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STUDY ON THE BENEFICIATION AND TECHNOLOGICAL TESTS OF THE OSTUACAN (CHIS.,-MEXICO) BAUXITIC CLAYS

Project No. UD/UC/Mex/86/203
Contract number 87/28

FINAL REPORT

MEXICO CITY / MEXICO
BUDAPEST / HUNGARY
1988
Revised edition

STUDY ON THE BENEFICIATION- AND TECHNOLOGICAL TESTS OF THE OSTUACAN (CHIS.,-MEXICO) BAUXITIC CLAYS

FINAL REPORT

Compiled by

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I N T R O D U C T I O N

According to a contract between UNIDO (Vienna-Austria) and Aluterv-FKI (Budapest-Hungary) the latter has taken on to carry out the beneficiation and technological testing of the lateritic bauxite being the subject of the exploration work of Consejo de Recursos Minerales (Mexico City-Mexico) in order to explore on the basis of an economic analysis the feasibility of establishing a Mexican alumina industry.

Contract number 87/28
UNIDO project No. UD/UC/Mex/86/205
Activity code UD/UC/05/31.8.F

The research program consists of three main parts:

- Sample collecting in Mexico
- Beneficiation and technological tests in Hungary
- Preparation of a Pre-feasibility Study in Mexico

According to point 2.1 of the Contract Aluterv-FKI had to inform both UNIDO and CRM on the results of the laboratory tests carried out in Hungary in 1987 in the form of an Interim Report to be prepared before the end of 1987. This duty was fulfilled in December, 1987.

Further beneficiation and technological tests were carried out partly with the original samples, partly (because some qualities were used up in the 1987 beneficiation tests) with new ones supplied by CRM in early 1988.

Partly due to the temporary lack of samples, partly due to the fact that it took sometime until Consejo de Recursos Minerales finally decided that it would not send its specialist to Hungary to observe the technological tests, the latter were concluded only in April, 1988, about two months behind schedule.

However, even then it was impossible to complete the Draft Final Report in Hungary since it seemed to be very important to include the latest data and experiences on the raw material reserves. Therefore, four Hungarian experts travelled to Mexico with the manuscript of the Draft Final Report prepared to about 80 % (mostly in typed form) partly to complete it during their stay, partly to discuss it in detail with the experts of the Consejo de Recursos Minerales, what they have actually done.

Furthermore, they have worked out the following chapters to be included in the Pre-feasibility Sdudy:

4. Plant size
5. Analisis and definition of the production system
6. Machinery and equipment (mining, beneficiation plant, alumina plant)
7. Auxiliary equipment
9. Production programme
10. Labour (partially available)
11. Investment (on the basis of internacional prices - to be checked by the Mexican side).

The chapters that would be remain to be completed by the Mexican side refer to

1. General background - Introduction
2. Market, costs, prices
3. Raw materials, supplies
8. Land and civil construction
12. Cost of production, depreciation
13. Sales
14. Financial couchisions

According to the Contract (UF/Mex/86/203) the Mexican side obliged to complete the Pre-feasibility study with the help of on ALUTERV-FKI team and UNIDO professional backstopping staff.

I. GENERALITIES

I/A LOCATION AND INFRASTRUCTURE OF THE BAUXITE AREA AND ALUMINA PLANT

Ostuacán area lies in Chiapas state East of the Isthmus of Tehuantepec and North of the range of Sierra Madre del Sur at the Northern latitude of 17°25' - 17°32' and Western longitude of 93°15' - 93°25'.

The zone of prospection is about 16 km long and 7 to 10 km wide, covering 130 km^2 in extension.

Ostuacán is situated 160 km SSE of Villahermosa (capital of Tabasco state with about 260,000 inhabitants). A first class road with an excellent surface and a width of 8 m leads from Villahermosa up to Teapa.

The distance between Teapa and Estación Juárez is 42 km on a 6 m wide 2nd grade asphalted road. Estación Juárez and Ostuacán are connected by a 55 km long road from which 40 km is asphalted, the rest is of Mc'Adam type.

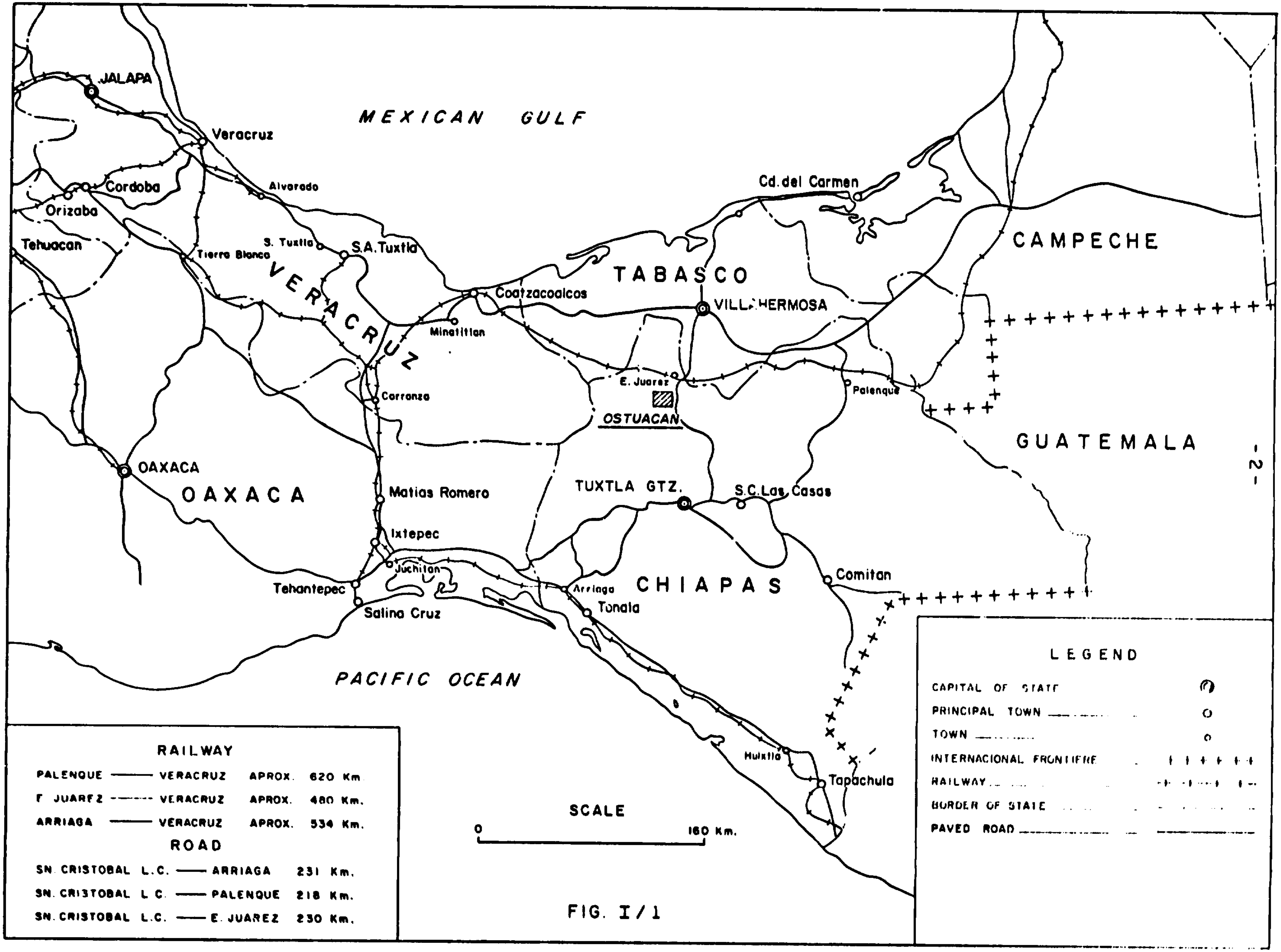
The roads from Estación Juárez up to the bauxite area are in a bad condition, they are dissected by numerous landslides. There are 10 to 12 bridges to cross to reach Ostuacán, many of them are not suitable for the transportation of any industrial material in large quantities. Because of the very uneven topography the road leads through many curves, making the traffic very difficult and dangerous. It takes about one and a half hours to cover the distance of 55 km by car.

