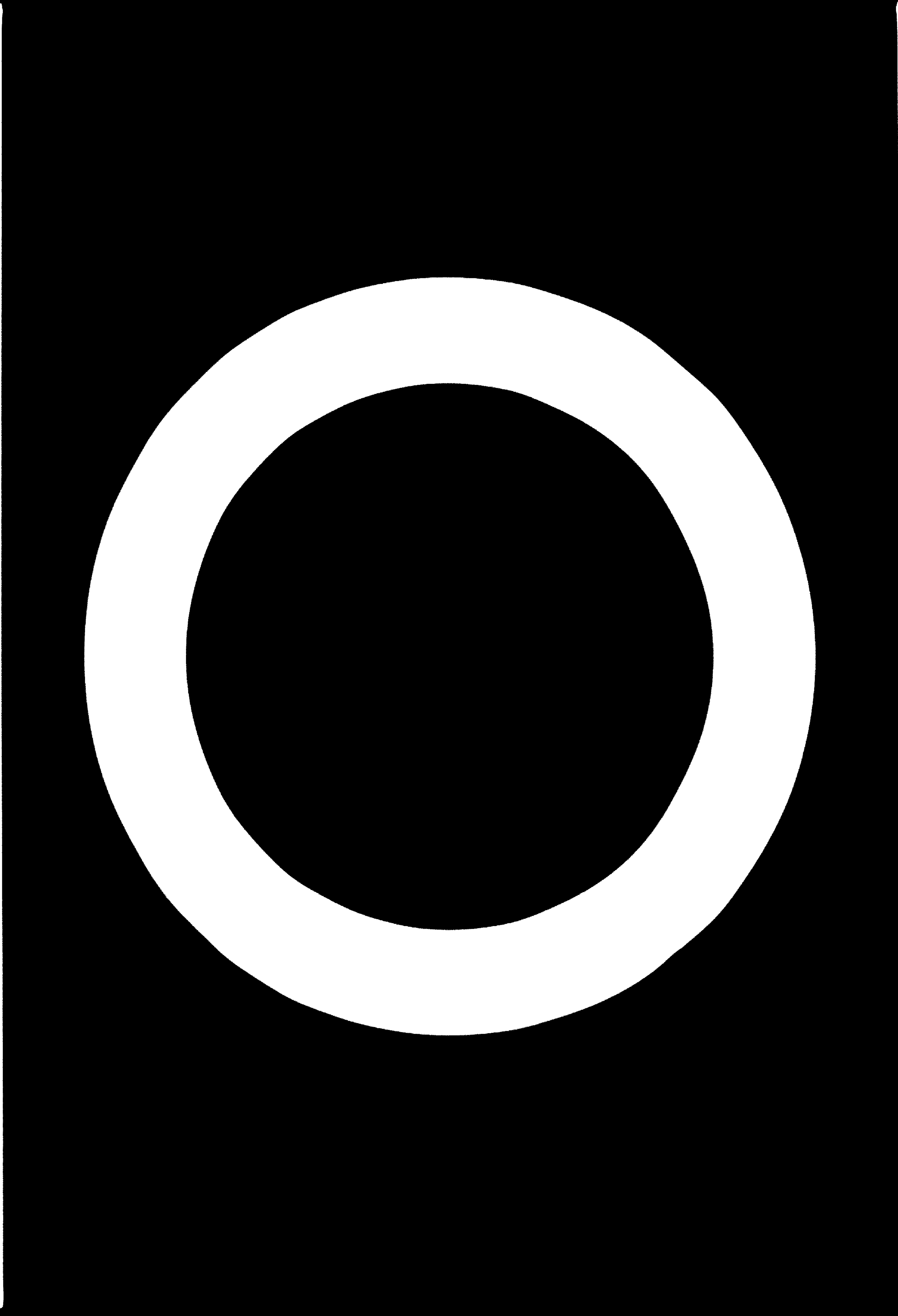


VI. ANNEXES



ANNEX - A

PRELIMINARY QUESTIONNAIRE



A. MINING

1. Technical aspects

1.1 General

a) Brief description of orebody indicating if mining exploitation is open pit or underground. Orebody thickness Dip Nature of country rock and of side walls.

b) Mineralogical aspects. Type of mineralization Useful and other related minerals. Mercury grade and percentage of other elements (especially arsenic, if any).

c) Type and characteristics of gangue

1.2. Investigations

a) Indicate if the orebody has been fully investigated and by which methods. Study of reserves.

b) Indicate if exploratory borings at short run have been carried on.

1.3. Mining

1.3.1. Ore breaking

a) Are explosives used?

b) Is drilling performed with hammers? What is the hammer's weight?

c) Indicate if extensive blasting is performed.

d) Are drilling rigs used?

e) Type and manufacturer of drilling bits.

1.3.2. <u>Loading and transportation</u>

a) Indicate which methode and type of equipment are used.

1.3.3. <u>Firing</u>

a) Type of explosives used.

b) Indicate if microdelays are used.

c) Is fragmentation favorable?

d) Consumption of explosives by ton of ore.

e) Indicate if the presently used explosives could be substituted by more economical ones.

1.3.4. <u>Mining dimensions</u>

a) Indicate the present dimensions of a working face and if they are estimated as optimum.

b) Indicate if side walls nature do not allow dimensional changes.

c) Indicate the present height between floors.

d) If open pit exploitation, indicate slopes, height of banks and track width.

1.3.5. <u>Direction of mining</u>

a) Indicate if mining is performed, in a mine section, by advance or by retreating systems.

b) Is it necessary to timber the head and base galleries?

1.4. <u>Preparatory workings</u>

a) Present dimensions.

b) Advance in m/month

c) Indicate if such workings are performed outside of the orebody.

1.5. <u>Timbering</u>

a) Indicate if timbering is performed, and type of timbering materials used both in the mining and in the preparatory workings.

1.6. <u>Results obtained with the present mining systems</u>

a) Indicate if present ore grade could be improved with a change in mining methods.

b) If such is the case, are ore and waste mixed with the result of a drop in grade?

c) With the present system, what percentage of ore is broken?

d) Indicate if a mass or a selective mining is presently preferred.

e) Indicate if the ore granulometry obtained is useable for the metallurgical plant.

f) Indicate which factors, in addition to explosives, dimensions and arrangement of shotholes, may have an influence on granulometry.

g) Indicate if the mine workings are concentrated in order to get a saturation in transport and extraction operations, and if the orebody allows such concentration.

1.7. <u>Miscellaneous</u>

Are power, compressed air, extraction, etc. capacities suitable?

1.8. <u>Safety</u>

a) Accident rates

b) Main causes

c) Accident prevention measures presently adopted

d) Does the exploitation have a repercussion on the surface?

2 Economical aspects

2.1 Tonnage presently mined

2.2 Present payroll distribution by categories

2.3 Indicate if specialised manpower is available

2.4 Present outputs per man hour in

- Ore breaking, tons man-hour
- Loading and transport, tons man hour
- Total mine interior (if applicable), tons man-hour
- Total mine, tons man-hour

2.5 Explosives consumption kg ton of ore

2.6 Power consumption kwh ton of ore

2.7 Compressed air consumption m^3 ton of ore

2.8 Bits consumption units ton of ore

2.9 Efficiency of existing loading, transport, etc. equipment

B METALLURGICAL PLANT

1 Technical aspects

1.1 Present flow-sheet of ore preparation installations, up to kiln feeding bin, showing capacities

1.2 Present flow-sheet of metallurgical plant and mercury recovery installations

1.3 Operating data

1.1.1 Ore characteristics

- Average ore grade
- Main ore constituents
- Granulometry
- Humidity

1.1.2 Equipment data

- Rotary kilns dimensions
- Cyclone type and efficiency. Loss of head
- Condensing system dimensions and materials used in its construction
- Exhaust fans characteristics (Flow, depression)
- Settling tank dimensions
- Characteristics and dimensions of stack line and stack
- Characteristics of soot machine

1.1.3 Operating parameters

a) Operating efficiency of rotary kilns

$$\frac{\text{annual working hours}}{24 \times 365}$$

b) Capacity efficiency

$$\frac{\text{Actual capacity}}{\text{Theoretical capacity}}$$

c) Metallurgical efficiency

$$\frac{\text{Hg (obtained)}}{\text{Hg (content)}}$$

- If figures a), b) and c) above are considered unsatisfactory, indicate possible causes
- Indicate which metallurgical variables are controlled, showing figures and location of the controls in the installation
- Hg content in cyclone dust and tons/day of dust

- Gas flow (Nm^3/hour) in relation to weight of treated ore (tons/hour).

- Hg in gas (grs/Nm^3) at stack inlet.

2. <u>Economical aspects</u>

- Tons/man-hour of calcined ore
- Fuel-oil (or other fuel) in kgs/ton of calcined ore
- Power, kwh/ton of calcined ore
- Percentage of Power, Fuel, Labor, Supplies, repairs and maintenance costs in total operating costs per ton of treated ore and per flask.
- Plant investments per ton of treated ore in U.S. dollars.

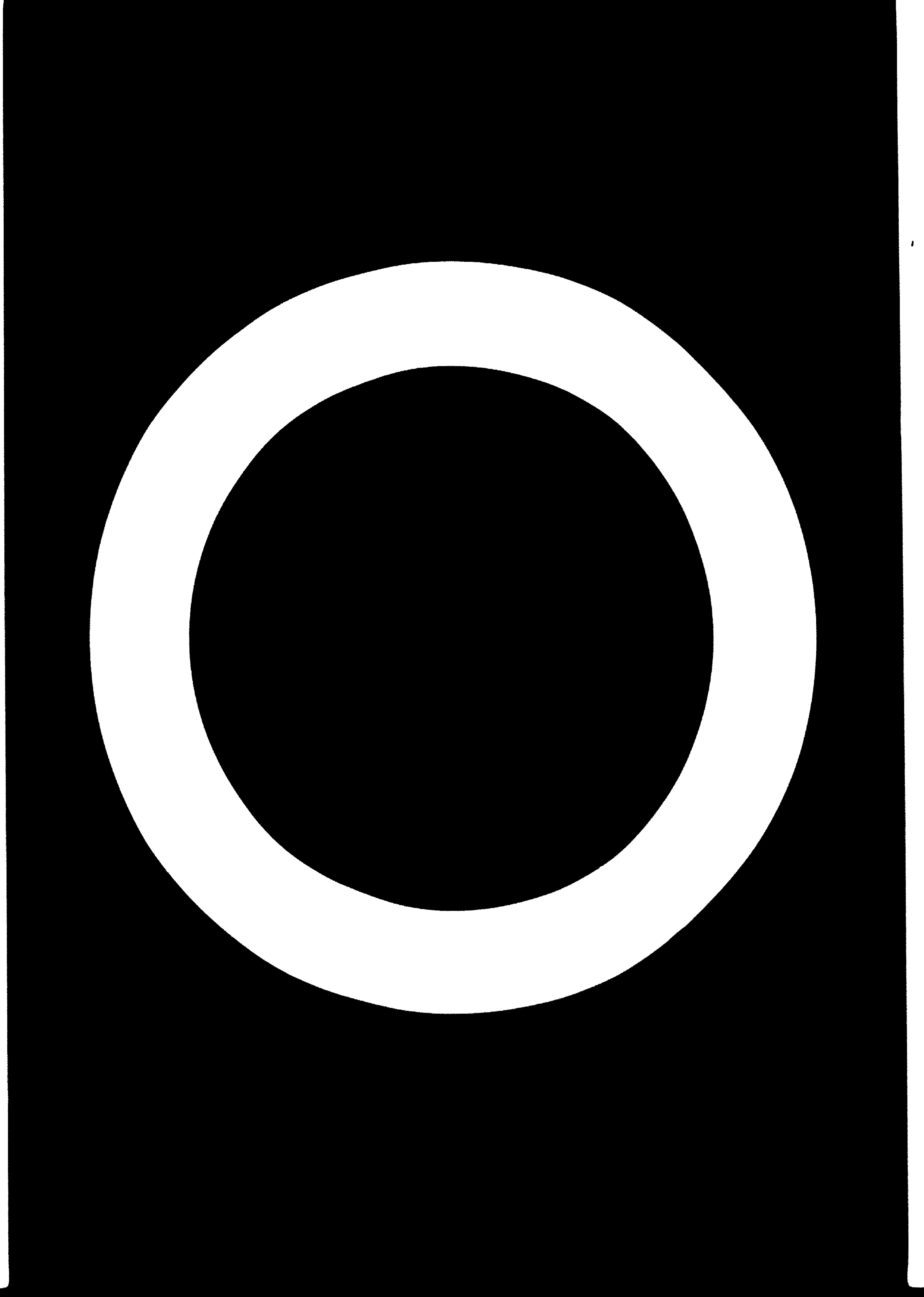
3. <u>General</u>

- Indicate what is the critical ore law taking in account present costs and sales prices.

- Indicate cost repercussion of Mining plant and Metallurgical plant on ton of treated ore and flask, respectively.

ANNEX - B

DATA CARD



Assistance for Non-Ferrous Metals,

MERCURY EXTRACTION (TURKEY)

DATA CARD

Name of Mine

Owner

Geographical Location

Coordinates

X =

Y =

Z =

Bibliography and References

Geological Description

Ore Body Geomorphological Description

Topographical and Climatical Description

INVESTIGATIONS

Chemical Analyses

Geological Studies

Geophysical Survey

Geochemical Survey

Investigation of Bore Holes Pits and Galleries

Pilot Plant

Investigations and Studies Proposed

[illegible]

Resources description

Verified Tonnage

Probable Tonnage

Possible Tonnage

TECHNICAL TABLE

Description of Exploitation Methods

Mining Extraction Tonnage

Concentration or Process Method Description

Tonnage, Concentrated or Processed

Description of Transport Methods to Metallurgical Factory or Sales Destination

MISCELLANEOUS

Personnel (Technicians and Workers)

Machinery

Water Supply

Power Supply

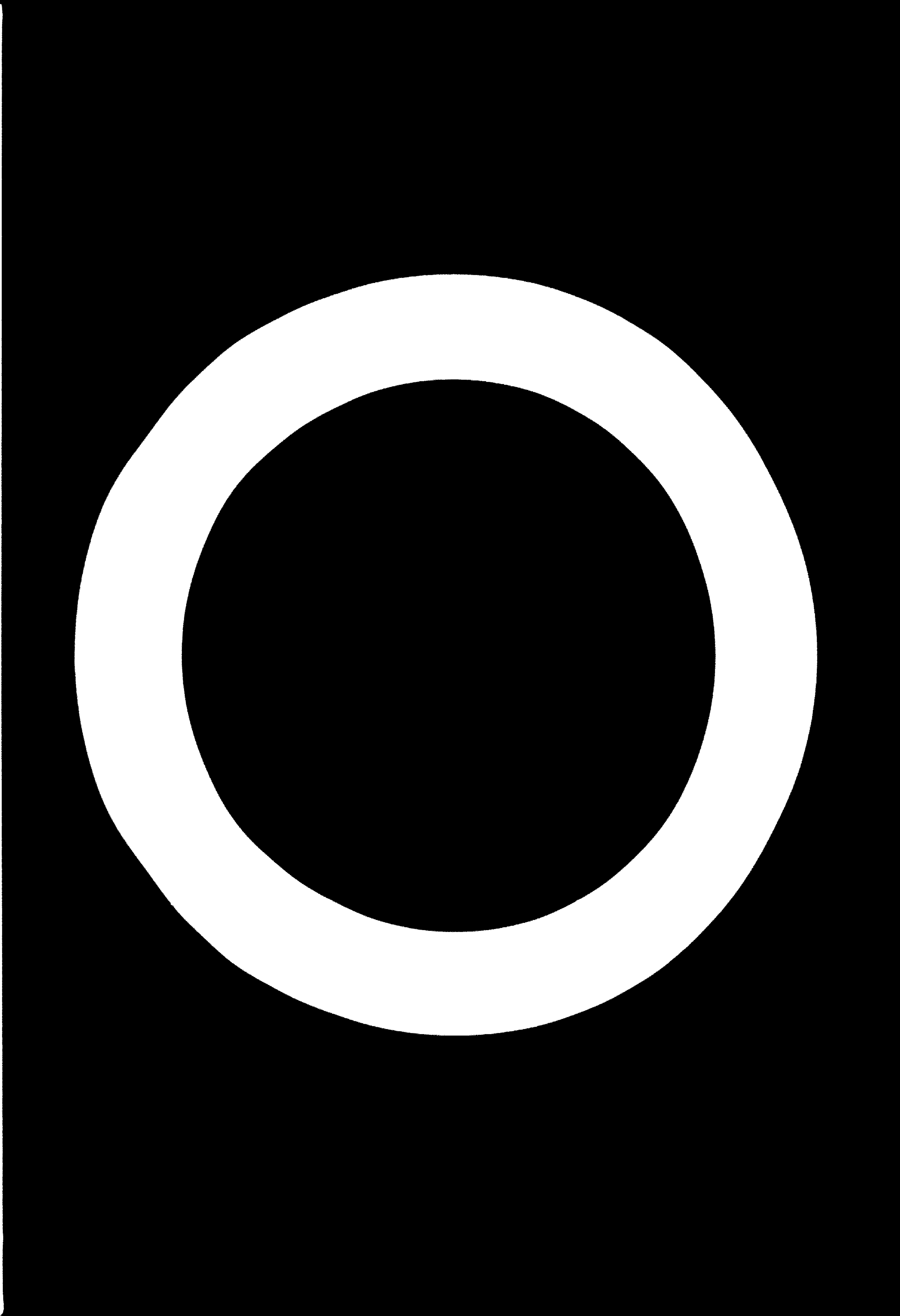
Other Supplies (Fuel, oil, explosives, wood....)

REMARKS

Client: UNIDO	Consulting Firm [illegible]	

ANNEX - C

EFFECT OF EXCESS AIR ON MERCURY CONDENSING



1. Introduction

In this Annex calculations for the roasting of a mercury ore are developed using fuel-oil in the combustion. Combustion process with variable percentages of excess air is analyzed together with the influence that such excess air has on mercury condensing.

For numerical calculations the average analysis of Konya ore fed to the plant during 1971 has been used. However the calculation process is general and may be quickly applied to any ore whose analysis is known.

2. Basic assumptions

2.1 Ore analysis

We will assume an analysis corresponding to the average ore of Konva during 1971 (See Section III, paragraph C.3.1.1.).

2.2. Fuel-oil analysis

An average analysis corresponding to the formula $(CH_2)_n$ is assumed. Therefore, C = 85.7%, H_2 = 14.3%. The available data for the fuel-oil consumed at the studied plants, show a sulphur content of 2.5 percent. Normally this element substitutes for the hydrogen, and so the following analysis for the fuel-oil has been estimated

C = 85.7%, H_2 = 11.8%, S = 2.5%

2.3 Fuel-oil consumption

35 kg/ton of ore

2.4 Calcines analysis

The calcines analysis for Konya during 1971 indicate that only 61 kg of carbonates from the 145.2 kg existing in one ton of ore are dissociated during roasting.

3. <u>Combustion process</u>

For the complete combustion of 14 grams of fuel-oil (stochiometric reaction) the exit gases, not taking into account for the moment, the mercury vapor, neither nitrogen and not combined oxigen from excess air, are:

a) Combustion gases from fuel-oil (CO_2, H_2O, SO_2)

b) Water vapor from humidity of the ore

c) SO_2 from roasting of sulphides

d) CO_2 from carbonates dissociation

Combustion air is assumed dry.

The weights of the above gases for 14 gr. of fuel are then:

• CO_2 from fuel-oil	44	gram.
• CO_2 from carbonates	11	"
• H_2O from fuel-oil	18	"
• H_2O from humidity	16	"
• SO_2 from sulphides	26	"
• SO_2 from fuel-oil	7	"
	122	gram

From the above weights the mols of gases correspon-ding to the combustion of 14 grams of fuel are then

$$CO_2 \quad \frac{44 + 11}{44} \quad 1.25$$

$$H_2O \quad \frac{18 + 16}{18} = 1.88$$

$$SO_2 : \frac{33}{64} = 0.51$$

(If amount of sulphur content in the fuel is compensated by the hydrogen, 0.17 mols less of water result).

Then the necessary mols of oxygen for the theoretical combustion, i.e. the amount corresponding to the chemical reactions are:

For C of fuel :	1 mol	
For H_2 " " :	0.33	(0.50-0.17) mol
For S, total :	0.51	
- Total	1.85 $\sim$ 2 mols.	

In practice, it is necessary, in order to ensure complete combustion, to use an excess of air. Let **x** be, the number of mols of air actually used.

The combustion reaction is, then:

$$(CH_2) + x\,(O_2 + 3.76\,N_2) + 0.25\,CO_2 + 0.71\,H_2O + 0.51\,S$$
$$1.25\,CO_2 + 1.71\,H_2O + 0.51\,SO_2 + 3.76 . x . N_2 + (x-2)O_2 \quad (1)$$

From (1) it is derived immediately that the percent in volume of CO_2, O_2 and H_2O in gases are, respectively

$$\%\,CO_2 = \frac{1.25 \cdot 100}{1.25 + 1.71 + 0.51 + 3.76 . x + (x-2)} \quad (2)$$

$$\%\,O_2 = \frac{(x-2) \cdot 100}{1.25 + 1.71 + 0.51 + 3.76 . x + (x-2)} \quad (3)$$

$$\%\,H_2O = \frac{1.71 \cdot 100}{1.25 + 1.71 + 0.51 + 3.76 . x + (x-2)} \quad (4)$$

Formulas (2) and (3) allow the immediate determination, in function of the percent excess air used $\left(\frac{B - A}{A}\right)$ 100 or the corresponding percentages of CO_2 and O_2 in exit gases.

6. Mercury condensation

6.1 Percent of mercury vapor in gases

For each ton of ore (i.e., for every 35 kg of fuel oil used), 0.2784% of SHg are fed to the kiln, or 2.784 kg.

Reaction

$$\underset{(232.6)}{SHg} + \underset{(32)}{O_2} \longrightarrow \underset{(200.6)}{Hg} + \underset{(64)}{SO_2} \qquad 6$$

shows that for every 35 kg of fuel the amount of mercury vapor produced is

$$\frac{2.784 \times 200.6}{232.6} = 2.4 \text{ kg}$$

and for the stoichiometric combustion (14 grams of fuel):

$$2.4 \times \frac{14}{35} = 0.96 \text{ grams}$$

which correspond in mole to

$$\frac{0.96}{200.6} = 0.0048 \text{ mole}$$

Therefore, the percentage in volume of mercury vapor in kiln exit gases will be

$$\% Hg = \frac{0.0048 \times 100}{[illegible]} = \frac{0.0048 \times 100}{[illegible]} \qquad 6$$

[illegible]

Then the volume of mercury vapor entering is

[illegible]

The volume of mercury vapor leaving is (from Dalton's law) equal to $\frac{P_1}{P_2} V$ where P_1 is the partial pressure of the mercury vapor and P_2 the partial pressure of the non-condensable gases after leaving the condenser.

The partial pressure of the mercury vapor P_1 and of the water vapor P_w is found by looking up the pressure corresponding to several temperatures in Tables [illegible] and [illegible] appended.

Supposing barometer at 760 mm. the partial pressure of the non-condensable gases is

$P_2 = 760 - P_1 - P_w$

Then the volume of mercury vapor leaving is

$$\frac{P_1}{760 - P_1 - P_w} V$$

The percentage of the mercury vapor remaining uncondensed is therefore

$$\frac{\dfrac{P_A}{760 - P_A - P_W}}{\dfrac{0.0000}{0.76 [illegible] - 0.24}} \quad 100$$

or,

$$\frac{P_A \; (0.76 [illegible] \; 0.24)}{(760 - P_A - P_W) \; 0.0000} \quad 100$$

Formula [illegible] allows the [illegible] of the [illegible] of mercury vapor remaining [illegible] for several [illegible] conditions, [illegible] for different [illegible] used to [illegible] and for different gas temperatures of gases leaving the [illegible]. In Table [illegible] appended [illegible] have been [illegible], together with different amounts of [illegible]

[illegible]

[illegible]

[illegible]

[illegible]

TABLE C-2

MAXIMUM VAPOR PRESSURE OF WATER (1)

(Or pressure of saturated steam)

Temperature °C	Pressure mm Hg	Temperature °C	Pressure mm Hg
[illegible]	[illegible]	[illegible]	[illegible]

(1) Below 0°C pressures are those of ice

Table [illegible]

[illegible]

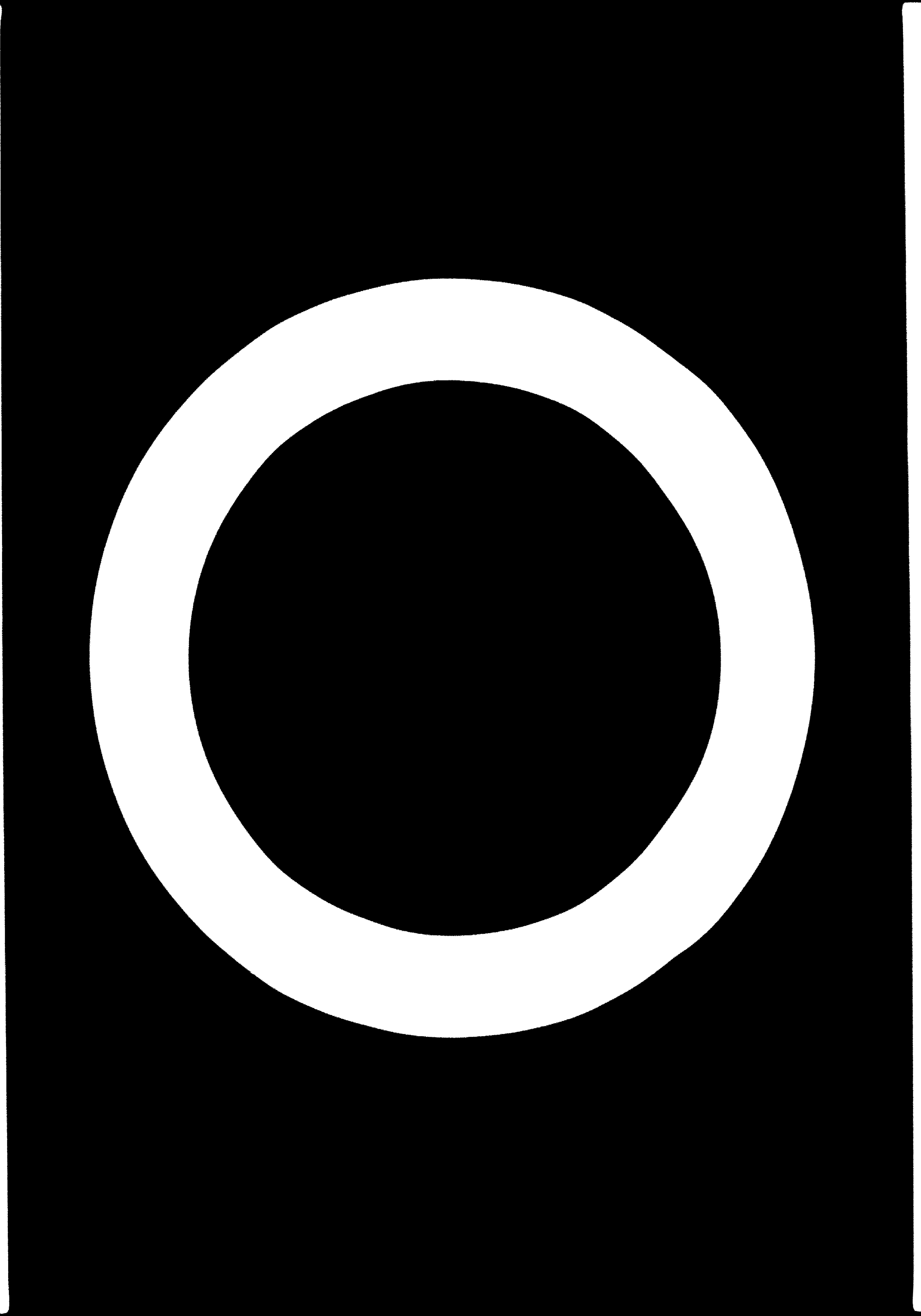
[illegible]	Percentage of [illegible] (1)	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[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]	[illegible]	[illegible]

Source: Own elaboration

Note (1): Percentage of [illegible]

ANNEX - D

CONTROL INSTRUMENTATION



General

In this annex general specifications are [illegible] for the type of instrumentation for the control of exit gases [illegible] temperatures and pressures.

2 Exit gases

Recent trends in exit gases analysis favor the use of oxygen analyzers. The main [illegible] of these [illegible] responds to the [illegible] of oxygen [illegible] in predetermined [illegible] and to maintain a predetermined low limit of oxygen in the exit gases of the process.

Oxygen measurement contributes to a good [illegible] of the process as present available oxygen analyzers have a good sensitivity, high precision, [illegible] and operating reliability.

The main elements forming an oxygen analyzer system are:

a) Oxygen measuring probe with [illegible] electrodes
b) Feeding line for reference air with [illegible]
c) Analogic current transmitter and amplifiers
d) Electronic recorder

Dimensions of electronic recorder and panel [illegible] are shown in appended brochure [illegible]

3 Temperatures

Temperature measurements are performed through a thermometric probe, and recording is made through electronic recorder transistorized.

Dimensions and specifications for both probe and electronic recorder are included in brochures TI D [illegible] and FCC [illegible] 102 R annexed.

- **Pressure and depression measurement**

Pressure system used at [illegible] and [illegible] is adequate. Evidently more elaborate systems could be used [illegible] the manometers presently installed for measuring low gauge pressures in places like [illegible] which [illegible] 0.2 [illegible] 1.2 [illegible]. This is particularly suitable for measuring low pressures and vacuums in furnaces, flues and combustion chambers.

The instrument consists of a deflection tube [illegible] two bells floating on a paraffin [illegible] seal.

However such type of instruments are rather expensive and its inclusion in the general instrumentation set is hardly advisable at this time as the present gauges used at [illegible] and [illegible] are sufficient to measure changes in pressure in the system caused by leakage or by plugging. With such pressure gauges properly [illegible] at frequent intervals, suitable steps may be taken to alleviate the trouble.

- **Panel mounting**

Drawing D-1 included shows the very simple arrangement of above described control instruments in a central panel.

Such panel includes instrumentation for both kilns (Units I and II).

- **Estimated costs**

The following cost figures are only orientative and are included only as a general guidance for the Turkish personnel.

Figures below correspond to instrument costs for each one of the plants of Konya and Halıbaşı

a) Oxygen analyzer and auxiliary equipment

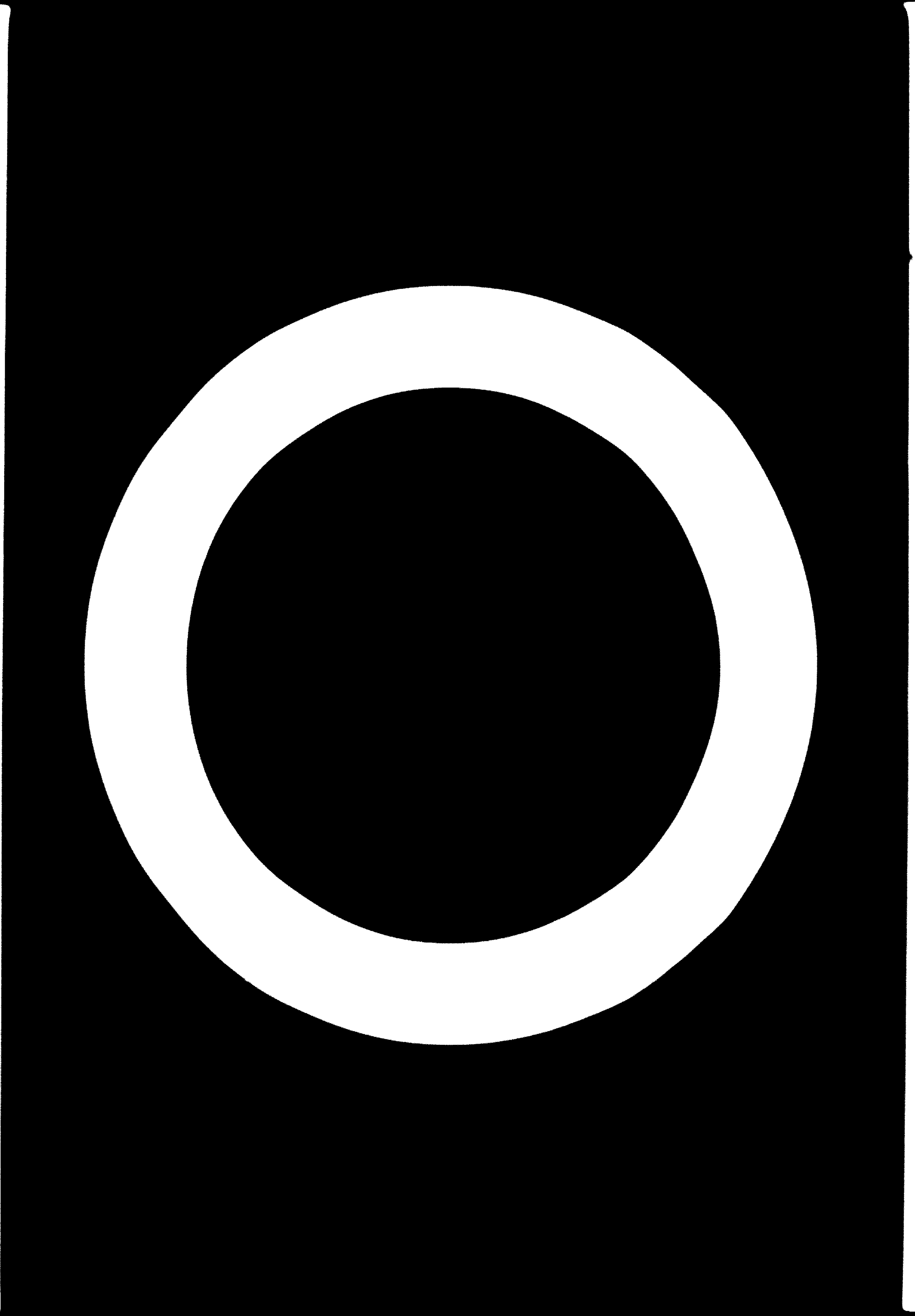
Oxygen sampling probes (2)	94 000 TL
Feeding tubes for reference air (2)	1 050
Analogic current transmitters (2)	18 360
Electronic recorder 2 pens (1)	12 220
Total	125 130 TL

b) Temperature measurement and recording

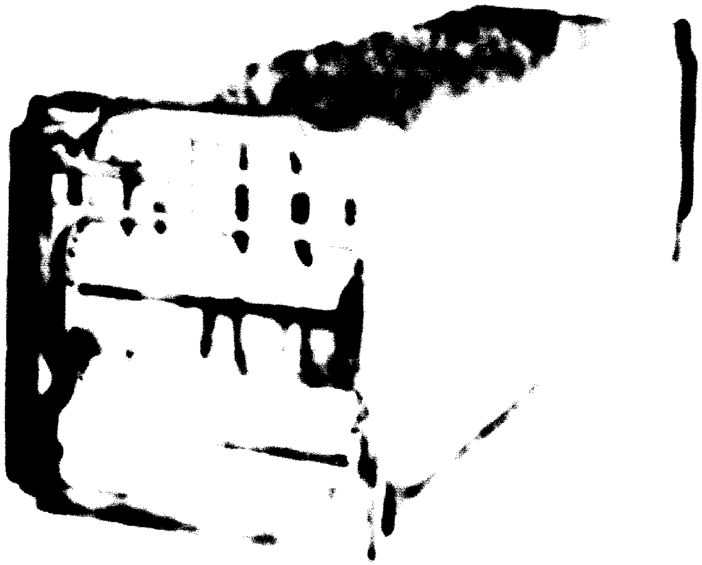
Thermometric probes (6)	9 672 TL
Electronic recorder (6 pens) (1)	26 000
Total	35 672 TL

9 Notes

Brochures included correspond to one of the several reliable manufacturers of control instruments available in the market. Therefore, inclusion in this report bears no relationship whatsoever with any specific preference or recommendation for the purchase of such instruments from any given supplier.