The nearest railway station to Ostuacán is at La Crimea (40 km of Ostuacán). The railway runs between Mérida and Coatzacoalcos. Both are important harbours of the Gulf of Mexico (see Fig. I/1).

The recommended location of the alumina and beneficiation plants is shown ing. Fig. I/2.

I/B ENERGY SUPPLY

The area belongs to a very important zone of Mexican oil production. Both fuel oil and petroleum gas are easily available.



MEXICAN GULF
PACIFIC OCEAN
JALAPA
Veracruz
Cordoba
Orizaba
Alvarado
Tehuacan
Tierra Blanca
S. Tuxtla
S.A.Tuxtla
VERACRUZ
Coatzacoalcos
Minatitlan
Carranza
TABASCO
VILLAHERMOSA
Cd.del Carmen
CAMPECHE
E. Juarez
OSTUACAN
Palenque
GUATEMALA
OAXACA
OAXACA
Matias Romero
Ixtepec
Juchitan
Tehantepec
Salina Cruz
TUXTLA GTZ.
S.C.Las Casas
Comitan
CHIAPAS
Arriaga
Tonala
Huixtla
Tapachula
RAILWAY
PALENQUE —— VERACRUZ APROX. 620 Km.
E. JUAREZ ------ VERACRUZ APROX. 480 Km.
ARRIAGA —— VERACRUZ APROX. 534 Km.
ROAD
SN. CRISTOBAL L.C. —— ARRIAGA 231 Km.
SN. CRISTOBAL L.C. —— PALENQUE 218 Km.
SN. CRISTOBAL L.C. —— E. JUAREZ 230 Km.
SCALE
0
160 Km.
LEGEND
CAPITAL OF STATE
PRINCIPAL TOWN
TOWN
INTERNACIONAL FRONTIERE
RAILWAY
BORDER OF STATE
PAVED ROAD

FIG. I/1

LOCATION OF ALUMINA AND BENEFICIATION PLANT

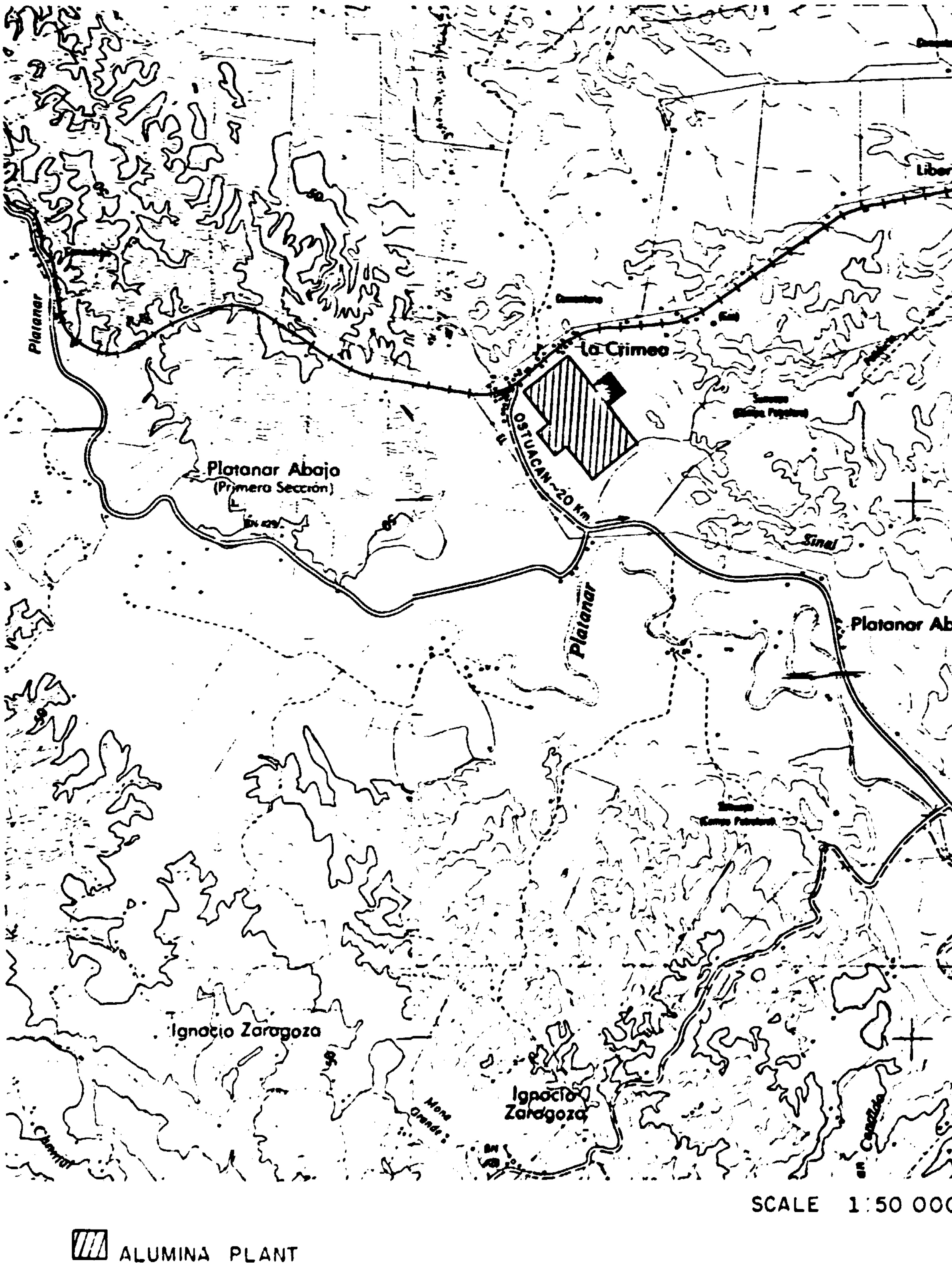


FIG I/2

High tension electric energy can be received from the Peñitas hydro-electric power station situated 10 km from Ostuacán (total capacity 420 MW).

I/C TOPOGRAPHY AND CLIMATOLOGY

As lateritization and bauxitization highly depend on the morphology and as the latter plays an important role in the mining activity and transportation as well, special attention was paid to this question.

The zone of prospection lies on the foothills of Sierra Madre del Sur at altitudes between 200 and 600 m above sea level. The highest point in its vicinity is represented by the Chichonal volcano with an elevation of about 1100 m.

The morphology of the area is characterized by a very dense grid of stream valleys (see Fig. I/3).

According to their shape and extension the hills can be grouped into three main types:

1. almost pefectly rounded single cones, the area of the hill tops not more than 200 to 300 m^2 each,

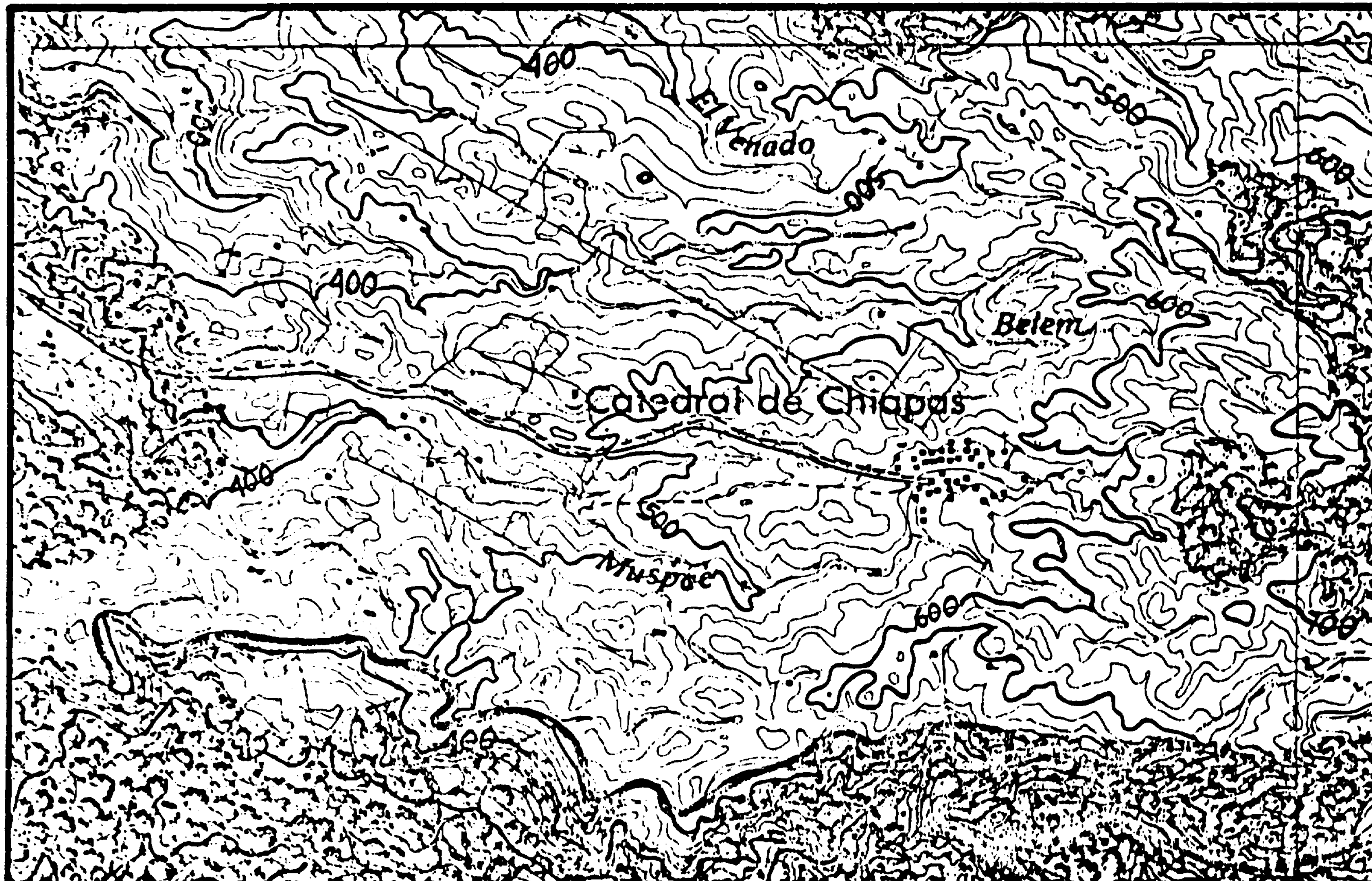
2. elongated single hills (200 to 300 m long, 40 to 60 m wide on the top), representing about 8000 to 20,000 m^2 each,

3. complex Y or V shaped hills with top surfaces of 10,000 to 30,000 m^2 each.

The hills have a relative altitude of 10 to 40 m, are bordered by very steep slopes with grades of 30 to 40° and at least in one direction they have a less steep side with a slope of about 15° (see Figs. I/4 and I/5 and photoes 1 through 3).

The area of prospection belongs to the moist-hot, rainy zone of the tropical belt, where the annual mean temperature is above 22°C and even the mean of the coldest months is above 18°C.

The average of the annual mean precipitation is between 3500 and 4000 mm. The monsoon period begins in June with more than 300 mm rainfall and terminates in November, when the mensual mean is less then 300 mm. The precipitacion culminates in September, when it exceeds 550 mm in average. There are four months only, when the rainfall is 200 mm or less, namely February, March, April and December.



CATEDRAL DE CHIAPAS

DETAIL OF THE OSTUACAN BX. REGION
SHOWING THE TOPOGRAPHY

FIG. I / 3

SCALE 1: 25 000
(ENLARGED FROM 1: 50 000)

GEOLOGICAL SKETCH ON THE CATEDRAL AREA

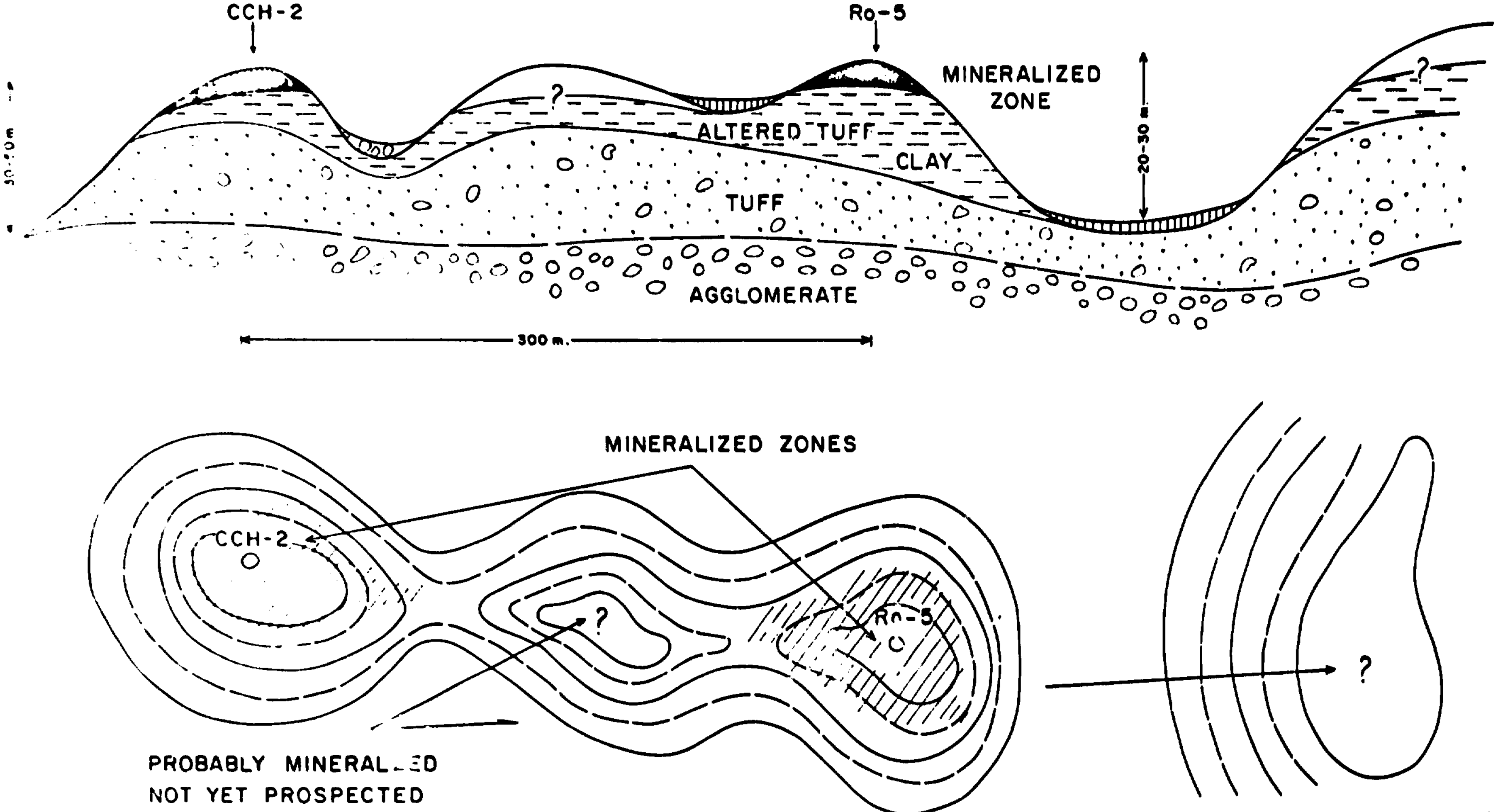


FIG. I/4

DR. KOMLÓSSY
1988

Photo 1

Photo 2

Photo 3

I/D GEOLOGY

General geological make up

The OSTUACAN bauxite region is situated on the Northeastern flank of the SIERRA de CHIAPAS. The oldest formation exposed on the surface consists of Tertinary detrital sediments (Eocene sandy gravels, silt-stone, Oligocene gravelly sand, Miocene clayey sand, etc.). These marine sediments are covered by a very Recent volcanic blanket of andesitic pyroclasts.