E-24 MINIATURE ELECTRONIC RECORDER



FUNCTION

[illegible]

B- 560

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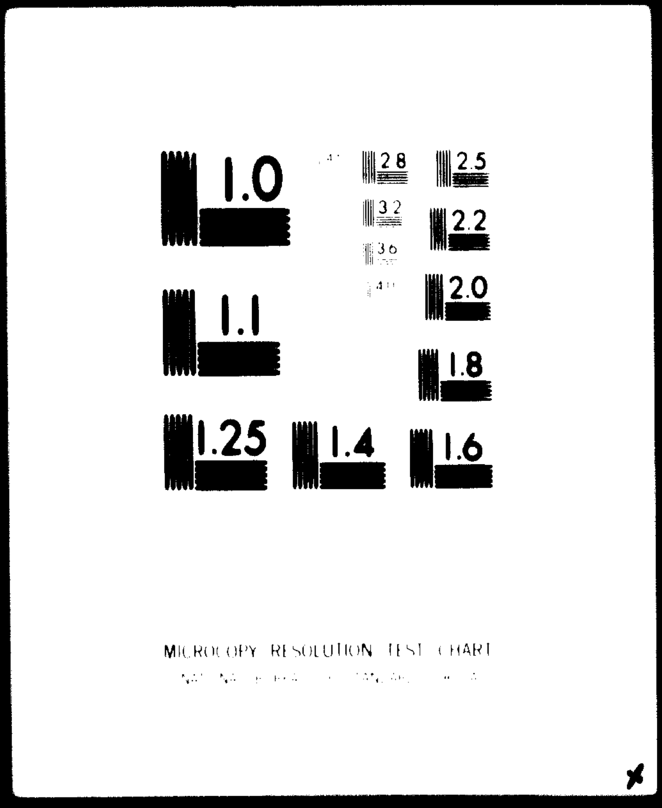

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D

CHARACTERISTICS

SPECIFICATION	WITH AMPLIFIER FOR HIGH-LEVEL INPUT	WITH AMPLIFIER FOR LOW-LEVEL INPUT
SPAN	e.m.f. : potentiometer adjustment: from 0.5 to 5 V with left-hand zero. from ±0.5 to ± 2.5 V with centre zero. current : by shunting R (Ω) / Scale (mA): 50 — 0 - 20 100 — 0 - 10 or 10 - 50 250 — 0 - 4 or 4 - 20 1000 — 1 - 5	e.m.f. : 2 ranges with potentiometer adjustment : ● 5 mV to 50 mV ● 50 mV to 0.5 V. Either range selected by changing two resistors. For resistance thermometers, span is adjustable by potentiometer from 10 to 110 Ω.
SETTING OF LOW END OF SCALE	Can be 1 V on 5 V range.	From 0 to 55 mV on both ranges (limited to half the span for narrow span radiation detectors). From 80 to 220 Ω for resistance thermometers.
ACCURACY	± 1 % of span.	
DEAD BAND	0.5 % of span.	
INPUT IMPEDANCE	At balance : 100 kΩ minimum Unbalanced : 100 MΩ minimum	At balance : 50 KΩ minimum Unbalanced : 100 MΩ minimum
SOURCE RESISTANCE	25 kΩ maximum.	5 kΩ maximum.
RESPONSE TIME (full scale)	Less than 5 sec.	
AMBIENT TEMPERATURE	0 to 45 °C.	
EFFECT OF TEMPERATURE CHANGE	0.03 % of span per °C.	0.05 % of span per °C.
MAXIMUM RELATIVE HUMIDITY	70 % Resistance to humidity is limited only by its effect on the chart paper.	
LINE VOLTAGE	110/127/220 V ± 10 %, 50 or 60 Hz. Set inside instrument.	
CONSUMPTION	1 pen : 12 VA. 2 pens : 18 VA. 3 pens : 25 VA. 4 pens : 31 VA.	
EFFECT OF POWER-LINE VARIATIONS	± 10 % voltage : 0.3 % of span 40 to 60 Hz frequency : 0.2 % of span.	± 10 % voltage : 0.2 % of span 40 to 60 Hz frequency : 0.2 % of span.
DIFFERENTIAL MODE REJECTION at 50 Hz 1 V r.m.s. : span	3 % of span with zero at left 4.5 % of span with centre zero.	3 % of span.
COMMON MODE REJECTION	With 1 V span and 1200 Ω source resistance. + Terminal: 0.1 % of span for 135 V d-c.; 1 % of span for 100 V r.m.s. 50 Hz – Terminal: 0.1 % of span for 135 V d-c.; 0.1 % of span for 100 V r.m.s. 50 Hz	With 5 mV span and 1000 Ω source resistance. + Terminal: 0.5 % of span for 30 V d-c.; 1 % of span for 180 V r.m.s. 50 Hz. – Terminal: 0.5 % of span for 30 V d-c.; 2 % of span for 180 V r.m.s. 50 Hz.

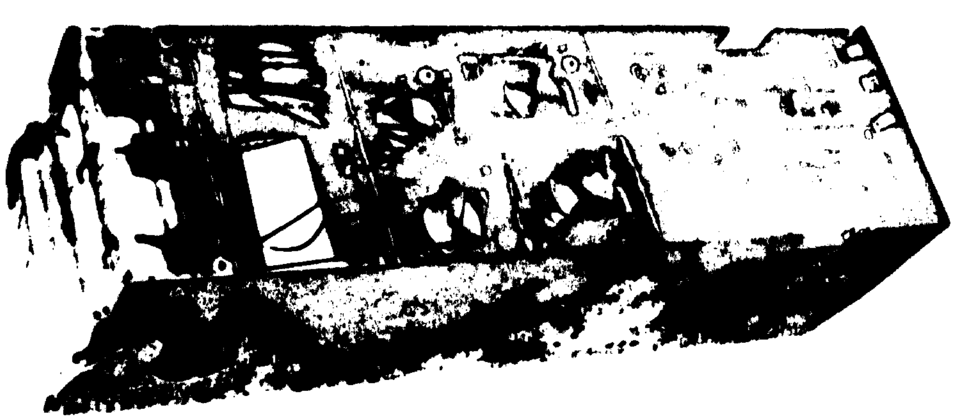
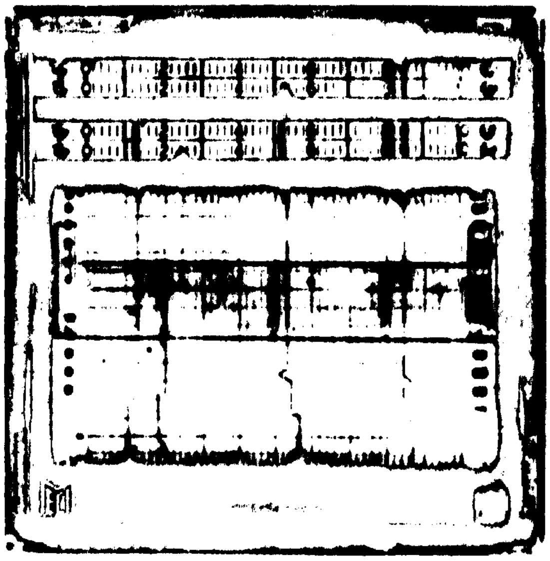


CHART DRIVE

Folding Chart



Reroll Chart

Chart width : 4" (100 mm).

a) Re-roll for 53 ft long chart ; one month at 3/4" (20 mm) per hour.

b) «48 hours» self-curling chart.

c) Folding chart with 40 mm (19/16") pleats. Folds down into plastic storage container.

Chart speeds :

- **Standard** : 20 mm (3/4") per hour single speed.
  20 and 600 mm (3/4" and 24") per hour two-speed.
- **Optional** : 10 - 60 - 120 - 200 - 360 - 600 - 1200 - 3600 and 7200 mm/hr to DIN 16 230 et 43 831.

On request, the recorder can be provided with 2 independant chart-drive motors for basic and high speed, selected either by a switch on the chassis or by remote control. Standard combinations : 20 - 120, 20 - 600, 20 - 1200 mm/hr.

DESCRIPTION

All components are mounted on a chassis which can slide out of a metal case. The instrument will operate when partially withdrawn for inspection or adjustment.

Fixed scales : one per pen (up to 4) removable.

Choice of markings :

- 0 to 10, linear on one side, square-root on reverse side ;
- − 10 to + 10 for centre zero ;
- Blank scale ;
- Special markings on request.

Connections :

At rear, on terminal board with cover ; all terminals are marked.

Mounting :

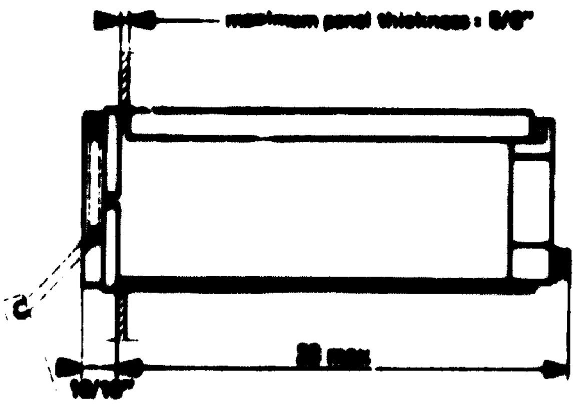
- Flush on board ;
- Close mounting possible.

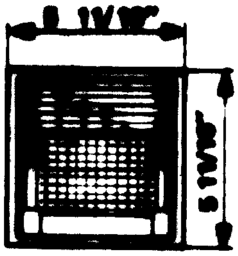
Colour : grey

Overall dimensions : 144 x 144 x 603 mm (5 11/16" x 5 11/16" x 24")

Weight : 20 lbs

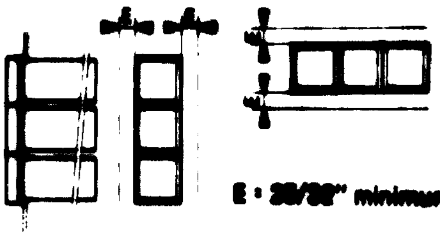
DIMENSIONS





CLOSE MOUNTING

VERTICAL HORIZONTAL



E : 25/32" minimum

Board Cut-out

Single	$5\ ^7/_{16}" \pm {}^1/_{32}" \times 5\ ^7/_{16}" \pm {}^1/_{32}"$
Close mounting of n recorders	$5\ ^7/_{16}"\ ^{+0}_{-1/32}" \times (n \times 5\ ^7/_{16}") - {}^{13}/_{32}"$

SUBSIDIARIES

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MARQUÈS DE MONTEAGUDO
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Tel 233-125

PORTUGAL : SORVAL RUA ARTILHARIA UM 104 4 G LISBOA

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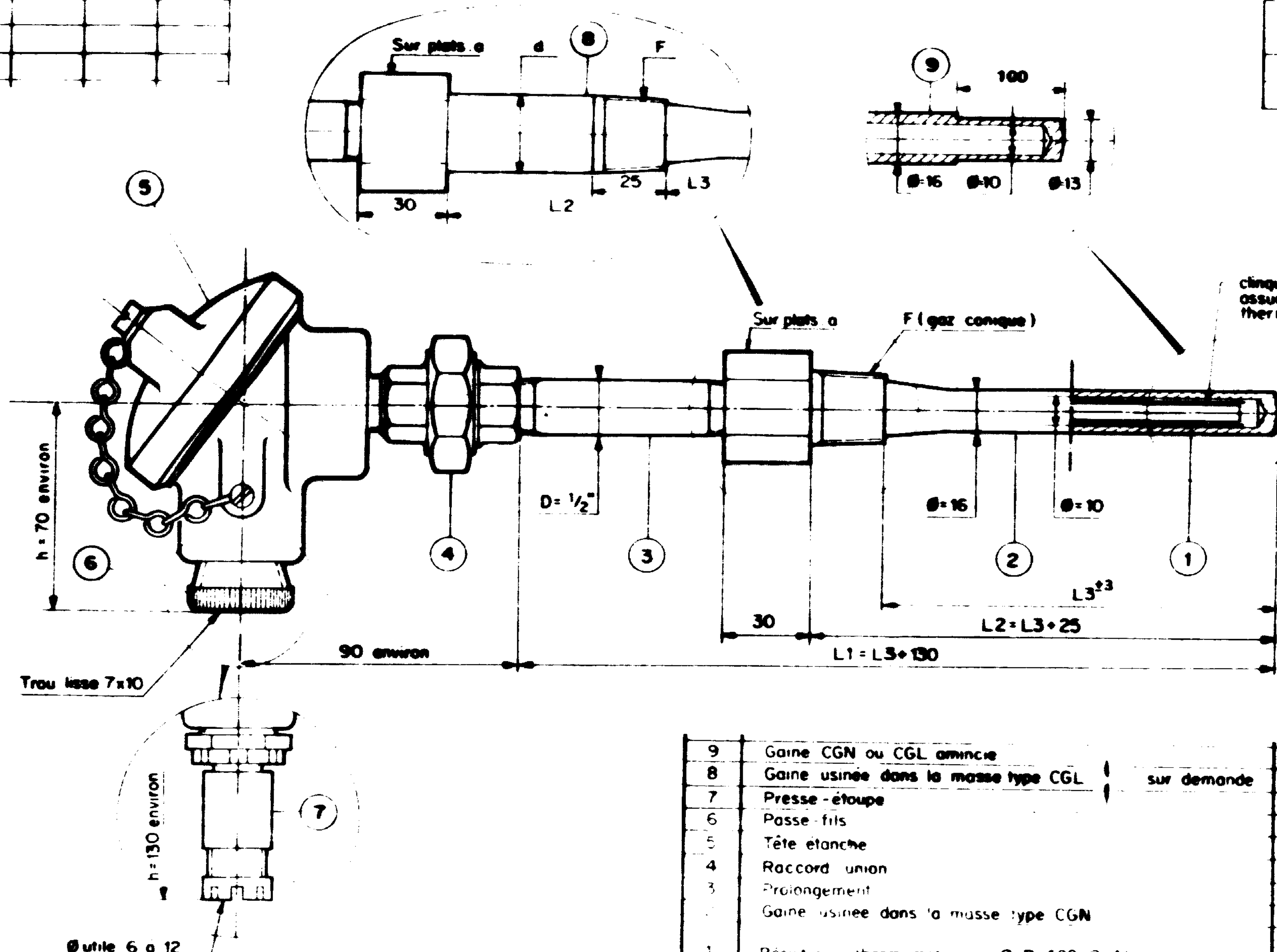
11-78

MECI C

Repères	L2	L3	F

F	a	d
1/2"	29	21
3/4"	35	27
1"	35	34



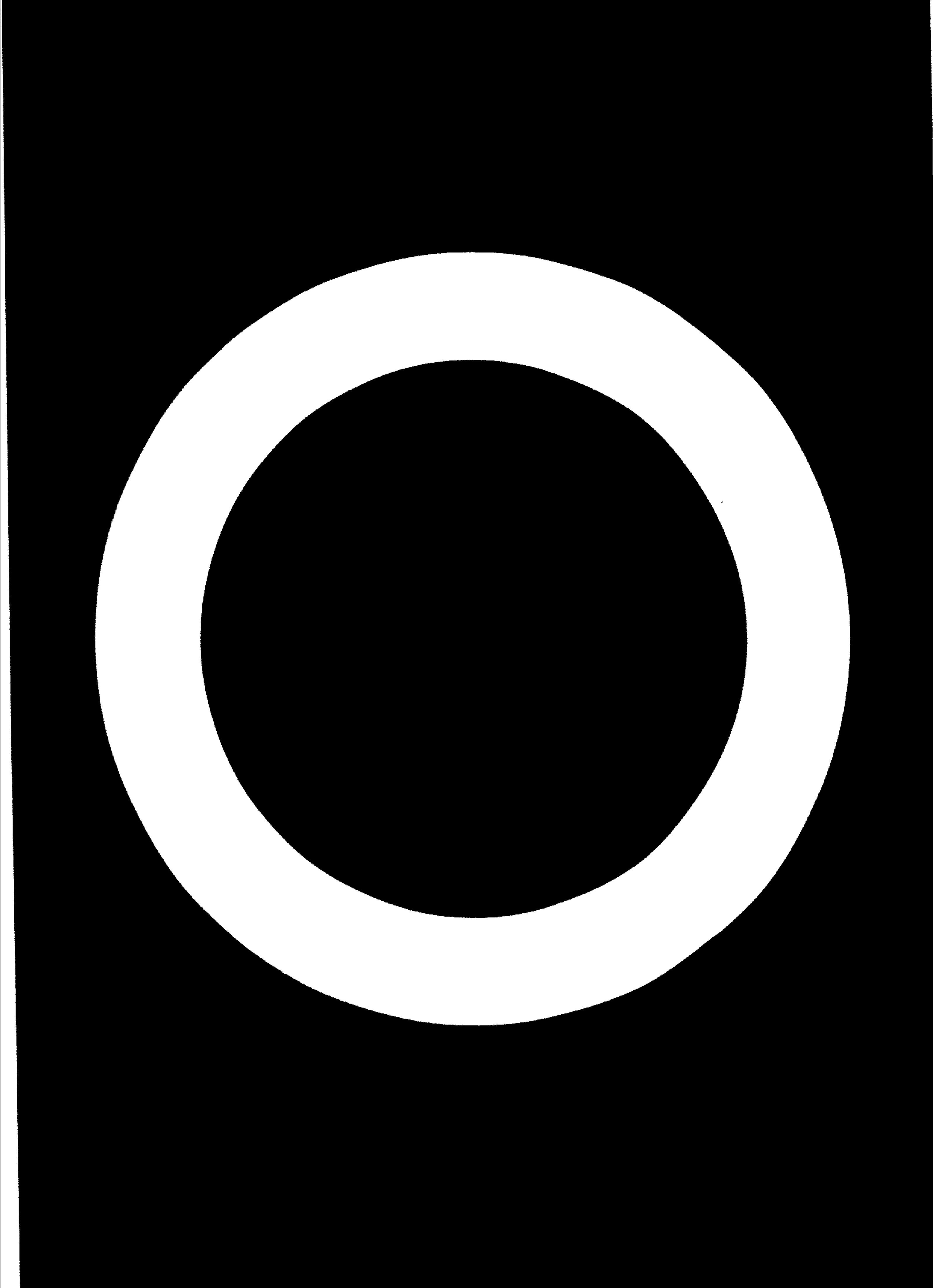
Rep	Désignation		Matière
9	Gaine CGN ou CGL amincie	sur demande	
8	Gaine usinée dans la masse type CGL		
7	Presse - étoupe		Alliage léger
6	Passe - fils		Bakelite
5	Tête étanche		Fonte
4	Raccord union		Fonte
3	Prolongement		
	Gaine usinée dans la masse type CGN		
1	Résistance thermométrique (C D 103 C A)		Platine 100 Ω à 0°C et 138,69 Ω à 100 °C

Mars 1965

S.A. MECI - 123 Bd de Grenelle - Paris XV - Tél. 306.90.00 Telex 25 754

CANNE THERMOMÉTRIQUE C D 6 C 4

PLAN TCD6C4



TRANSISTORISED ELECTRONIC RECORDER

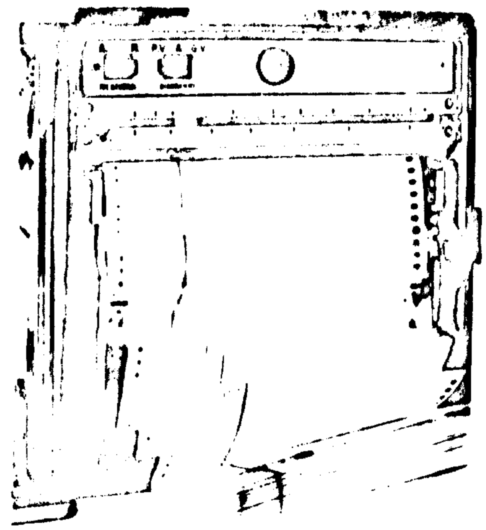
MINIPONT – TYPE D

MERCI

FCC - A - 102 - K

Recorder 10.70

The **MINIPONT**, which was the first miniature instrument to be put on the market, ten years ago, is already well known to industrial technicians and research workers. Today, it is in no way out-dated, in spite of the appearance of a very large number of small-format instruments ; the numerous improvements make keep it both up-to-date (an all-silicon **transistor version** is now available with intrinsic safety *) and polyvalent, offering a combination of qualities rarely encountered.



REDUCED DIMENSIONS

The reduced format (190 x 193 mm) and the availability of a two-block version reducing from 550 to 400 mm the panel-mounting depth required, make the **MINIPONT** ideally suited for equipping industrial control panels and consoles. The advantages of the MINIPONT's small dimensions are no less appreciated in the laboratory; workers have available a precise and readily handled measuring instrument, which can be used in portable test equipment and in experimental vehicles.