The volcanic series is some 40 to 60 m in thickness. Its lower part is built up mainly by agglomerate, while the upper part by tuffs. In the later some boulders of volcanic rocks can be found. The horizon of the soft and porous volcanic tuffs is highly altered; lateritized and at places bauxitized.

The older and main structural lines rank along NW-SE directions, the subordinate and younger ones are in N-S and E-W directions, determing the orientation of the dense grid of valleys and of the elongated hills.

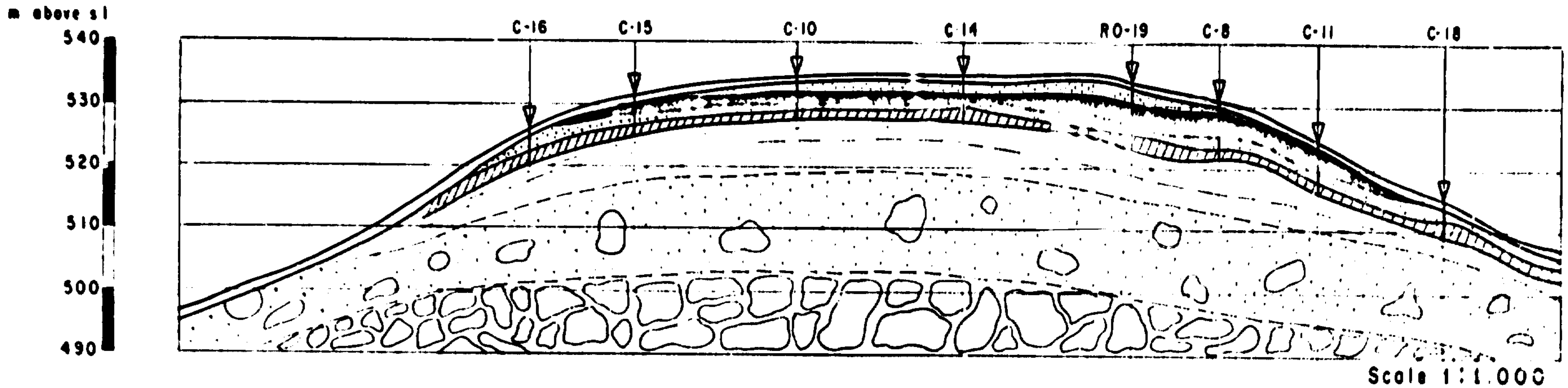
Geology of the bauxitic laterite

The lateritization is controlled by the drainage conditions of the parent rock, i. e. by the morphology and by the extreme high porosity of the young volcanic tuff. As it was pointed out before, the horizontal extension of the bauxitized zone is restricted to the hill tops.

The total lateritized horizon is between 10 to 15m in thickness. The best leached zone of the lateritic profile is generally situated at two to five m below the surface as it is shown in Figs. 5 and 6.

As far as lithological characteristics are concerned the laterite is an earth-like, loose, porous soil. Its colour varies among reddish-yellow, yellowish red and brownish red. During the lateritization on the hill tops the bauxitization has commenced as well. In the course of this process gibbsite nodules with a max. diameter of 1.5 cm have been formed. These form relatively harder concentrates in the kaoline-rich soil. This physical cha-

CROSS SECTION ON C. DE CHIAPAS DEPOSIT



LEGEND

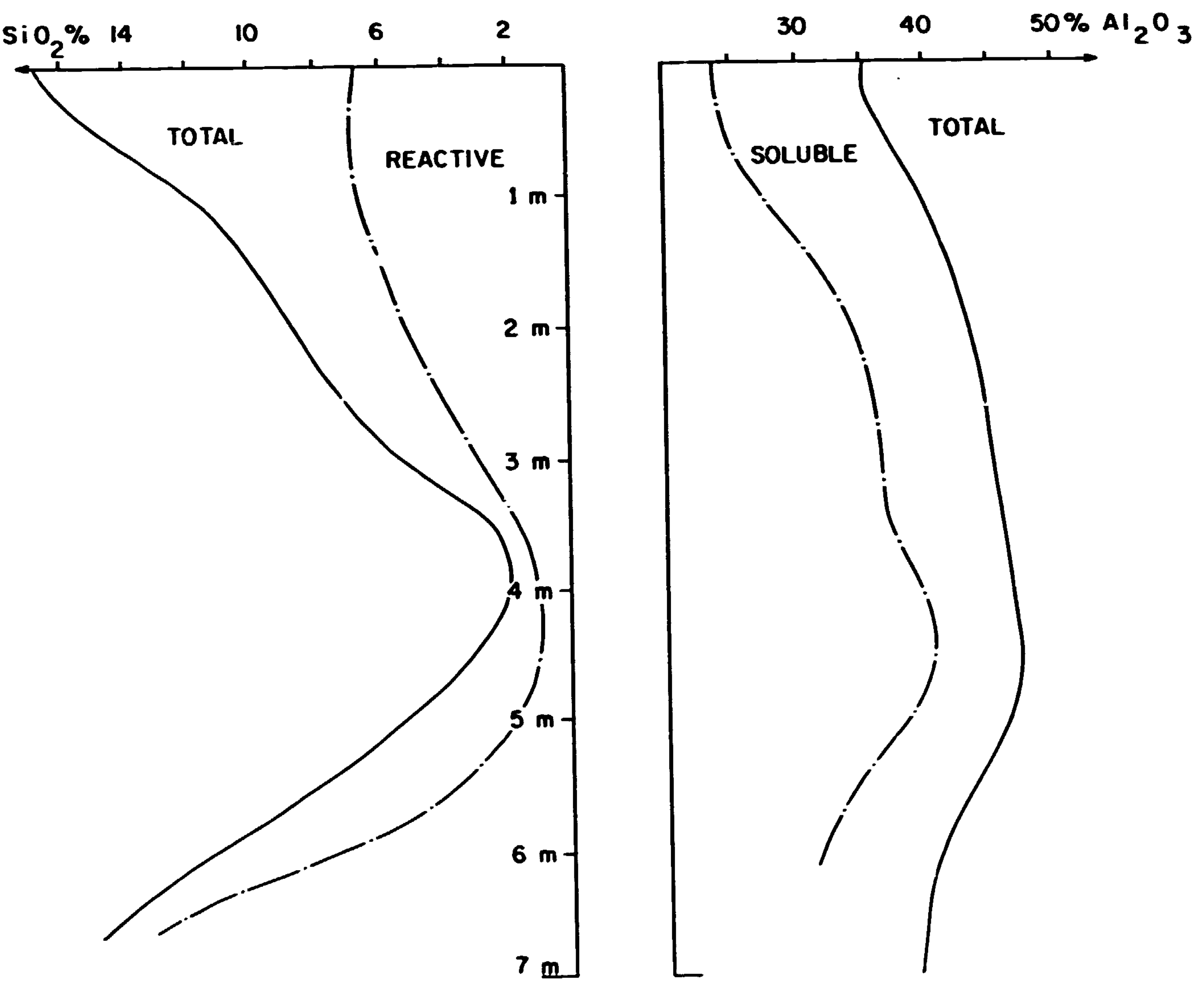
- Top soil
- Bayer grade bauxite (CPT-1-2).
- Bauxitic laterite (~CPT-3)
- Clay
- Altered tuff
- Tuff with volcanic boulders
- Agglomerate
- Bore hole with number

FIG. I/5

Compiled by: A. Morales 1988
CRM. Geologist

Ro-78

GEOCHEMICAL SECTION OF THE LATERITIZED ZONE



DR. KOMLÓSSY
1988

FIG. 1/6

racteristic of the bauxitic laterite makes this formation suitable for beneficiation. This primary quality of the bauxitic laterite depends on the amount of the harder gibbsitic nodules, i.e. on the proceeding of the bauxitization.