GREAT ADAPTABILITY

The **MINIPONT** performs :

- direct measurement of temperatures, by means of thermocouples, resistance thermometers or radiation pyrometers ;
- indirect measurement of any physical phenomenon which has first been transposed into a proportional electrical signal: flow, pressure, level, speed, gas concentration, pH, conductivity, etc....

Certain original features make possible the following adaptations :

- the measuring range may be altered, simply by replacing a set of connection strips.

The **MINIPONT** may also be equipped with two switch-operated measuring ranges.

- In the measurement of temperatures by thermocouple, the nature of the couple may be changed by altering a terminal bridging. The instruments are provided with reference junction compensation circuits, prewired, for the four standard types of thermocouple.
- The function of the measuring bridge may also be modified. It is very easy to change from a Wheatstone bridge to a potentiometer circuit.
- The multi-point instrument provides for the use of 6 points of measurement; the double range models may be equipped with an automatic range selection device.
- The chart (useful width : 120 mm) travels at one of three speeds, selected simply by changing the position of a pinion. For greater flexibility, we can, on request, provide an electrical gear-change device which multiplies the basic speed by 10 (or 60).

On request, the **MINIPONT** instrument may be equipped with a folding strip chart rack interchangeable with the normal roll strip chart.

STANDARDISED CONSTRUCTION

The **MINIPONT** is presented in a metal case containing a sliding tray on which are mounted two standard sub-assemblies, linked together by a lock-in connector : the electromechanical block and the electronic block. The electronic section can be brought out of the case, for inspection or adjustment, without any interruption to its operation; in this position, it is possible to disconnect the chart frame to replace the chart. The two sections are interchangeable, and can easily be removed, from the front of the case, for replacement or maintenance purposes; there are no connections to be unscrewed or unsoldered.

A COMPLETE RANGE OF AUXILIARY DEVICES

A range of complementary devices makes the MINIPONT adaptable for the most varied auxiliary functions .

- Signalling, operation of alarms, "measurement" failsafe devices.
- 2 - or 3 - speed discontinuous control; continuous control by pneumatic controllers - PAT, CAT, DAT or DIAT.
- Electric or pneumatic remote transmission of the measured variable, to a controller, an indicator o a recorder.

* The intrinsic safety is guaranteed for the outer input circuit. In that case, certain options are excluded. Please consult us. (Certificates n° 153.249 of the LCIE and n° BM 3 - 4.125/151 E of VERITAS office).

SPECIFICATIONS

The type D MINIPONT has an all silicon transistor electronic section. The exclusive use of semi-conductors reduces heating up and guarantees a stability sufficient to have enabled us to eliminate any testing or standardising devices from the front plate. The whole of the transistorised electronic section is interchangeable with the valve unit, which can be substituted for the former instantaneously.
Special care has been taken with the insulation, increasing the instrument's insensitivity to extraneous voltages, and making possible the measurement of sources whose potential is considerably different from that of the mass (sleeved couples under power, for example). On the multi-point version, a fifth commutator may be provided, for switching the screen of each measuring line when the detectors are energised with respect to ground.
Further, the slide wire is mounted in a dust proof case ensuring long life and infrequent maintenance.

Measuring circuit

Function	Electrical span	Guaranteed accuracy
Pyrometer for thermocouples	100°C minimum	±(20µV + 0.15% e + 0.5% E)
Millivoltmeter	2 to 200 mV	±(5µV + 0.15% e + 0.5% E)
Wheatstone bridge	2 to 185 Ω(5 to 500°C)approx.	±(0.2°C + 0.4% E)

E electrical span
e amplitude of zero offset, if any

Nominal response time	: 2 s.
Dead band	: 0.15 % of electrical span
Maximum source resistance	: 5000Ω
Input impedance at unbalance	: 5000Ω
Influence of extraneous voltage	: for a variation of less than 0.3% of the electrical span, it is possible to apply :

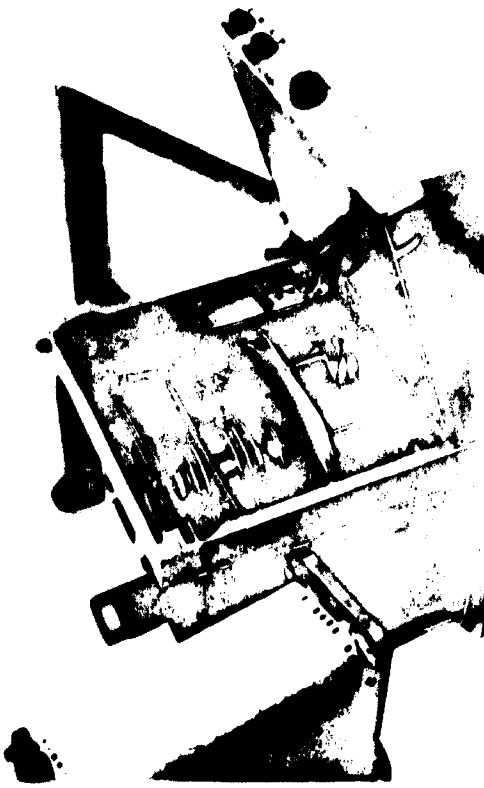
-between one measurement terminal and the ground terminal : a direct voltage equal to 10 000 times the electrical span (CMR, 150 db for 1000Ω source resistance) or an alternative voltage equal to 10 000 times the electrical span, but equal, at most, to 60 V (CMR, 150 db for 1000Ω source resistance).
-between measuring terminals : an alternative voltage equal to 60% of the electrical span.

Maximum ambient temperature	: 45°C
Multi-point instruments	: 6 points (can be adapted for 2, 3 or 4).
-Nominal switching speed	: 7.5 s. (2.5 s. on request). Commutation may be stopped by operation of a switch or remotely controlled.

RECORDING

Useful chart width	: 120 mm
Visible length of chart	: 100 mm
Length of chart roll	: 25 m (1 month's recording at 30 mm/h)
Printing	: Continuous curve on SP model,series of dots on MP model
Chart speeds	: 15, 30 and 60 mm/h., by change of pinion. Remote control available.
POWER SUPPLY	: 110-127 or 240 V ±10% 50 Hz. Measuring bridges are powered by voltage source stabilised with Zener diodes.
Consumption	: SP instrument : 30 VA MP instrument : 40 VA
Connections	: At rear of instrument, by terminals. (Plug-in connectors if required).
DIMENSIONS	: 193 x 190 x 533 mm (terminal output) 193 x 190 x 546 mm (connectors).
Colour	: Grey
Weight	: 22 kg

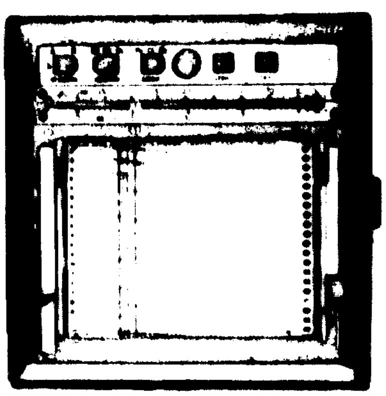
CONTROL FUNCTIONS



Micro-contacts and additional slidewires can be added to the **MINIPONT**, for operating controls, alarms, signals and electrical remote transmission systems. But these elements are not mounted in an isolated fashion; they are grouped on plug-in control function units. These control modules have a bushing, integral with the measuring shaft, on which are mounted the slidewire cursors and the cams for operating the contacts. The corresponding wiring is always provided, to that a control unit may be added to the MINIPONT without any modification to the latter. This degree of standardisation does not, however, diminish the flexibility of the instrument; the control unit is not designed specifically for a certain function, and the distribution and use of contacts over one or several points of measurement may be selected as desired. The control unit may contain up to 2 slidewires or 7 contacts. A detailed description will be found in the leaflet NAC 102 L.

OTHER MINIPONT INSTRUMENTS

TYPE A MINIPONT RECORDER SPECIAL MEASUREMENTS



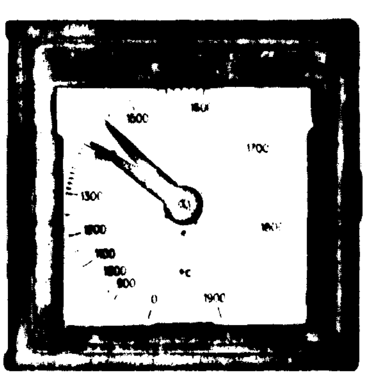
In the type A **MINIPONT**, more than 10 000 of which are already in service, the valve-type electronic system has been retained, to enable the instrument to be adapted for certain special applications.

The type A **MINIPONT**, which thus complements the type D, and has the same format, is now reserved for measurements of particular types : pH and oxydo-reduction potentials involving a very high source resistance, electrolytic conductivity measurements requiring the use of a-c powered bridges, measurement of temperature of alternator rotors by the Kelvin bridge method.

A special "Marine" version of the type A **MINIPONT** may also be provided; in this model, the mechanical elements have been specially strengthened so as to answer the requirements of the French Navy in the matter of resistance to shock.

As the metrological characteristics of this instrument are largely dependent on the application for which it is required, it is advisable to consult us about each specific case.

ROUND DIAL MINIPONT INDICATOR



If no recording of the measured variable is required, the **MINIPONT** can be supplied in a single-point indicator version, with a round dial (graduated circumference 375 mm). The pointer is integral with the measuring shaft.

By means of external switching boxes (see bulletin T 11) the indicator can be linked to any one of a certain number of detectors.

Like the recorder, the indicator may be equipped with optional devices for operating alarm, signal or control systems.

MECI 123, BOULEVARD DE GRENELLE (75) PARIS 15e Téléphone : 306-90-00 306-27-00
Société anonyme au capital de 24 000 000 de francs Télégramme MEC

OPTIONS

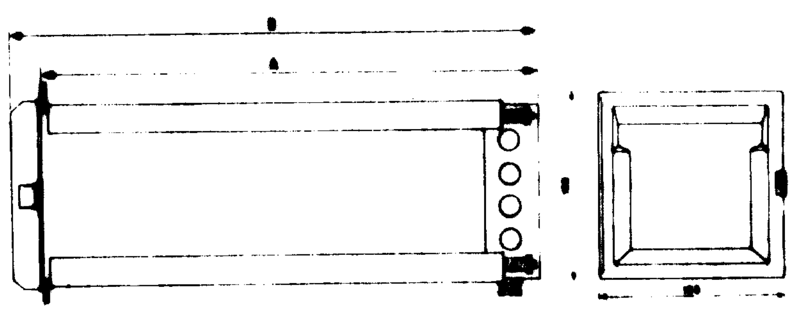
Apart from the control function assemblies (see page 4) the Type D MINIPONT can, on request and at additional cost, include a certain number of additions of alternatives. The principal options are indicated below.

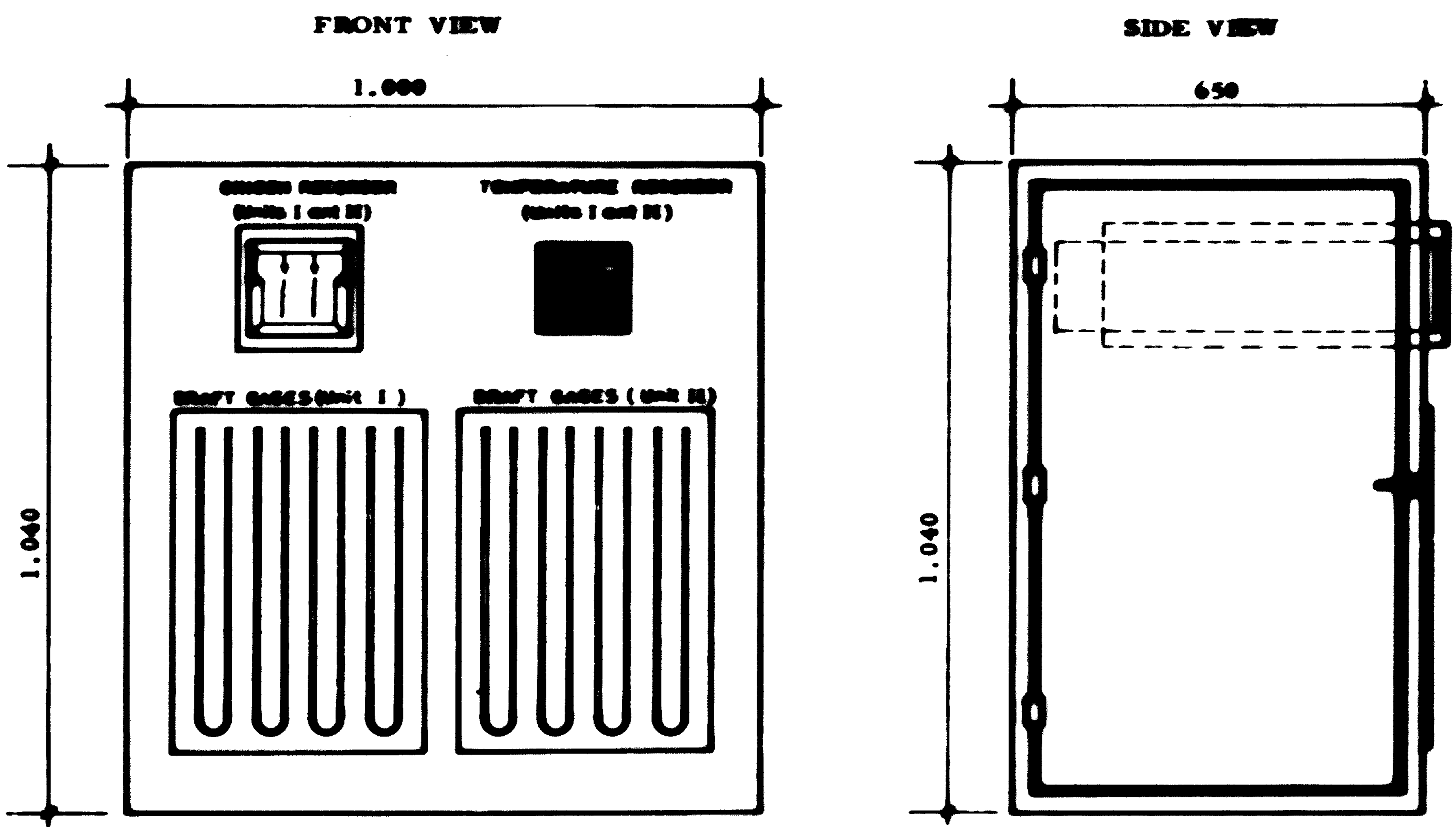
Since it is not always possible to equip a single instrument with several options, the customer should consult our branch offices.

Optional function or device	Observations
"Measurement" failsafe · measurement pointer driven to one end of scale in event of break in measuring circuit.	Included on request · all millivoltmeters and pyrometer-potentiometers.
"Intrinsic safety" feature, for measuring circuit.	
Two measuring ranges, choice by manual selector switch.	
Two measuring ranges, automatic selection.	Multi-point recorders.
A fifth commutator, for external synchronisation or for connecting screen of each measuring line when detectors are energised with respect to ground.	Multi-point recorders.
Inversion of bridge.	
2, 3, 4 point recording (terminals bridged, print wheel changed).	Multi-point recorders.
2.5 sec. switching speed.	Multi-point recorders.
Electrical speed change (multiplies basic speed by 10 or 60). Manual or remote control.	
Folding strip chart.	
2 module version.	Requires use of plug-in connectors.
Pneumatic transmitter incorporated.	
Plug-in connectors.	Obligatory, for instruments with pneumatic transmission, or two-module versions.
Black case.	
Special, "Series 06" door.	
Chart lighting.	

MOUNTING DIMENSIONS

Version		A	B
Output by terminal		500	533
Output by connector		513	546
2-module	electromechanical	367	400
	electronic	380	393