Chemical and mineralogical make up of the raw material and classification

As it is shown in Fig. 6 a gradual change of the alumina and silica contents can be observed in the lateritized profile. The best leached part of the profile - where the extractible Al_2O_3 (AA) is more than 30% and reactive SiO_2 (RS) is less than 7% - can be considered as "bauxite" - not in the terms of petrology, as it is not a lithificated rock, but in the terms of the alumina industry, as it may be a commercial grade ore. Naturally this does not mean that a material just complying with these conditions could be commercially processed in a profitable way, only that we calculate with it as a potential raw material and we will include it into the reserve if it does not hurt (excessively) the profitability of alumina manufacturing.

Artificially the "bauxite" layer can be divided into two parts:

1. good quality bauxite, where AA $\geq$ 30% and RS $\leq$ 5% (however, as an average AA $\geq$ 35%)
2. marginal quality bauxite, where AA $\geq$ 30% and RS is between 5 and 7%.

These are the CPT-1 and CPT-2 types, respectively. See details in Chapter I/F.

Out of the 440 bore-holes, which were drilled up to now in the area, 110 penetrated "bauxite". On the basis of the analyses carried out by CRM's laboratory in Tuxtla Gutierrez, the most frequent AA values are between 33 and 38% and the maximum exceeds 40%. The most frequent RS values are between 4 and 5%, the average is about 5%, the minimum is 1.5%.

For the total Al_2O_3 (TA) and total SiO_2 (TS) contents much less data are available. For the commercial average of the deposits TA is probably between 42 and 45%, whereas TS is probably between 7 and 8%.

The amount of total Fe_2O_3 is between 18 and 20% in the less thoroughly leached upper and lower horizons, whereas it exceeds 25% (in some cases even approaches 30%) in the best leached intervals. The average of the CPT-1 and CPT-2 type ores of the deposit is about 23%.

The TiO_2 content is less than 2% in the upper horizons of the laterite profile, whereas the maximum values exceed 3%. The average of the commercial ore may be 2.7 to 2.9%.

LOI may be between 23 and 25%; in the best quality intervals LOI values exceed 25%.

In the immediate overburden and bedrock of the commercial grade bauxite Al-rich laterites can be found, in which the "bauxite" layer is "sandwiched" in. This Al-rich laterite can not be processed economically by the [illegible] process, as its AA content is generally less than 30% and its RS conte[illegible]s more than 7%. This part of the laterite profile can be utilized by using BENEFICIATION.

As the quantity of the commercial grade ore seems to be limited at the OSTUACAN occurrence, this part of the laterite section was also taken into consideration as raw material of the alumina plant.

That part of the laterite section is considered as <u>beneficiation grade material</u> in which the AA $\geq$ 25% and RS $\sim$ 11% (however, the average of AA should not be less than 28%).

This is the CPT-3 type material. (See Chapter I/F).

The 14 component chemical composition of the original and beneficiated Ostuacan samples is given in Table III/1.

<u>Mineralogical Analysis</u> has only been made on composite samples collected in 87 and 88 and on beneficiated concentrates. They are considered as characteristic samples. (See Chapter I/F).

I/E PROSPECTION AND RESERVES

Up to now some 440 bore holes were deepened on the area of 130 km^2 from which 110 can be regarded as productive ones containing CPT-1 and CPT-2 grade bauxite. Some other 150 penetrated CPT-3 beneficiation grade bauxitic laterite.

At first a regular prospection grid of 500 x 500 m was applied, however it has been learnt that only the hilltops are worth prospecting. On the other hand every hill has its own characteristic degree of bauxitization. Therefore, all hilltops should be prospected during the reconnaissance even if they are closer to each other than 500 m. As the CRM could not terminate the prospection of the Ostuacan occurrance even on a reconnaissance level, the estimation of the reserves is not sufficiently reliable, moreover, the bauxite deposits should be prospected on a much more detailed preliminary level, so that the reliability of the prospection (and reserve estimation) should satisfy the demand of any industrial decision.

Up to April, 1988 only five deposits were prospected on a preliminary level, from which three contain CPT-1 and CPT-2 grade bauxite. These are:

- Herradura,
- Catedral de Chiapas and
- Hidalgo

According to a rough estimation the reserves and their quality are displayed in Table I/2. The reserves can be regarded as proven ones with ± 20% of reliability. With the probable reserves (± 30%) the total amount may be 50% more. The calculation has been made on dry basis using a 114 $^t/m^3$ value for the ore content of 1 m^3 volume.

Table I/2

	Herradura kt	AA %	RS %	C. de Chiapas kt	AA %	RS %	Hidalgo kt	AA %	RS%	Total kt	AA %	RS%
CPT-1	157.6	37.2	3.0	57.0	33.5	3.3	11.4	33.9	4.3	226.0	36.1	3.1
CPT-2	65.5	32.8	6.7	28.5	31.1	5.7	37.0	31.7	6.4	131.0	32.1	6.4
CPT-1-2	223.1	35.9	4.1	85.5	32.7	4.1	48.4	32.2	5.9	357.0	34.6	4.3
CPT-3	172.0	28.6	10.3	69.8	27.5	7.9	87.2	29.7	8.9	329.0	28.6	9.4

A total of 0.7 - 0.8 mil. to CPT-1-2 bauxite can be taken into account in the OSTUACAN area as probable reserve, from which about 0.6 mil. tons can satisfy the CPT-1 quality requirements (AA $\geq$ 35% and RS $\leq$ 5%). As far as the quantity is concerned, at least 6.0 million tons of CPT-1 bauxite is required for establishing even a 100,000 t/y capacity alumina plant. (See Chapter I/G).

As far as the preliminary (and detailed) prospection is concerned, as the lateritization is very strictly controlled by morphology, a very special method was proposed to the CRM's staft in 1987 and has been applied ever since. The suggested prospection pattern for one hill is presented in Fig. I/7. The distance between two points depends on the slope of the hill top.

It has to be mentioned that no topographical maps of suitable scale are at CRM's disposal. Only toposheets of 1:50,000 scale or the enlarged varieties (1:25,000) are used. At least 1:10,000, but preferably 1:5,000 scale contour maps are needed to carry out reliable geological prospections. For that reason surveying of the productive hills and their vicinity is necessary and the most urgent task of bauxite prospection.

The manual equipment used by CRM does not satisfy the demand of prospection because in many cases the productive zone has not been penetrated. In the case of Herradura more than 30% of the drillings were stoped in the

PLAN OF THE DETAILED PROSPECTION
OF THE HILL TOP OF CH-2

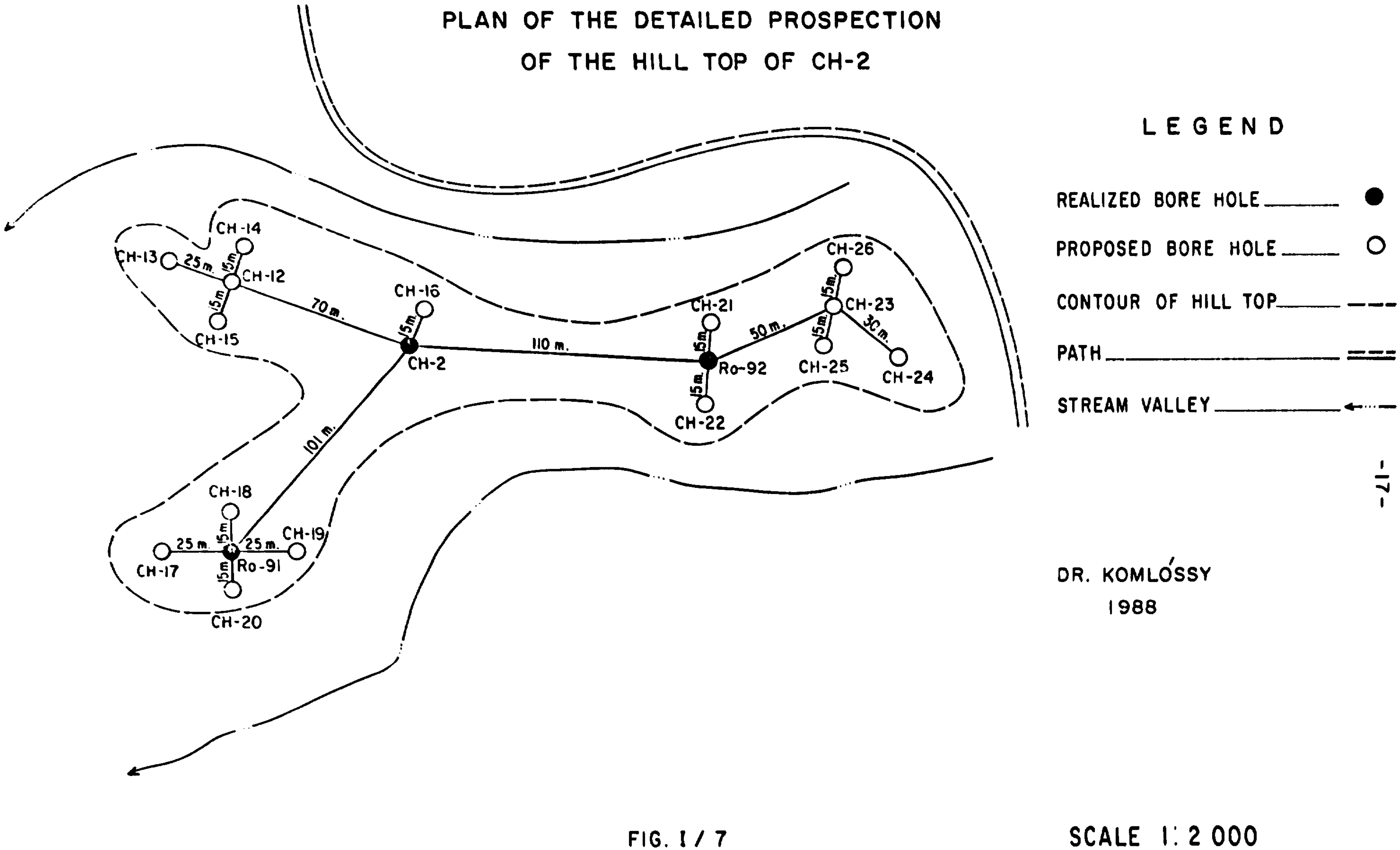


FIG. I / 7

bauxitic zone, as the maximum capacity of the manual equipment is 7 m. Since the wall of the bore-holes is not sufficiently stable and a deterioration of the samples taken by manual equipment can not be avoided, mechanized rotary coreing machines are advised to be used at the OSTUACAN bauxite prospection with a capacity of 30 m and a core diameter of 55 mm as a minimum.

I/F CHARACTERISTIC SAMPLES

In the Techno-economic Study for the Commercial Extraction of Alumina from Bauxitic Clays (Interim Report, 1987) Aluterv-FKI gave a detailed description, how the characteristic samples were taken and pointed out, that no representative sample can be collected from a deposit when it is not adequately prospected.

In 1987, the samples were selected on the basis of the AA and RS analyses of the bauxitic laterites, carried out by the Tuxtla Gutierrez laboratory of CRM.

CPT-1 has been collected for the case if the exploration found large amounts of bauxite of a quality better than the present average.

CPT-2 is a raw material of a submarginal quality and slightly inferior to the presently known average. This sample was collected for the case if the explorations can not identify sufficient amounts of good material and so the lower quality layers of the laterite profile would also have to be included into the production.

CPT-3 does not satisfy the quality requirements of the bauxite considered to be of commercial grade. It has been collected for the purpose of beneficiation tests.

Each sample is a composite sample blended from 6 to 8 bore-holes.

For continuing the beneficiation tests in Hungary new samples were were requested from CRM in 1988 called CPT-1/88, CPT-2/88 and CPT-3/88. These samples were collected according to the previously applied principles. In their chemical composition the samples taken in 1987 and '88 are not absolutely identical, but their differences are acceptable. The chemical composition of the characteristic samples is displayed in Table III/1 and their mineralogical make-up in Table III/2.

Mineralogical analysis has only been made on composite samples collected in 87 and 88 and on beneficiated concentrates. They are considered as characteristic samples. (See Chapter I/G).

According to the results of the quantitative XRD analyses carried out in Aluterv-FKI the main Al_2O_3 bearing mineral is gibbsite. Its amount is between 55.8 and 45.3% in the 7 investigated natural samples and about 50% in the average of the commercial grade ore (CPT-1-2/87 and 88). (See details in Table I/1). The amount of kaolinite can be calculated from the percentage of RS, it is between 10.1 and 22.5%. Among the analyzed natural samples CPT-2/88 contains the largest amount of quartz (3.4%), however, it must be mentioned, that its quantity may theoretically reach even 8% in the upper laterite horizons. The predominant iron mineral is goethite, while hematite plays a subordinate role. The XRD analyses indicate remarkably high amounts of magnetite. The values of 6.2 to 7.8% are strikingly high. It has to be mentioned that XRD graphs have also indicated 0.5 to 1.6% diaspore.

It can also be seen in Table I/1 that the total amounts of the detected minerals is between 93.9 and 96.9%. The missing material may be partly the residual alkali and earthalkali content of incompletely altered feldspars, partly some analyzed and not analyzed contaminants. The latter usually do not disturb the processing of the ore.

Table I/1

MINERALOGICAL COMPOSITION OF OSTUACAN BAUXITES-LATERITES AND BENEFICIATED CONCENTRATES

No. of samples / minerals in %	CPT-1/87	CPT-2/87	CPT-3/87	CPT-1/88	CPT-2/88	CPT-3/88	CPT-123	CPT-2D	CPT-3D
gibbsite	55.8	48.3	48.0	50.6	45.4	45.3	49.4	50.5	58.0
diaspore	0.5	1.6	0.9	0.5	1.4	0.7	1.2	1.4	0.7
kaolinite	10.1	16.6	22.5	13.7	20.5	22.5	16.4	14.6	15.7
quartz	0.4	3.3	1.7	0.7	3.4	1.9	1.9	5.8	1.9
goethite + alumogoethite	12.1	9.8	9.7	12.0	9.7	9.4	10.4	9.8	7.2
hematite	7.3	7.0	4.7	7.4	5.2	5.4	6.1	6.5	6.4
magnetite	7.8	6.4	6.8	7.6	6.2	6.9	7.0	5.3	4.8
anatase	2.9	2.3	2.2	2.6	2.1	2.0	2.2	2.5	2.0
total:	96.9	95.3	96.5	95.1	93.9	94.1	94.6	96.4	96.7

D means beneficiated concentrate

I/G MINING AND TRANSPORTATION

Taking into consideration that the area is still not explored, the quantity and quality of the later possibly available bauxite are not known yet. Neither are the relations between and the relative locations of the orebodies known, nor the topographical maps required for the designing of the mines available.

The following rough estimate is based on the assumption that after carrying out the necessary exploration work some 6 million tons of good quality bauxite (Av. $Al_2O_3 \geq 35\%$; Re. $SiO_2 \leq 5\%$) will be available in the area. This could support a 100,000 tpy alumina plant. In the case that this assumption comes true, sufficient amounts of submarginal (Av. $Al_2O_3 \geq 30\%$; Re. $SiO_2 \leq 7\%$) ore or beneficiable low quality (Av. $Al_2O_3 \geq 28\%$; Re. $SiO_2 \sim 11\%$) raw material would be available (7 or 12 million tons, respectively) to design the alumina plant for a capacity of 200,000 tpy. In the first case no beneficiation would be required, in the second one a beneficiation plant would have to be set up.