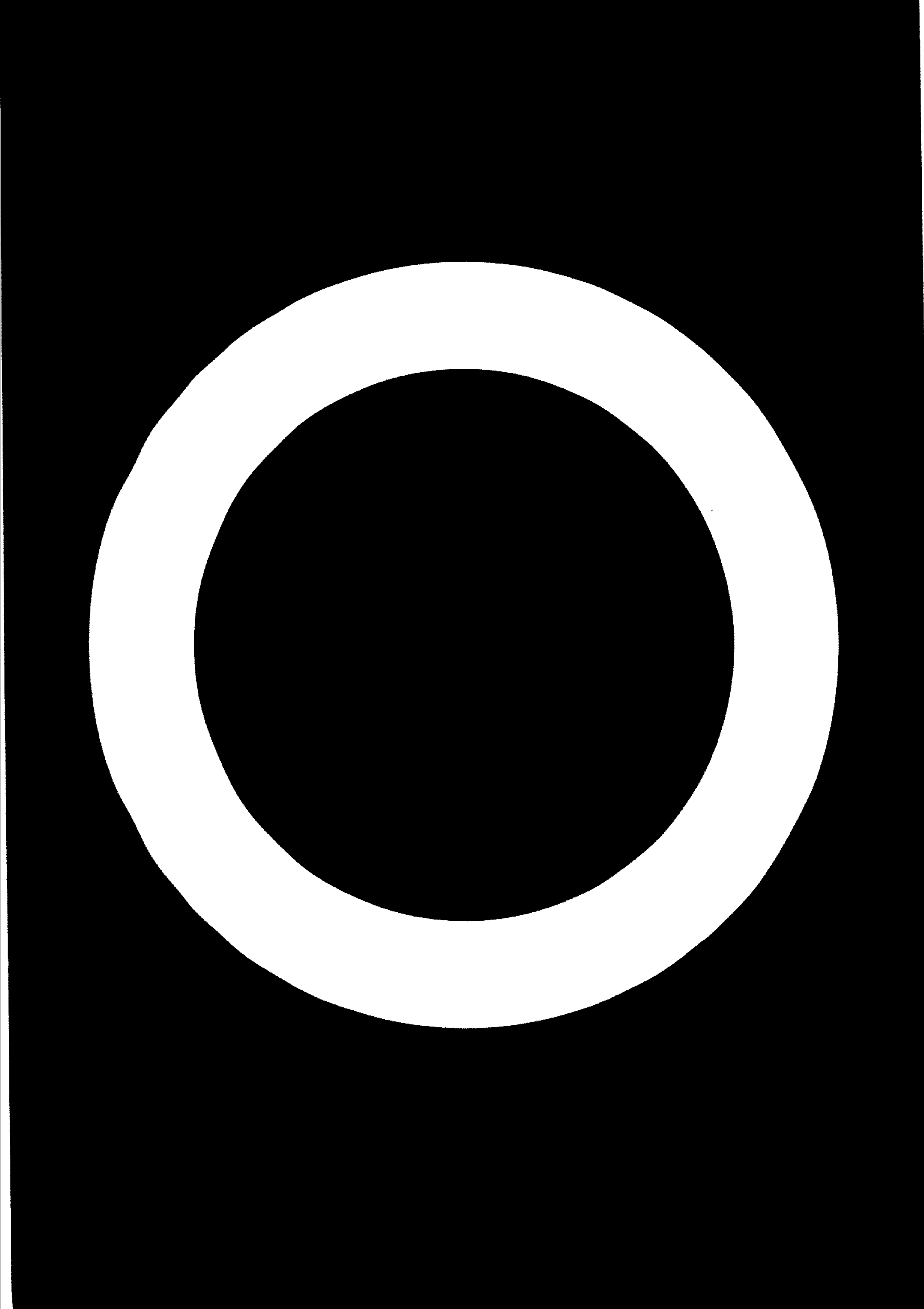




DRAWING D-1

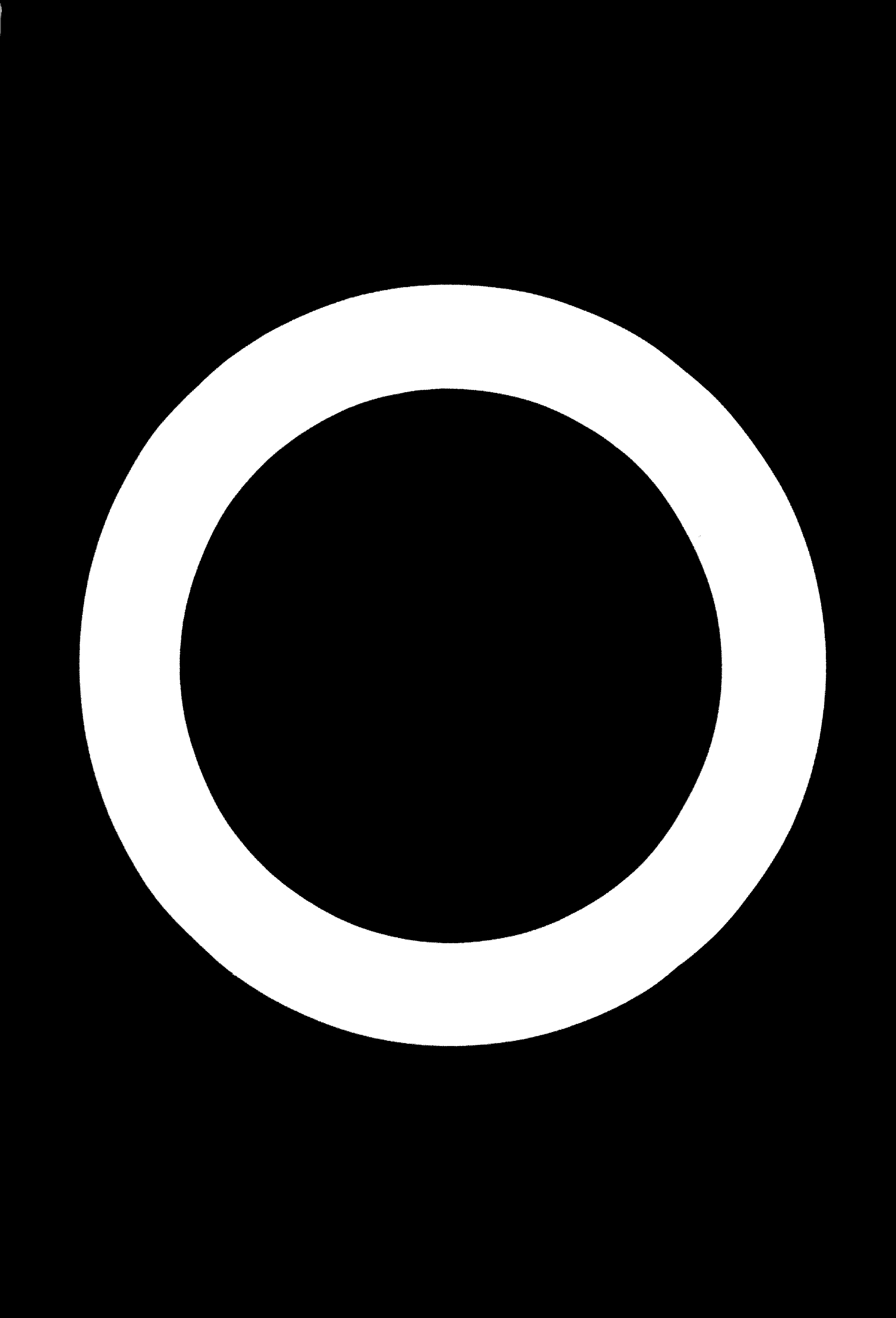
SCHEMATIC PANEL ARRANGEMENT FOR CONTROL INSTRUMENTATION

SCALE 1:10



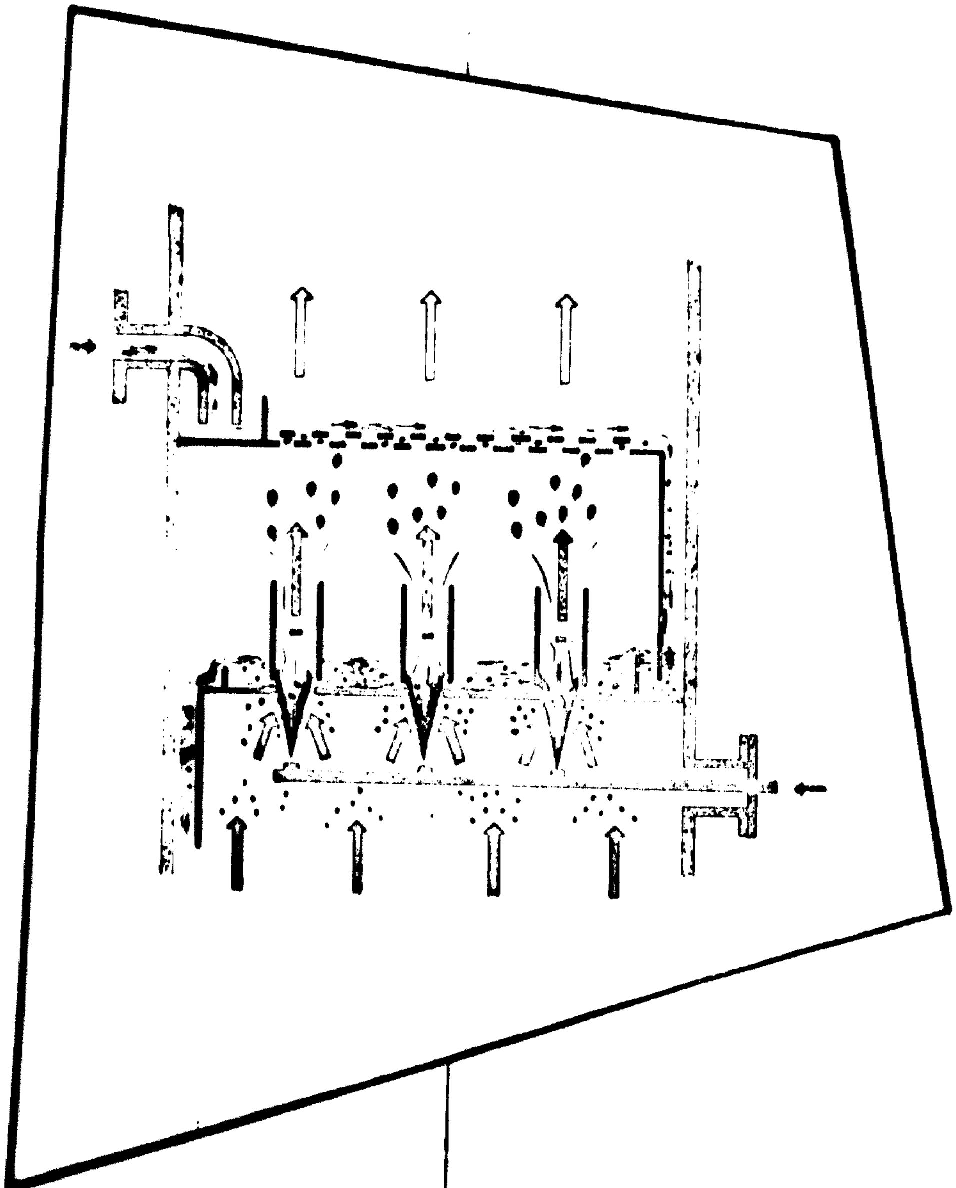
ANNEX - E

GAS SCRUBBERS



Bulletin 962

FOR THE OPTIMUM IN GAS PURIFICATION, PRODUCT RECOVERY AND CONTROL OF ATMOSPHERIC POLLUTION BY WET SCRUBBING



SWEMCO

HIGH EFFICIENCY GAS SCRUBBERS

SERVING INDUSTRY

Experience

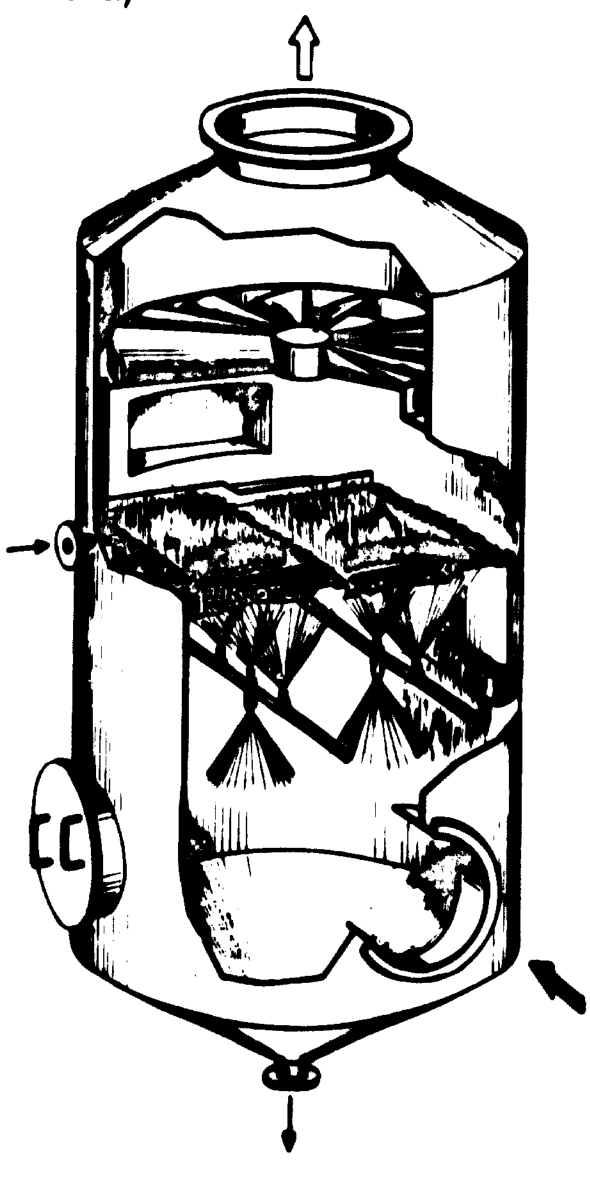
SWEMCO has available the accumulated experience of over 30 years of designing and manufacturing WET SCRUBBERS. SWEMCO WET SCRUBBERS offer optimum efficiency; the outcome of many years of research, pilot plant study, engineering and proved experience. Proved, and even still improved, methods of wet scrubbing are available to you that allow SWEMCO WET SCRUBBERS to deal effectively with virtually any Gas Cleaning problem. SWEMCO serves industry in such diversified fields as atomic energy . . . iron and steel . . . chemical and petrochemical . . . petroleum . . . pulp and paper . . . fertilizer . . . mining . . . building . . . non ferrous metals and transport.

SWEMCO

Wet Scrubbers

COMBINE EFFICIENCY AND ECONOMY

proved by success



SWEMCO WET SCRUBBERS incorporate the improved and up-to-date design of the contact device contained in the original Harmon patents and known as the multi-jet impingement baffle plate. SWEMCO impingement baffle plates are of simpler design and manufacture, resulting in a lighter yet stronger plate, a more accurate yet less expensive essential item of plant.

The plate consists essentially of a perforated plate and a baffle grid assembly. The two sections are fitted together such that an individual baffle is mounted directly over each perforation at the point of maximum velocity.

OPERATING PRINCIPLES

Flows of liquor and gas are countercurrent. Liquor supplied to the impingement baffle plate ensures that the upper surface is completely submerged. The gas in passing up through the perforated sheet is subdivided into many thousands of jets. These jets impinge at high velocity on the wetted baffles located above each perforation. The actions of impact and sudden change in direction form the basis for the improved efficiencies of this direct contact device.

Each unit is individually designed. The type and number of impingement baffle plates depend on the application and desired performance. Units have been installed containing from 1 to 12 plate stages.

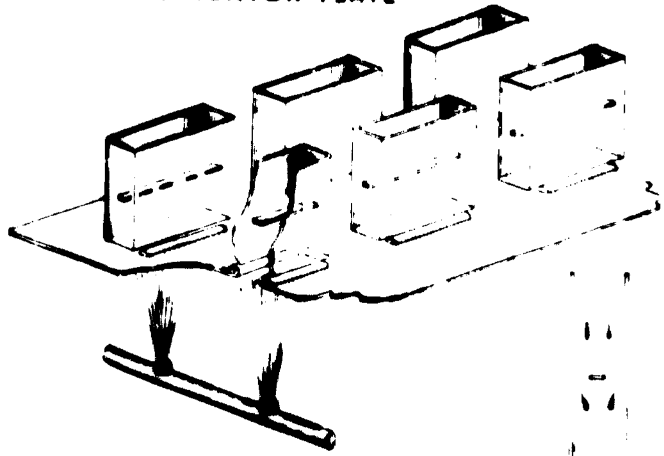
SIMPLE YET EFFECTIVE

VENTOR PLATE

GET MAXIMUM WORK FROM MINIMUM ENERGY!

With the addition of the patented VENTOR plate to our range, SWEMCO HIGH EFFICIENCY SCRUBBERS result in optimum performance; maximum return for pressure drop. The VENTOR plate alone allows us to tackle "sticky" substances and "difficult-to-wet" particles. In conjunction with impingement baffle plates, sub-micron dusts or fumes are effectively removed.

TYPICAL 'VENTOR' PLATE



THE VENTOR PLATE

The VENTOR plate is fitted on a diaphragm, similar to the impingement baffle plate, horizontally in the tower. The plate consists of a series of rows of rectangular slots having a chimney situated over the slots. The throat of each slot is specially rounded, and a baffle is located within each chimney.

SECTION THRO' 'VENTOR'

OPERATING PRINCIPLES

Liquid is passed across the plate, and a series of low pressure nozzles inject atomised liquid directly into the throat of each slot. The gas can only pass through the "rounded slots", where it is accelerated to a high velocity and impinges upon the liquid stream. The gas decelerates in passing up through the chimney and on meeting the baffle, further acceleration and impaction is obtained giving effective agglomeration.

The actions performed above require energy, which is expended in terms of pressure drop. The VENTOR can be designed or adjusted to give any desired result, due to the accepted relationship between gas cleaning performance, particle size and pressure drop. The VENTOR reduces power consumption to a minimum.

PERFORMANCE CHARACTERISTICS OF SWEMCO HIGH EFFICIENCY GAS SCRUBBERS

Combination of VENTOR and impingement baffle plates can give virtually 100% gas cleaning efficiency. VENTOR for agglomeration, impingement baffle plates for removal—in a single tower.

A single impingement baffle plate removes over 97% of all dust over 2 microns in size at pressure drop of 1½" w.g. Plates having different size perforations and velocities available.

Low water consumption of 1-3 g.p.m per 1000 c.f.m. Low liquor pressures. Recirculated liquors do not impair efficiency. Slurries with up to 65% solids have been handled.

Possible to clean, cool and absorb gases separately or in one operation in the same unit. High rates of heat and mass transfer obtained.

A single impingement baffle plate can be compared to up to 6 feet of standard packing on cooling and many absorption and stripping jobs.

Flexibility: Wide variation in flow possible. Able to interchange all plates. Pressure drop simply adjusted to any desired figure; same size tower whether p.d. of 35" w.g. or 3.5" w.g.

Unit compact due to high allowable tower velocity and optimum tray spacing. Simple liquid distribution, ensured intimate gas/liquid contact; no channelling possible.

8 Efficient anti-entrainment device to ensure negligible droplet carry-over.

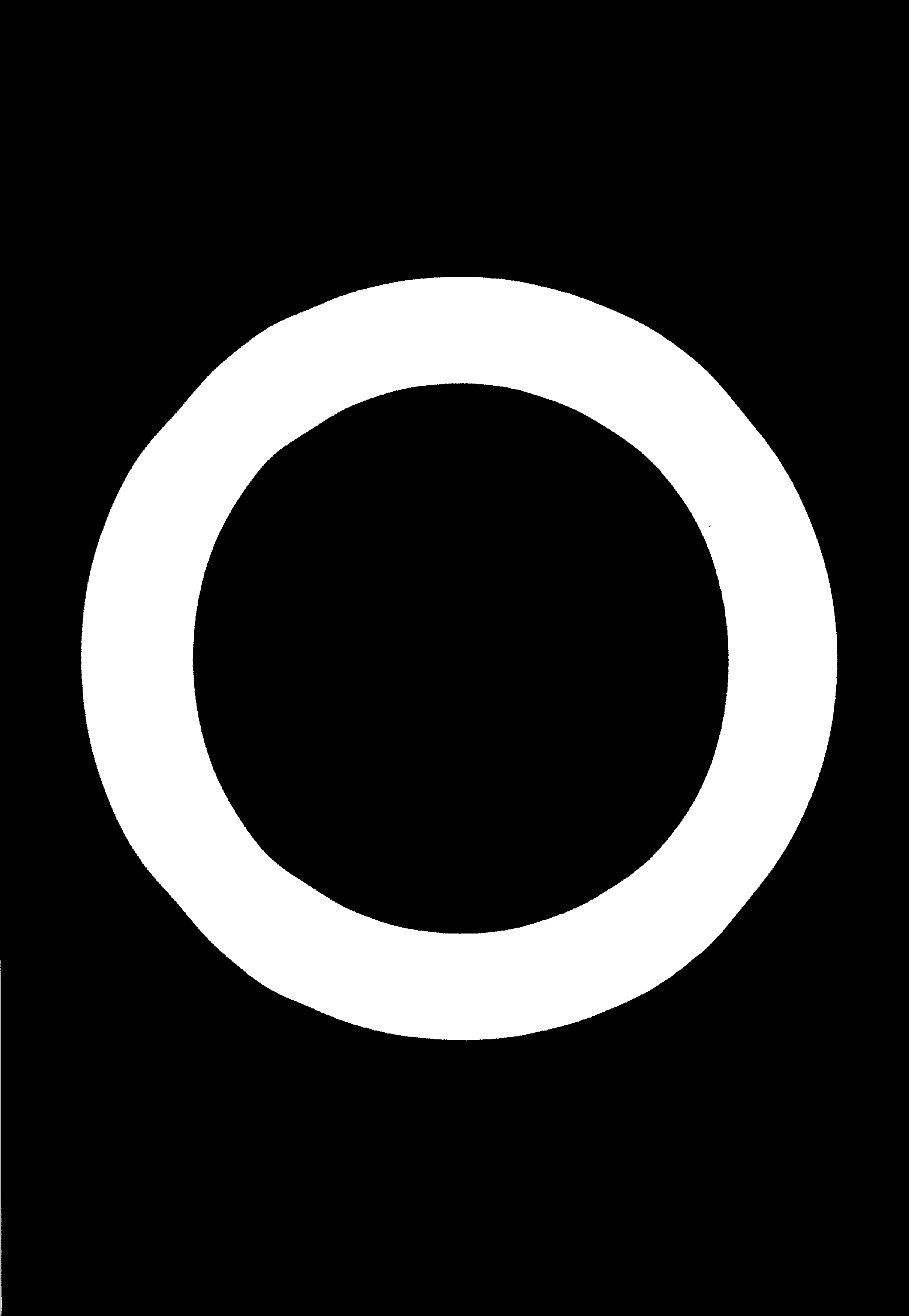
9 Versatility: Most known corrosive gases and liquids handled; temperature is no problem. Plates simply fitted and removed; can usually be installed in existing tower. No moving parts and low maintenance.