According to the technological tests carried out on natural samples (CPT-1; CPT-2) and on a beneficiated product (CPT-3D) the amounts of alumina, which can be produced of one (dry) ton of raw material are the following: (for details see Chapter IV)

CPT-1 (good) quality:	335 Kg.
CPT-2 (submarginal) quality:	286 Kg.
CPT-3D (beneficiated low) quality:	341 Kg.

On the basis of the above figures the following table shows the raw material requirements for a twenty year supply of an alumina plant (dry basis):

QUALITY	Av. Al_2O_3 %	Re. SiO_2 %	100,000 TPY (Variant A)	200,000 TPY (Variant B)	(Variant C)
Good (CPT-1)	55.5	4.6	6,000,000 t	6,000,000 t	6,000,000 t
Submarginal (CPT-2)	50.4	6.9	-	7,000,000 t	-
Beneficiated (CPT-3D)	55.9	6.7	-	-	5,900,000 t
(Low quality (CPT-3)	28.0	10.6	-	-	12,000,000 t)
Total to be processed	-	-	6,000,000 t	13,000,000 t	11,900,000 t
(Total to be mined	-	-	6,000,000 t	13,000,000 t	18,000,000 t)

The bauxite demand of the alumina plant and of the possible beneficiation plant would be fulfilled by operating the mines in two shifts for 250 days per year. (Some 50 days of production are expected to be lost during the rainy season.)

Variant A: For a 100,000 tpy alumina plant processing CPT-1 ore this is:
- 300,000 tpy ore (dry basis),
- 1200 tpd ore (dry basis),
- 2000 tpd ore, calculated with a yearly average of 40% adhesive moisture (wet basis).

Variant B: For a 200,000 tpy alumina plant processing a 6:7 blend of CPT-1 and CPT-2 quality ores this is:
- 650,000 tpy ore (dry basis),
- 2600 tpd ore (dry basis),
- 4333 tpd ore (wet basis).

Variant C: For a 200,000 tpy alumina plant processing a 1:1 blend of CPT-1 and CPT-3D quality ores this is:
- 900,000 tpy ore (dry basis),
- 3600 tpd ore (dry basis),
- 6000 tpd ore (wet basis).

During the rainy season the mass of the wet ore may amount to 2400 tpd

(Variant A), 5200 tpd (Variant B) and 7200 tpd (Variant C), respectively, calculating with a maximum adhesive moisture content of 50% (wet basis). This extra transportation requirement may be compensated by concentrating the mining operations to the area closest to the alumina plant for the rainy season and by transporting the ore from the largest distances during the driest months of the year, when its moisture content would be the lowest.

Mining Activity and Prospection

The mining activity would be a very simple open pitting. The area would be prepared by the removal of the vegetation and of 0.5 to 1 m top soil. This would be followed by the mining prospection involving a mechanical core drilling along a 10 x 10 m grid (using water flushing). The following number of boreholes would have to be drilled every day:

- Variant A: 5 to 15 boreholes (calculating with a typical thickness of 1 to 3 m for CPT-1 quality),
- Variant B: 5 to 15 boreholes (calculating with a typical thickness of 2 to 6 m CPT-1 plus CPT-2 quality),
- Variant C: 7 to 20 boreholes (calculating with a typical thickness of 2 to 6 m for CPT-1 and CPT-3 qualities).

The number of drilling machines has been calculated according to the maximum number of boreholes to be drilled (see Chapter V/A).

The raw material is loose, easily exploitable, it could be mined by a simple loading machine. The expected average stripping ratios are 1.5 t/t for Variant A and 0.5 t/t for Variant B. In case of Variant C the stripping would be negligible.

There would be no need to transport the overburden, since it could be pushed by CAT-988 machines into the valleys separating the hills utilizing the opportunities offered by the morphology of the area.

Besides the CAT-988 smaller machines (CAT-950) have also been taken into consideration for loading (see Chapter V/A).

Raw material transportation

The bauxite would be supplied to the alumina plant (and to the beneficiation plant as well in the case of Variant C) immediately from the mining area, where it would be loaded by front end loaders into 30 t dumper trucks.

The alumina (and beneficiation) plant(s) would be located near La Crimea. Taking into consideration that it is prohibited to transport mining products on public roads, a road suitable for heavy dumper trucks should be constructed. The road could be designed only after exploring the area, so the distances can only be estimated.

An average of 25 km distance has been calculated with, of which 20 km would fall on a secondary road and 5 km on service roads.

Calculation of the transportation time:

1.	Shift change, refueling	15 min.
2.	Trip to the pit (average speed 20 km/h	15 min.
3.	Positioning for loading	2 min.
4.	Loading	15 min.
5.	Trip to the alumina (and beneficiation) plant(s) and back (speed on service roads: 10 km/h; speed on secondary road (loaded): 20 km/h; speed on secondary road (unloaded): 30 km/h)	160 min.
6.	Unloading at the plant(s)	3 min.
	Subtotal of items 3 through 6	180 min.

By calculating with two shifts (8 hours each) an active time of 13 hours per day has been taken into consideration. This does not include the time for refueling which would take place at a petrol station to be set up along the main secondary transportation road.

According to these calculations

(780 min - 15 min): 180 min = 4.25 turn arounds per day (two shifts)

can be calculated with. For safety's sake 4 turn-arounds per day have been taken into consideration. This way a 30 ± dumper truck may haul an average of 120 tons of wet ore per day. The following number of dumper trucks are required for the different variants:

Variant A:	2000 tpd:	120	=	17 trucks
Variant B:	4333 tpd:	120	=	36 trucks
Variant C:	6000 tpd:	120	=	50 trucks

10% should be added to these figures as spares (See Chapter V/A).

II. BENEFICIATION

In 1987 Aluterv-FKI carried out the basic beneficiation test series, the results of which were compiled in the "Techno-Economic Study for the Commercial Extraction of Alumina from Bauxitic Clays-Interim Report".

The test series were carried out using both the CPT-2 and the CPT-3 samples. The conclusion could be drawn that though Sample CPT-2 could be beneficiated, however, the losses were so high that presently it should not be taken into consideration as a viable alternative. However, Sample CPT-3 has given encourageing results. The best results were attained after 20 minutes soaking in a pH = 12 solution of 1 g/l sodium pyrophosphate:

AA was increased from 33.19% to 44.72%,
RS was reduced from 10.43% to 5.47%

with a mass extraction of 62% in the + 25 μm concentrate. This way a commercial grade raw material could be won. The detailed results can be found in Chapter 5.212 of the above Interim Report.

All the techno-economic figures of the present Report concerning beneficiation and the processing of beneficiated material are based on the 1988 beneficiation tests carried out on Sample CPT-3/88.

II/A. BENEFICIATION TESTS CARRIED OUT IN 1988

First of all control beneficiation tests have been carried out with the samples received in 1988 and corrrresponding in their composition more or less to those of 1987. Subsequently larger amounts of concentrates have been produced for the digestion tests.

In the course of the control tests it has been attempted to find the optimum concentration of the dispersing and peptizing reagents in order to reduce the consumption of these chemicals. The tests have shown that the previously used 1 g/l sodium hexametaphosphate concentration is the optimum one. Using a lower concentration no significant beneficiation could be attained, whereas at higher concentrations the amount of the concentrate has been reduced to such an extent that the slight improvement of the product quality could not compensate the loss of available alumina. The results of these tests are shown in Table II/1.