10 SWEMCO WET SCRUBBERS protect proceeding equipment, and reduce ancillary plant items to a minimum.

COMPACT YET COMPLETE

ANNEX - F

THE ALMADEN-CENIM PROCESS FOR THE TREATMENT OF MERCURY STUPPS



1. <u>Introduction</u>

In all mercury recovery plants that treat the cinnabar though pirometallurgical methods, stupps are produced. The rea son for the formation of stupps is based, in that during the con densation process, a part of the mercury droplets are coated - with dust and oily matters, which impede their coalescing.

For the recuperation of the mercury contained in the stupps several processes have been proposed, but the most wide ly used system is the hoeing of the stupps after mixing with uns laked lime. If only water were present in the stupps, mixing with lime would be enough to the coalescing of all mercury droplets, but as normally solids and fats are also included, the only drop-lets which may be able to coalesce are those of biggest sizes, therefore, a residue is left, and as such residue has a high con tent of mercury, it is necessary to recycle it to the kiln with the following inconveniences.

a) <u>Toxicity</u>

Each time that hoeing of the stupps takes place there is a strong emission of water vapor, that contains a great amount of mer cury, which produces an environment content of mercury high er than the maximum admissible concentrations of 0.1 mg/cu. m.

b) <u>Recycling</u>

As the volume of recycled mercury is about the same as that one produced, it is compulsory to give back to the kiln the produced stupps. Such procedure causes great unbalanced ope rating conditions and strong losses.

2. <u>The ALMADEN-CENIM process</u>

The above inconveniences have encouraged the investiga tions of a new process for the handling of stuppe. The process has been developed jointly by the National Center of Metallurgical Investigations (CENIM) at Madrid and by the Almaden mercury plant at Almaden (Spain). (1)

The investigation aimed at the obtainment of the following objectives:

a) Possibility of process automation
b) Vaportight operation
c) Residues to be disposed of should bear no mercury
d) Any residue with mercury content (if any) must be in enough - small amounts allowing its treatment in an auxiliary furnace of very small size.

The process is based on the action of caustic reagents at about 100º C on the stuppe, with the consequent hydrolisis of the oily film and the cleaning of the dust which coats the mercury droplets.

The process is schematically shown in diagram appended.

It has been found that the stuppe contain a part of mercury in oxidised form, and in this process remains in solution.

Therefore, after treatment of the stuppe by the Almaden-Cenim process it is obtained:

a) Metallic mercury in liquid form, which is easily siphoned out of the reactor.
b) A solution where all mercury contained into the stuppe in the form of oxides or sulphates is found.

---

(1) Such process is covered by patents. Therefore only its general lines can be disclosed in the present description

c) The solid particles, that due to their very fine size are not retained by the cyclones, remain in suspension in the solution

In order to illustrate the possibilities of application of the Almadén-Cenim process to the case of Turkey, some general considerations are made below, taking by way of an example the case of Konya. It is important to emphasize that to determine the final possibilities of application of the process to the Konya or Haliköy plants it is compulsory to perform thorough testing at laboratory and pilot plant scale with samples of their stupps, in order to ascertain the suitability of the process to such plants.

3. Metallurgical plant of Konya

3. 1. Treatment of the stupps with lime

3. 1. 1. Materials balance and condensing efficiency

According to the calculations made in Section IV of this Report, paragraphs 3. 1. and 3. 2. 2., the most significant fact is that during 1971, some 54 tons of mercury have been recycled at Konya, and that if such procedure could be discontinued, condensing efficiency would be increased in about 4%.

Therefore, the most important conclusion drawn from the above calculations is that recycling of the stupps should be discontinued.

Of course, such stupps must be treated in an auxiliary furnace in order not to shorten the mercury production almost to half. The treatment might be made in a retort of about 1 ton/day capacity.

3. 2. Treatment of stupps by the Almadén-Cenim process

If the treatment of stupps would be performed at Konya by this process instead of using the lime method, the efficiencies to be obtained are calculated in an approximate form below.

We assume that the efficiencies to be obtained would be similar to those of Almadén, as the characteristics of the Konya ores do not show unfavorable factors.

According to the aforementioned data, the stupps, disregarding the accompanying water, would contain:

Mercury	141.87 x 0.939 =	133.22 tons
Dust (already calculated)		36.00 tons

By the Almadén-Cenim process, the 94% of the mercury contained into the stupps is recovered and is ready to be bottled, and remaining 6% stays in the residue (this 6%, corresponds to a 5% of the oxidized mercury during condensing, and 1% to the very tiny droplets which do not coalesce).

Then,

Mercury to be bottled	133.22 x 0.94 =	125.23 tons
Mercury into residue	133.22 x 0.06 =	8.00 tons

If the residual pulp from the treatment is suppose to have 20% humidity, its composition would be:

Mercury Content	8	tons
Solids	36	tons
Water (20% of 36 tons)	7.2	tons
	51.2	tons

The above figure indicates, that now only 171 kg/day should be treated in a retort. If 4 operations are performed by day the needed capacity of the retort is only for 43 kg with -- 15.6% mercury. Its dimensions are so small, that design problems and investment would be negligible.

4. Conclusions

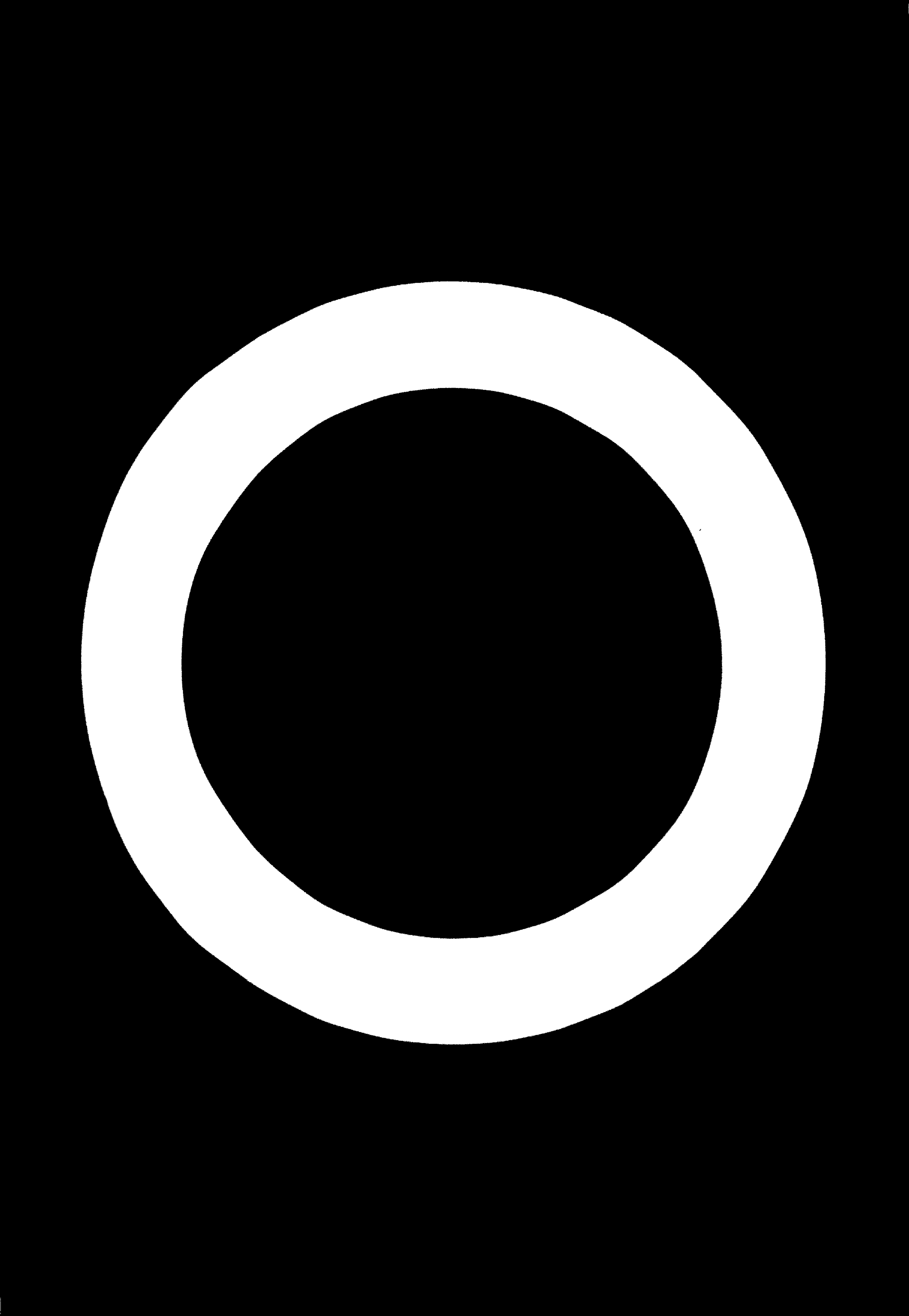
From the above it can be concluded:

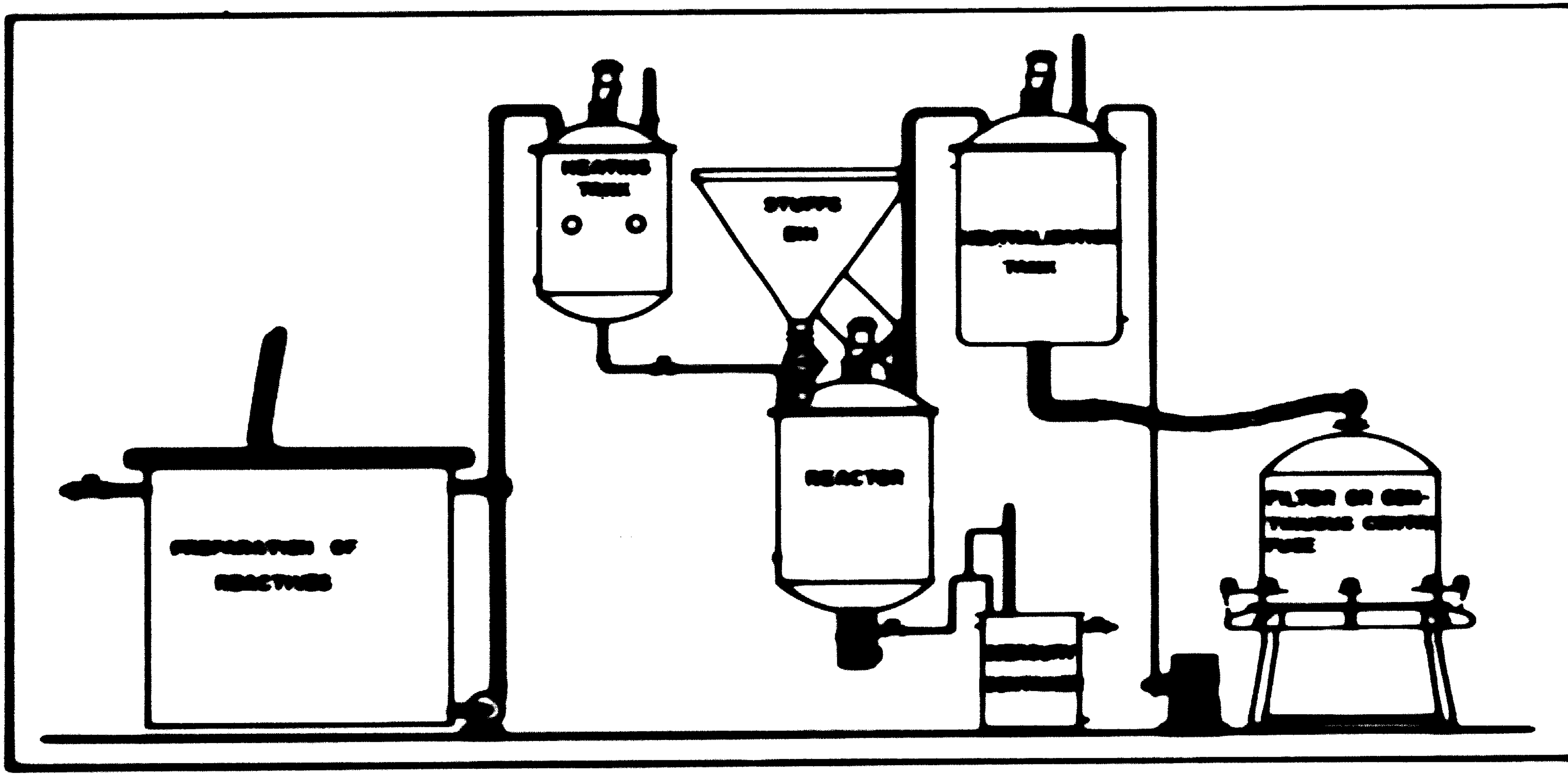
a) The recycling of mercury as a residue of the mercury hoeing method, carries along with mercury losses of about 4%.

b) To avoid recycling it is necessary to distillate the mercury into an independent retort. In the case of lime use the capacity of the retort should be 1 ton/day. This volume is excessive taking into account that the heating must be indirect.

c) With the Almadén-Cenim process the amount of residue to be treated is reduced to 170 kg/day.

d) The higher expenses of the Almadén-Cenim process are fully compensated by the much lower expenses of the retort processing of the residue. These advantages are shownin Table F-1 appended.



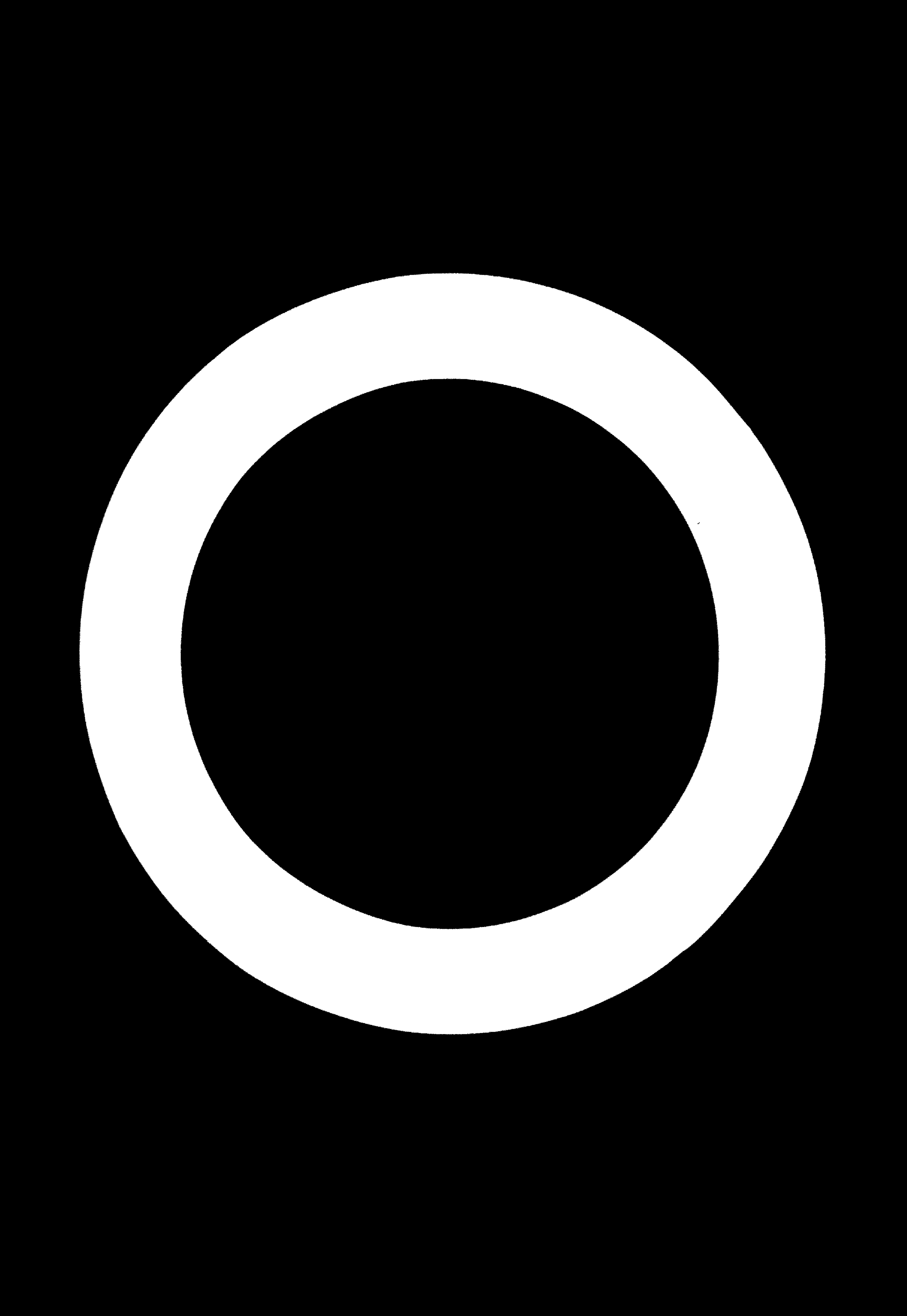


SCHEMATIC DIAGRAM OF THE [illegible] PROCESS FOR THE TREATMENT OF MERCURY STUFFS

TABLE F-1

COMPARATIVE RESULTS OF SEVERAL STUPPS TREATMENT PROCESSES AT KONYA PLANT

	Present process with recycling of stupps	Present process with retort treatment of residues.	Almadén-CENIM process
Mercury to be treated			
Mercury in ore	147,45 t	147,45 t	147,45 t
Mercury produced	122,5 t	127,48 t	127,48 t
Treatment of stupps			
Stupps to be treated	272 t	224 t	224 t
Expenses			
Reactives	3.200 $	3.200 $	1.500 $
Power	insignificant	insignificant	1.000 $
Depreciation	-	-	6.000 $
Treatment of residues			
Residues to be treated		300 t	51,2 t
Mercury content		54 t	8,0 t
Expenses			
Power		6.000 $	800 $
Depreciation		4.000 $	1.500 $
Total expenses			
Treatment	3.200 $	3.200 $	8.500 $
Recovery	-	10.000 $	2.300 $
Total	3.200 $	13.200 $	10.800 $
Mercury recovery			
Reovered mercury		5,34 t	5,34 t
Extra cost per Kg		1,87 $	1,42 $



ANNEX - G

ANALYSIS OF STOPPAGES

1. General

In this annex an analysis of the stoppages of the metallurgical plants of Konya and Haliköy is included.

Stoppages have been classified at Konya according to their origin, i.e., metallurgical stoppages, (slag formation, defective feeding of the kiln, etc.), mechanical stoppages (caused by breakdowns or failure in facilities of the plant), stoppages by repairs or maintenance (refractory relining, etc.), stoppages by motives foreign to the plant (power failure, lack of ore, etc), and miscellaneous stoppages (stoppages whose motives are not specified in the plant's records).

At Haliköy, where other record systems exist, it has not been possible to follow the above clasification, and another pattern of analysis has been adopted.

Stoppages duration are given in hours and where possible have been attributed separately to each one of the two kilns existing at Konya and Haliköy plants (Units I and II).

2. Stoppages at Konya Plant

2.1. Year 1971

2.1.1. Metallurgical stoppages

	Unit I	%	Unit II	%
Slag formation	120	65	126	86
Scrubbers cleaning	-	-	2	1
Defective feeding of the kiln	63	35	17.5	13
	183 h	100	145.5 h	100

2.4.2. **Stoppages foreing to the plant**

	Unit I	%	Unit II	%
Lack of ore	142	79	121, 50	79
Sticky ore	3, 50	1	4	25
Frozen ore	16, 50	8	1	0, 5
Change of ore stock	17, 0	9	15	9
Snowfalls	1, 0	0, 5	15	9
Lack of electric power	4, 0	2, 5	-	-
	184 h	100	156, 50 h	100

2.4.3. **Stoppages by mechanical breakdowns**

	Unit I	%	Unit II	%
Calcines vibrating feeder	51, 5	18	24	14
Calcines track	82, 0	28	42	25, 5
Conveyor belts	18, 0	6	1	0, 5
Crushers	6, 0	2	9	5, 0
Burner	7, 50	2, 5	3	2, 0
Electrical motors	6, 0	2	10, 50	6, 0
Air compressor	11, 0	3	12, 5	7, 0
Calcines bin	102, 5	38	65, 0	28, 0
Fuel-oil tanks	1, 5	0, 5	3	2, 0
	286, 0 h	100	170 h	100

2.4.4. **Stoppages by repairs and maintenance**

	Unit I	%	Unit II	%
Refractory relining or repair	120	100	588	100
	120 h	100	588 h	100

2.1.5. Miscellaneous

	Unit I	%	Unit II	%
	-	-	2.208	100
	-	-	2.208 (1)	100

2.1.6. Summary

	Unit I	%	Unit II	%
Metallurgical stoppages	183	24	145.5	4
Stoppages foreign to the plant	184	24	156.5	5
Breakdowns	286	37	170.0	5
Repairs and maintenance	120	15	588.0	18
Miscellaneous	-	-	2,208.0	68
	773 h	100	3,268 h	100

2.2. Year 1972 (January, February, March)

	Hours	%
Metallurgical stoppages	53.0	23.4
Stoppages foreign to the plant	61.0	27.0
Breakdowns	112.5	49.6
Repairs and maintenance	-	-
Miscellaneous	-	-
	226.5	100

3. Stoppages at Haliköy plant

3.1. Year 1971

Table G-1 annexed shows the different stoppages produced at the plant during 1971 and their reported causes.

3.2. Year 1972

Table G-2, shows the different stoppages produced during the months of January, February and March 1972.

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(1) January, October and November 1971

For February and March only total hours for stoppages are available without possible discrimination between Units I and II.

TABLE G-2

STOPPAGES AT HALIKOY PLANT DURING THE FIRST QUARTER OF 1.972

Month	Kilns	Ore shaker feeder	Exhaust fan	Cleaning	Miscellaneous	Total
January	Unit I	4, 30	18	2	-	24, 30
	Unit II	1	18	4	4	27, 00
Total		5, 30	36	6	4	51, 30
February						11, 45
March						26, 40

Unit= hours

TABLE G-1

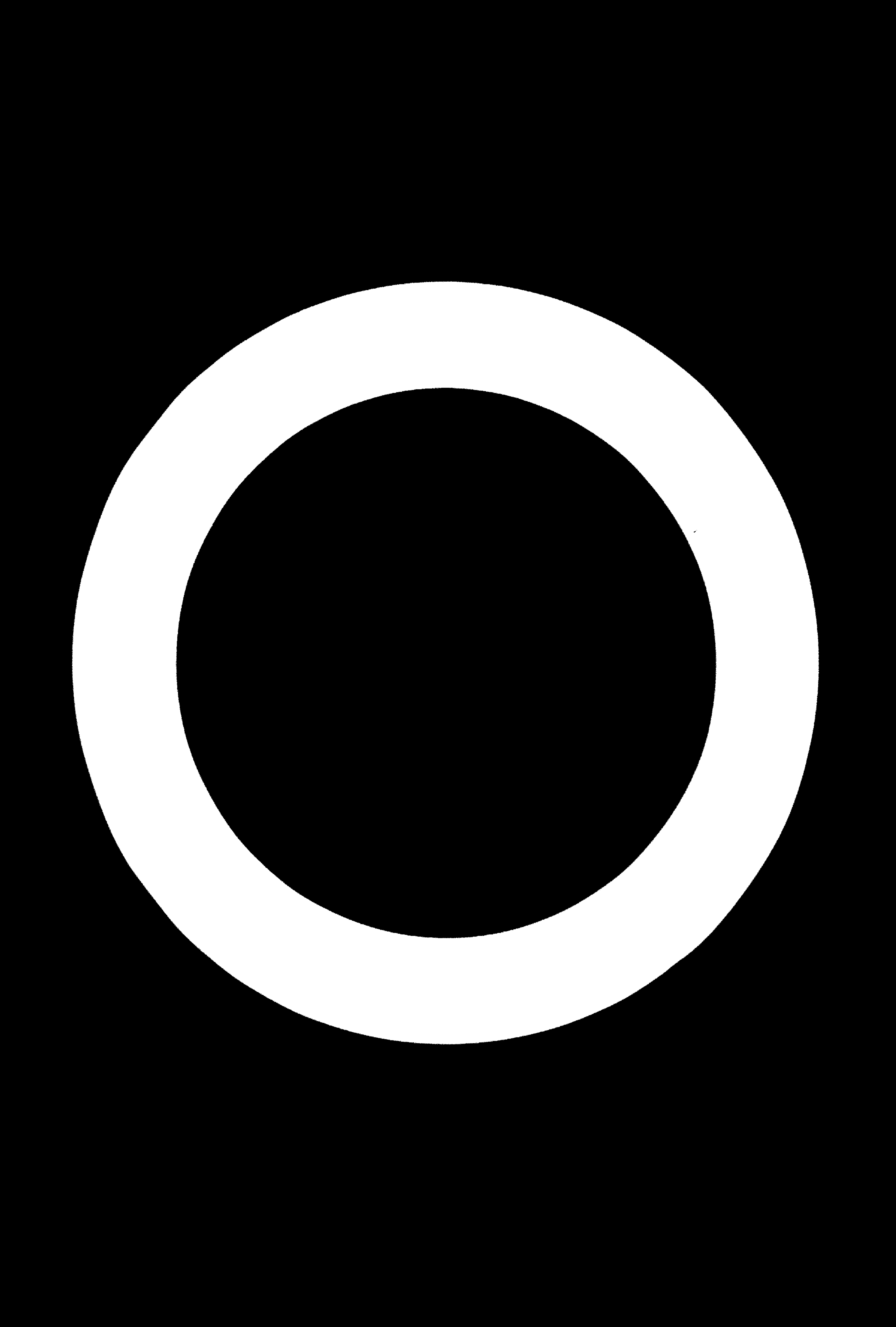
STOPPAGES AT HALIKOY PLANT DURING 1971

Month	Kilns	Ore shaker feeder	Combustion air fan	Exhaust fan	Burner	Electrical power failure	Cleaning	Miscellaneous	Total
January		9. 30		81			6		96. 30
February	450 (1)	6		2	2		4		464
March		7. 30		15. 30			6		29
April	35	2. 30	0. 30	3		2	3		46
May	60			3		2	5. 30		70. 30
June	512 (2)		12	7	0. 30	48	1. 30		581
July		16	8	2		12. 30	5		43. 30
August		35. 30	70		2	5	6		119. 30
September		30. 30	1	10	1. 30	2	7	6	58
October	215 (2)	4. 30	5. 30	19			4. 30		248. 30
November		9		24			4. 30		37. 30
December	4	38	2. 30	7	0. 30		4. 30		55. 30
Total (3)	1,276	159	100	173	6	71	57	6	1,848
Percentage	69%	8.6%	5.4%	9.4%	0.3%	3.8%	3.2%	0.3%	100%

- Units: hours
- Notes: (1) Unit I
  (2) Unit II
  (3) Rounded figures

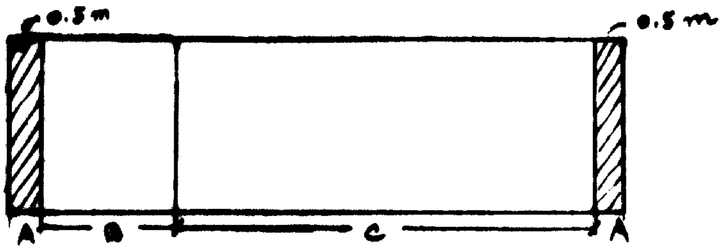
ANNEX - H

REFRACTORIES



1. **Specifications of presently used refractories and recommendations**

Both Konya and Halikōy plants use the following refractories, located according to the diagram below:



Their specifications are included in Table H-1.

The refractories used by us for the roasting of very abrasive ores, and which have shown lives above two and half years have the following specification. (see Table H-2).

The above refractory, has been also used with very good results for the roasting of very abrasive ore, of the breccia type at the metallurgical plant of Cordero Mine, Nevada (U. S. A).

TABLE H-1

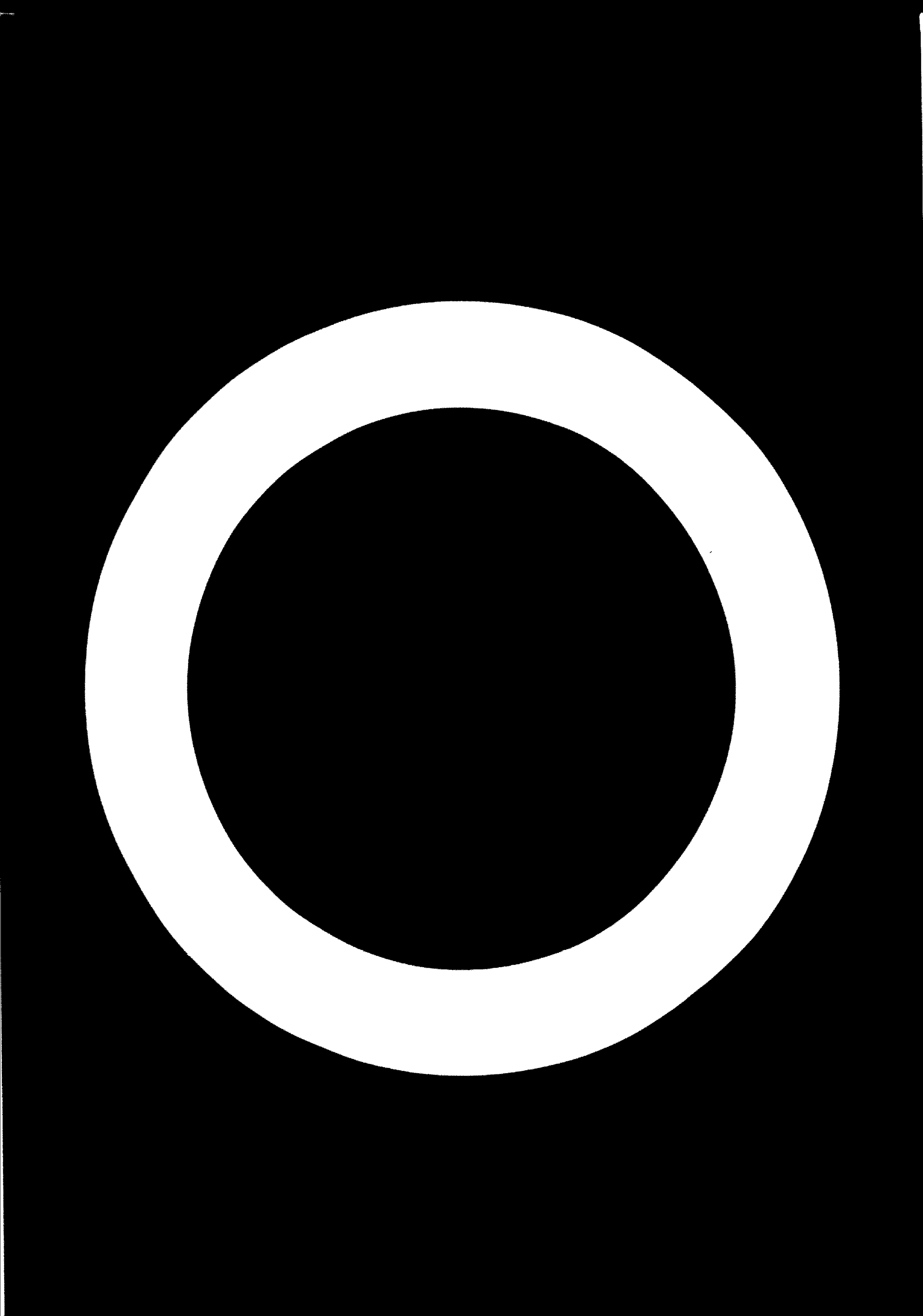
Type	Al_2O_3	SiO_2	Fe_2O_3	CaO	gr/cm^3 δ	Porosity %	Cold resistance Kg/cm^2	Softening T° C	Working T° C	Seger Cone
A	44-46	52-54	1.5-2	0.5-1	2.1	20	400	1,750	1,350	33-34
B	20-25	70-76	0.5-1.3	-	2.5	12-14	700	-	-	-
C	30-35	70	1-1.5	0.5-1	2.1	14-16	600	1,650	1,650	28-30

TABLE H-2

Al_2O_3	SiO_2	Fe_2O_3	gr/cm^3 δ	Porosity %	Cold resistance Kg/cm^2	Softening T° C	Working T° C	Seger Cone
40-50	45-50	3-4	2.5	10	500	1,850	1,500	36-37

ANNEX - 1

SAMPLING OF ORES DURING CONTINUOUS TRANSPORT



1. Introduction

In all concentration mills, in most loading or unloading installations and in many processing plants the control department has the problem of estimating the average quality of a batch of ore produced, loaded, unloaded or fed during a given length of time. The estimation of this average quality (content in element A, B, etc., sizing analysis, moisture content and so forth) is usually carried out by cutting increments of the ore to be analysed at regular intervals, either by hand or by a mechanical device.

Sampling is performed at a point where the ore transport circuit is broken, the sampling scoop travelling through the stream of ore and moving along a surface which we shall call the sampling surface. The successive increments are usually collected together in the same container and the bulk sample thus obtained is further sampled in several stages until the weight of the analysis sample is attained. The secondary sampling operations are made either on a continuous stream (automatically) or on the batch sample (by hand), according to the weight of the sample at each stage of reduction. The sampling stages generally alternate with the comminution stages.

The result of the analysis is an estimate of the unknown actual content of the batch being sampled. The purpose of this study is to recognize, and estimate, whenever possible, the cumulative errors which together constitute the total sampling error.

2. Sampling Errors

In the determination of the grade of an ore, different operations are performed which originate several errors that influence in the final result obtained.

Sampling may be classified into:

- Primary sampling
- Secondary sampling

Primary sampling is that performed on the feed-ore and before any modification in its structure takes place.

Secondary sampling includes all operations performed in order to prepare the laboratory sample.

Although, strictly speaking, the secondary sampling also covers the selection by the chemist of the portion for analysis from the laboratory sample, this final sampling stage is usually considered as part of the analysis and its variance is part of the analysis variance, since this latter is calculated by means of results obtained on different analysis samples taken from the same laboratory sample for practical purposes, therefore, the secondary sampling may be considered to be complete with the preparation of the laboratory sample usually a few hundred grams.

Both primary and secondary sampling induce several errors which may be classified as:

1) Discontinuity error that depends on the amount of sample taken and is independent of how the sample has been taken.

2) Grouping error, due to the uneven distribution of the ores and to the grouping of sampled particles.

3) Operating error, which depends on the way the sample is taken.

It is difficult to determine "a priori" which is the weight of each one of the above errors on the total sampling - error.

3. Errors in primary sampling

The primary sampling error is the consequence of the difference existing between the "actual" and the "ideal" prima ry sample. This error is essentially dependent on the way the pri mary sample has been cut and its study consists of a critical exa mination of the sampling machinery. Such study is merely outlined here.

Depending upon the type of sampler the cutter consists of:

- Type 1: a rectangular opening moving in a horizontal plane, the direction of the movement being perpendicular to the horizontal projection of the stream of ore;
- Type 2: a rectangular opening moving in a horizontal plane, the direction of the movement being parallel with the horizontal pro jection of the stream of ore;
- Type 3: a circular sector opening, revolving about a vertical or oblique axis;
- Type 4: a series of riffle splitters or any cascading device;
- Type 5: a revolving shutter, or
- Type 6: any container moved by hand

We shall consider only types 1 and 2 samplers as type 1 is the one existing at Konya, and type 2 at Haliköy.

- Type 1 and 2 samplers can be considered free from bias, what ever the distribution of the size and density fractions over the section of the stream, when the following conditions are satisfied:

C_1. the cutter speed is constant during the crossing of the mineral stream and is the same for all parts of the cutter;

C_2. the cutter is driven completely out of the stream on each side of it;

C_3. the opening is wide enough not to have a selective effect on the largest particles (at least three and preferably four times their diameter with a limited number of exceptions);

C_4. the head of the ore fall is as low as possible in order to pre vent the fines from flying (when the ore is dry) and the large particle from bouncing; the tendency to bounce is aggravated at high particle speeds;

C_5. the scoop is designed to prevent any overflow;

C_6. The cutter is free from clogging

C_7. the cutter speed is not so high as to allow racket-stroke effect; and

C_8. the cutter edges are sharp

Whenever one of these conditions is not fulfilled, a bias is likely to occur, the magnitude of which cannot be predicted.

When the opening of the cutter is determined according to condition C_3 above, and the speed of the cutter is regulated ac cording to condition C_7, discontinuity error dissappears or at least is greatly disminished. Grouping and operating errors are also diminished if conditions C_1 to C_8 are respected.

4. Errors in secondary sampling

These errors are the same ocurring in primary sampling but here each time a mass reduction and a grinding of the ore take place such errors appear at each reduction stage. Therefore it is necessary in order to reduce them to pay maximum attention to secondary sampling, avoiding fine ore losses and sample contamination. Samples must be carefully homogeneized, especially when taking the part of the sample to be sent to the laboratory.

5. Analysis Error

The analysis error may be determinated by replicate analysis of the same laboratory sample.

The variance thus calculated is valid only for specified condition, i. e. ore, element, method, chemist.

If V is the variance of a given analysis, and if a mean content is calculated from n results, obtained either from the same or from different laboratory samples, the analysis variance occurring is:

$$V_T = \frac{1}{n} \cdot V$$

The laboratory in each plant should know the variance V attached to each analytical method.

6. Influence on the total error

The influence on the total error of each one of the above considered errors is, on average, as follows:

- Primary sampling errors are about 10% of the total
- Secondary " " " " 60% " " "
- Analysis errors " " 30% " " "

7. <u>Interval estimation and confidence limits</u>

The object of this paragraph is to study the confidence interval given by the number of sampling analysis performed during our teste at Konya and Haliköy, i.e. to assess:

a) Limits of errore in ore grade resulting from the number of analysis made during the tests.

b) Minimum number of analysis that should have been made during these days in order to get a $\pm$ 10% limit of error.

c) Recommended procedure for the determination of the proper number of analysis to be made in the future in both plants.

In the following, mathematical derivation of the formulas has been omitted as only elementary statistical methods have been used.

7. 1. <u>Limits of errors</u>

7. 1. 1. <u>Konya</u>

For a probability coefficient of 0. 95, the error intervals have oscillated, according to the days investigated from $\pm$ 15% to $\pm$ 8. 3%.

The largest limit of error corresponded to the 24 April and the lowest to the 21 April.

According to the probability coefficient of 0. 95, we have for the 24 April:

$$P\left[0{,}2481 - t_{20}\frac{4}{\sqrt{21}}\sqrt{\frac{0{,}1302}{20}} \leq \theta \leq 0{,}2481 + t_{20}\frac{4}{\sqrt{21}}\sqrt{\frac{0{,}1302}{20}}\right] = 0.95$$

The value of t in tables (1) is $t = 2.086$, then the confidence interval is:

$$P\left[0.2084 \leq \theta \leq 0.2818\right] = 0.95$$

namely, the true value of the average mercury grade of the ore, was included, with a 95% probability between 0.2084 and 0.2818. As the calculated mean for this day was 0.2451, the limit of error was ± 15%.

Similarly for 21st. April:

$$P\left[0{,}2820 - t_{22}\frac{1}{\sqrt{23}}\sqrt{\frac{0{,}0644}{22}} \leq \theta \leq 0{,}2820 + t_{22}\frac{1}{\sqrt{23}}\sqrt{\frac{0{,}0644}{22}}\right] = 0.95$$

$t = 2.074$, then

$$P\left[0.2586 \leq \theta \leq 0.3054\right] = 0.95$$

The limit of error in this case was ± 8.3%.

7.1.2. Haliköy

Using same procedure, we have found:

- Largest limit of error: ± 20.9%

(Analysis made from the 12-18 shift of 7 May to 6-12 shift of 8 May)

- Lowest limit of error: ± 14.4%

(Analysis made from the 12-18 shift of 6 May to the 6-12 s of 7 May).

7.2. Minimum number of analysis to be performed

For a normal distribution and a sufficiently large size of the sampled population the size of the sample is derived – from

---

(1) "t" Tables of "Students" distribution

$$n = \frac{1.96^2 . S^2}{d^2}$$

where:

n= size of the sample

S^2 = populational quasi-variance

d = accepted limit of error

7.2.1. Konya

In this case, we have for the upper limit of error the day 24/IV/72, where

S^2 = 0.0062 (estimated though the sampling variance of the day)

d = 0.1 x 0.2451 (10 per cent of the sampling average of the day)

then, $n = \frac{1.96^2 \times 0.0062}{0.02451^2}$ = 40 analysis

7.2.2. Haliköy

For Haliköy the period with an upper limit of error was that one from the 12-18 shift of 7/V/72 to the 6-12 shift of 8/V/72, where

S^2 = 0.0117 (estimated through the sampling variance of the period).

d = 0.1 x 0.2844 (10 per cent of the sampling average of the period).

$n = \frac{1.96^2 \times 0.0117}{0.02844^2}$ = 56 analysis

7.3. Recommended procedure

The above results clearly show that the number of analysis must be increased both at Konya and Haliköy, up to a daily minimum of 40 and 56 respectively. With such a sample size, the limit of error would be below 10% for a probability coefficient of 0.95 during the days of the tests.

Now, it is very important to realize that if in the future, what is probable, the variance gets higher limits than those above corresponding to the days of the tests, 56 and 40 analysis might be insufficient.

Therefore, it is advisable to calculate every day the true limit of error that the selected number of analysis (56 for Haliköy and 40 for Konya) may introduce in the true mean of the day.

For this, and using the formula

$$n = \frac{1.96^2 \; S^2}{d^2}$$

we shall have for Haliköy.

$n = 56$

S^2 = sampling variance of the day

then, $d^2 = \dfrac{1.96^2 \; S^2}{56} \qquad d = \pm\ 0.2619\ S$

The percent of error will be:

$$\frac{d \times 100}{\bar{x}}$$

Where $\bar{x}$ is the mean of the day.

Similarly, for Konya,

$n = 40$

S^2 = sampling variance of the day

then, $d^2 = \frac{1.96\ S^2}{40}$

$d = \pm\ 0.3099\ S$

and the percent of error:

$$\frac{d \times 100}{\bar{x}}$$

Where $\bar{x}$ is the mean of the day

8. Influence of sampling errors in the calculated efficiencies

The above paragraphe clearly show the paramount importance that sampling procedures have into the calculation of a representative value of the average grade of the ore, and consequently in obtaining a repreeentative calculated efficiency.

As an example, and with the 22 analysis performed on the ore, the 21st April, at Konya, we have, assuming that for the 20th April the calculated mean of 0.262% were the true mean of the day, (See table IV-I of Section IV), the following possibilities:

a)

Day	% Hg	Ore, tons	Hg (content)	Hg (flasked)
20-IV	0.262	52.5	137.55	
21-IV	0.282	137.5	387.75	345

b)

Day	% Hg	Ore, tons	Hg (content)	Hg (flasked)
20-IV	0.262	52.5	137.55	
21-IV	0.2586 (1)	137.5	355.57	345

(1) Lowest limit of ...

c) Day	% Hg	Ore, tons	Hg (content)	Hg (flasked)
20-IV	0.262	52.5	137.55	
21-IV	0.3054 (2)	137.5	420.0	345

(2) Upper limit of grade

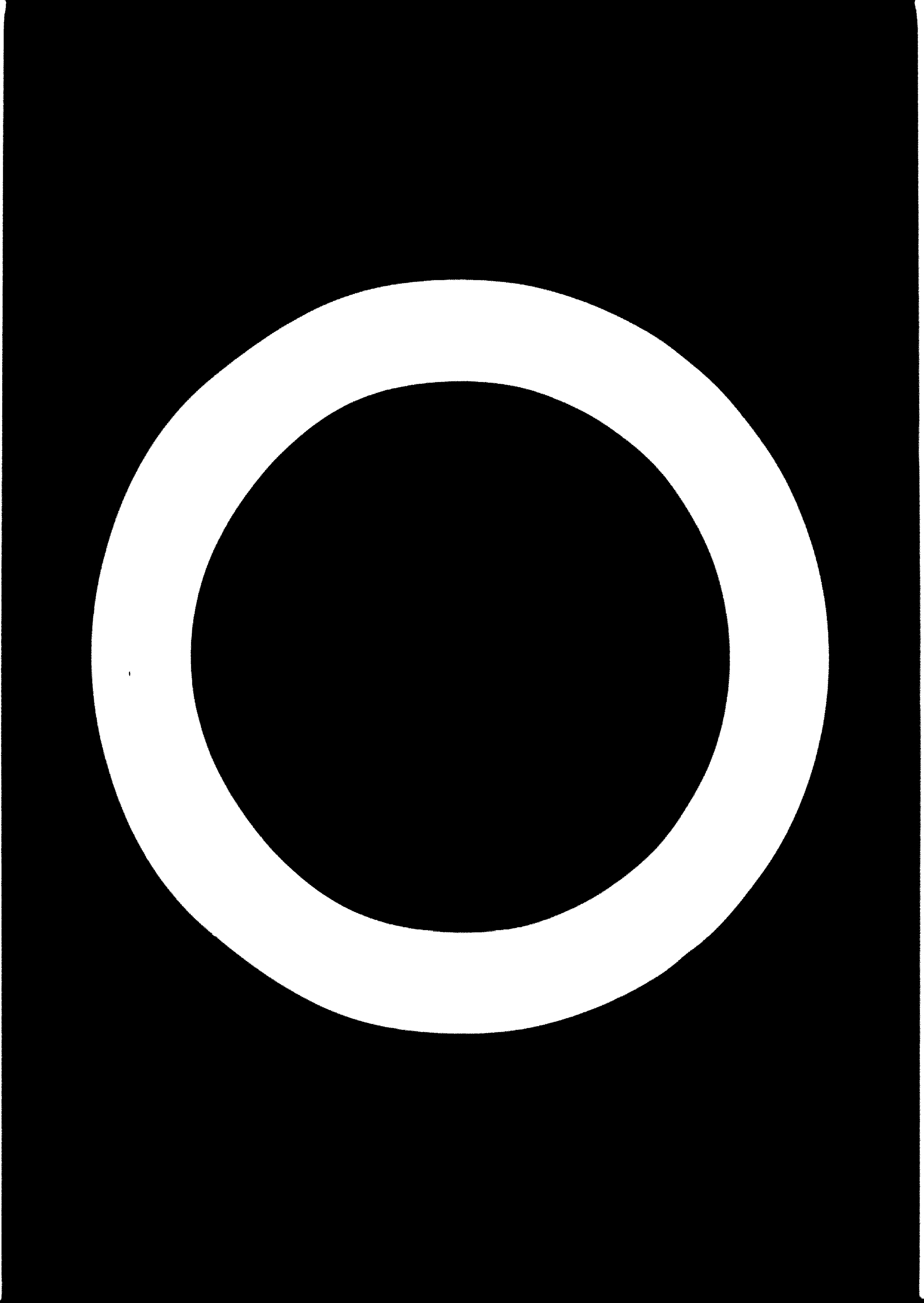
Therefore the range of possible metallurgical efficiencies that day have been:

a) $\text{Effc} = \dfrac{345}{137.55 + 387.75} = \dfrac{345}{525.3} = 65.7\%$

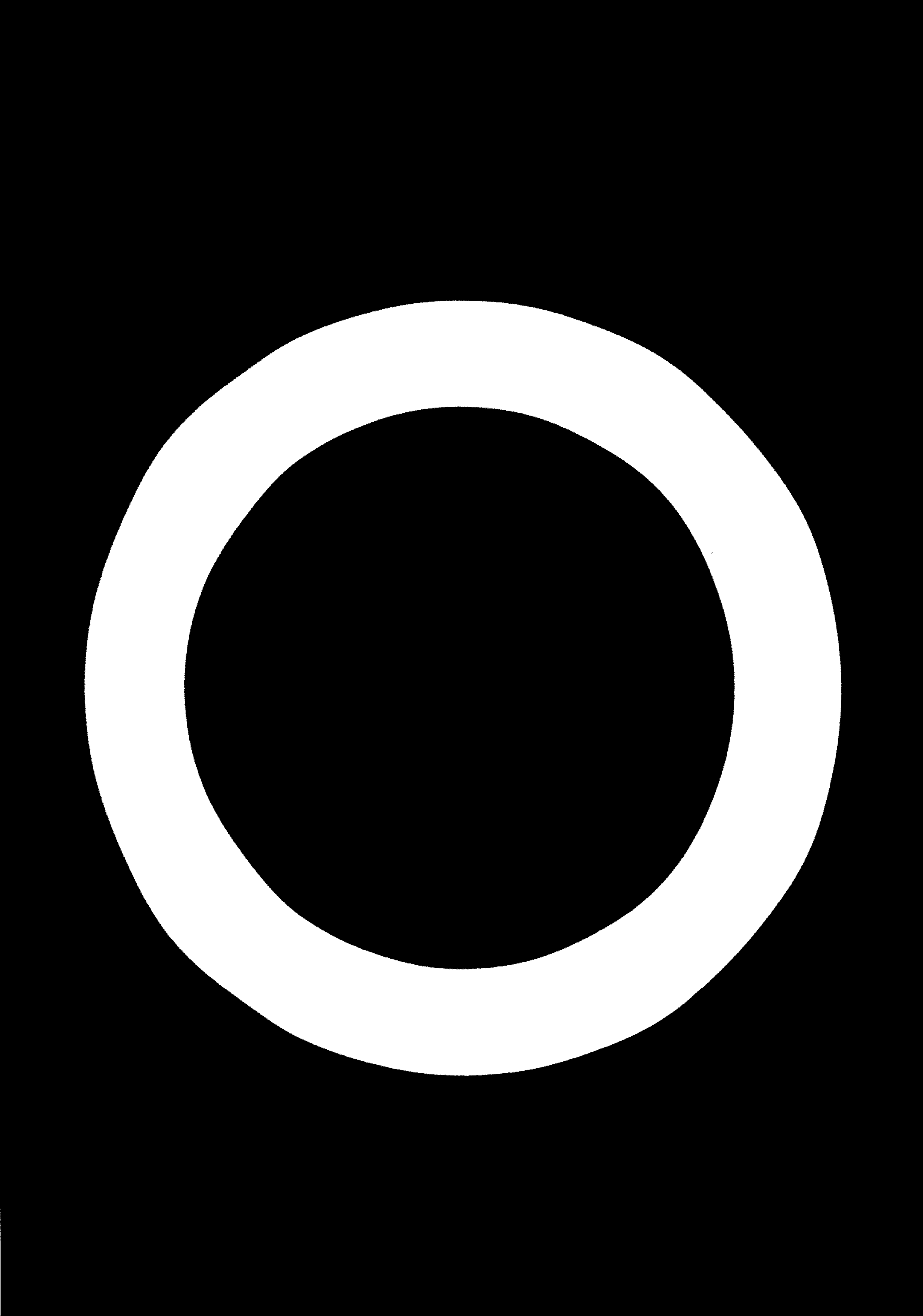
b) $\text{Eff} = \dfrac{345}{137.55 + 355.57} = \dfrac{345}{493.12} = 69.9\%$

c) $\text{Eff} = \dfrac{345}{137.55 + 420.0} = \dfrac{345}{557.55} = 61.9\%$

The above calculatione are self explanatory.



VII TERMINAL SECTION



VII. 1. SUMMARY OF RECOMMENDATIONS

The recommendation included in Section II of this Report could be summarized as follows:

a) Konya

The Konya plant is properly designed from a metallurgical standpoint but its layout makes difficult a proper process control.

Maximum interest should be paid in getting a regular feeding of the kilns, avoiding stoppages and getting good calcined ore.

Feed ore control is of paramount importance, together with the control of other variables of the metallurgical process.

A continuous and total attention from the part of the technical personnel to the above objectives will stimulate the attitude of labor in charge of the several services of the metallurgical plant, and will result in better process efficiency.

Great improvement in mining operations is necessary in order to get a regular supply of ore to the plant.

b) Haliköy

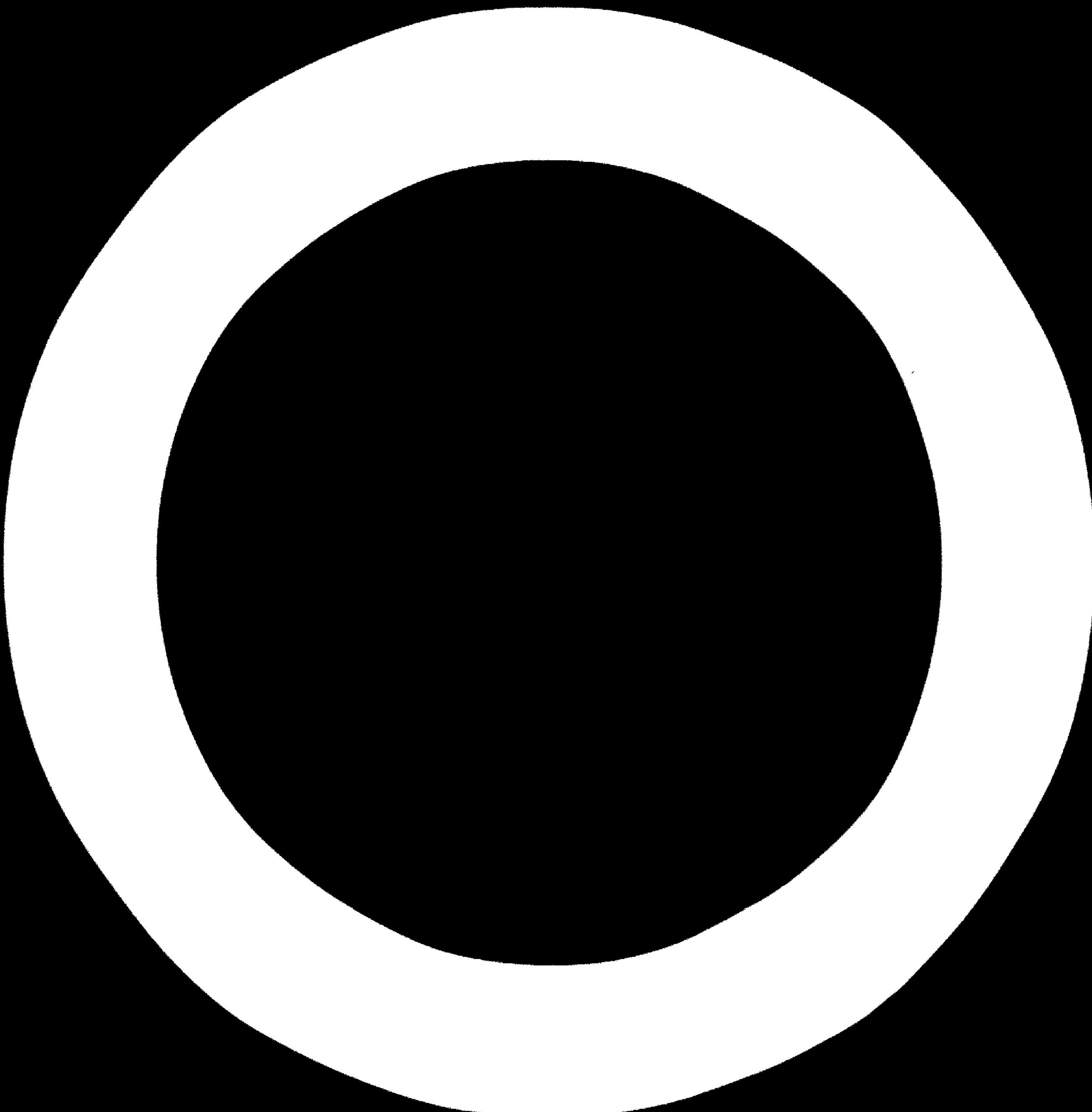
The Haliköy metallurgical plant needs a total reconversion of its present product facilities in order to achieve complete satisfactory working results.

A present situation of the mercury market does not advise high investments into new production facilities, the recommendations included in this Report should be adopted with the criteria to improve the present efficiency of the Haliköy plant with minimum investments.

Improvement of mining operations should be considered in order to prepare the Haliköy orebody for future expansions of the Metallurgical Plant.

VII. 2. REFERENCES AND BIBLIOGRAPHY

(**Note**: **Excluding specific mining references listed in the data cards of Section III,** to avoid duplication)



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