Table II/1

Variation of the composition of the +500 μm fraction as a function of the sodium hexametaphosphate concentration

SHMP conc. g/l	Mass recovery of the + 500 μm fraction, %	Reactive SiO_2 content %	Available Al_2O_3 content %	Recovery of the available Al_2O_3 content, %
0	45.1	9.4	35.7	48.6
1	44.3	5.7	43.1	59.9
2	37.5	5.3	43.9	51.9
4	33.6	5.0	44.6	46.9

If comparing the benefitiation results of samples CPT-3/88 (see Table II/2) and CPT-2/88 (see Table II/3) attained at the concentration of 1 g/l SHMP with those of 1987 (see Techno-Economic Study for the Commercial Extraction of Alumina from Bauxitic Clays, Interim Report) some differences may be observed, specially in the case of sample CPT-2/88. These may be due to two reasons. One of these is that the two samples are not identical (this is most apparent in the case of samples CPT-2/87 and CPT-2/88). The other reason is that this simple beneficiation method is very sensitive to the state of the sample. A basic precondition of a successful beneficiation is that the sample should be processed immediately after mining, in its original (run of mine with natural moisture content) state. The more the ore dries, the poorer the results, because some changes occur in its crystal structure at the contact with the air, which can not be reversed by subsequent remoisturization.

On the basis of this year's test results sample CPT-3/88 can be advantageously beneficiated, but CPT-2/88 only moderately. In the case of the latter the chemical composition improves just slightly, only the availa-

Table II/2

Beneficiation results of sample CPT-3/88

Agitation time: 20 min; 1 g/l sodium hexa-meta-phosphate + 1 g/l NaOH, giving a pH of about 12.

Sample	Mass recovery	Composition							
		Total Al_2O_3	Total SiO_2	Fe_2O_3	TiO_2	SiO_2 as quartz	Reactive SiO_2	Available Al_2O_3	Av.Al_2O_3 to R.SiO_2 ratio
	%	%	%	%	%	%	%	%	
Feed CPT-3/88		40.90	12.40	18.90	2.30	1.86	10.54	31.94	3.03
Small sample:									
+ 500 µm	44.34	48.00	6.72	13.80	-	1.03	5.69	43.16	7.59
250-500 µm	4.71	36.40	12.00	23.50	-	2.84	9.16	28.61	3.12
125-500 µm	5.36	31.40	9.10	33.10	-	3.80	5.30	26.90	5.07
63-125 µm	3.28	33.40	9.66	30.80	-	3.70	5.96	28.33	4.75
25- 63 µm	2.35	34.80	11.56	27.30	-	4.25	7.31	28.59	3.91
- 25 µm	39.96	35.80	19.47	20.64	-	2.11	17.36	21.04	1.21
	100.-								
From these:									
Concentrate (+63 µm)	57.69	44.68	7.54	17.35	-	1.59	5.95	39.62	6.66
Reject (-63 µm)	42.31	35.75	19.03	21.02	-	2.23	16.80	21.47	1.28
	100.-								
From the large sample:									
Concentrate (+63 µm)	48.06	45.20	9.20	17.30	2.00	1.67	7.53	38.80	5.15
Reject (-63 µm)	51.94	36.92	15.36	22.03	2.79	2.04	13.32	25.60	1.92
	100.-								

Table II/3

Beneficiation results of sample CPT-2/88

Agitation time: 20 min; 1 g/l sodium hexa.meta-phosphate + 1 g/l NaOH, giving a pH of about 12.

Sample	Mass recovery %	Composition: Total Al_2O_3 %	Total SiO_2 %	Fe_2O_3 %	TiO_2 %	SiO_2 as quartz %	Reactive SiO_2 %	Available Al_2O_3 %	Av.Al_2O_3 to R.SiO_2 ratio
Feed CPT-2/88		41.00	12.86	18.23	2.40	3.39	9.47	32.95	[illegible]
Small sample:									
+ 500 µm	23.38	44.80	12.20	14.80	1.80	3.87	8.33	37.72	4.53
250-500 µm	6.28	40.50	13.00	18.60	2.20	5.90	7.10	34.47	4.85
125-250 µm	6.10	36.70	10.00	26.60	3.10	5.34	4.66	32.74	7.03
63-125 µm	5.21	33.90	9.90	29.20	3.70	6.86	3.04	31.32	10.30
25- 63 µm	5.03	36.60	14.70	23.00	2.70	8.76	5.94	31.55	5.31
-25 µm	54.00	40.99	13.57	17.22	2.45	1.83	11.74	31.01	2.64
	100.-								
From these:									
Concentrate (+63 µm)	40.97	41.55	11.70	18.97	2.30	4.78	6.92	35.67	5.15
Reject (-63 µm)	59.03	40.62	13.66	17.72	2.47	2.42	11.24	31.07	2.76
	100.-								
From the large sample:									
Concentrate (+63 µm)	45.14	42.00	12.63	18.70	2.50	5.81	6.82	36.20	5.31
Reject (-63 µm)	54.86	40.18	13.05	17.84	2.32	1.40	11.65	30.28	2.60
	100.-								

ble Al_2O_3 and the reactive SiO_2 contents show significant improvements.

During the production of the larger concentrate samples to be used for digestion tests a cut-off size of 63 µm has been selected in order to improve the utilization of the raw material. The results of these tests are also shown in Tables 2 and 3. The benefitiation results of the small and large samples are more or less similar in the case of sample CPT-2/88 (the results attained with the large sample are even slightly better), however, in the case of sample CPT-3/88 both the mass recovery and the chemical composition of the concentrate are poorer with the large sample. This can only be explained by the modified beneficiation characteristics of the sample dried out in the meantime. As a result of this the sample became much more sensitive to the modified screening technics applied for the beneficiation of the larger sample.

II/B. DESCRIPTION OF THE SUGGESTED BENEFICIATION PROCESS

The task of the ore beneficiation plant unit would be to increase the amount of available acceptable quality ore in accordance with variant C (see Chapter IV for details) by a relatively simple and inexpensive procedure. A process including a soaking and a washing step could be advantageously applied for the soft, lateritic Ostuacán bauxites. The essence of this process is the loosening of the clayey matrix of the ore by some dispersing agent under gentle agitation in a medium dense slurry. (The dispersing agent would preferably be 1 g/l sodium hexametaphosphate or about 1.4 g/l sodium pyrophosphate in a pH = 12 solution provided by the addition of 1 g/l of sodium hydroxide, i.e. caustic soda) and a subsequent separation of the fine waste (mostly clay) minerals by wet sieving. The use of hydrocyclones for classifying is not suitable for this material because of its loose structure, as the centrifugal pumps to be applied before the hydrocyclones would disperse practically all of the ore into waste. In order to avoid the use of pumps the whole beneficiation plant should be situated so that the ore and the slurry should pass downwards from equipment to equipment by gravitation.

The dry mass of the bauxite to be processed in the beneficiation plant would be 600,000 tpy, corresponding to a wet material of 1,000,000 tpy as an average and up to 100,000 tpm during the rainy season. This material is loose and sticky severely restricting the choice of equipment to be used because of its difficult handling characteristics.

The beneficiation plant would operate in two shifts corresponding to the mining schedule. During the rainy season (when the mine would sometimes operate in only one shift), raw material sufficient for the second shift should be stored in a covered area provided for this purpose.

A beneficiation capacity of 150 tph (dry basis) should be provided for processing 600,000 tpy raw ore (dry basis) in 250 working days per year and two shifts (16 hours) per day. This would amount to 1200 t (dry) ore per shift, requiring a 600 m^2 area (e.g. 20 x 30 m) if the material would be stacked 3 m high. (The loose weight of the ore is expected to be about 0.7 t/m^3 dry basis.)

The raw ore dumped by the dumper trucks would be pushed by a bulldoz er (1) into the receiving bin (3) (see Figs. II/1 and II/2). The latter would have a volume of about 60 m^3. Occasionally the ore could be dumped im mediately into the bin. However it would be important to ensure the continuity of the operation, since a raw material stored for a longer time and drying completely could not be succesfully beneficiated anymore.

The receiving bin should be covered by a 200 x 200 mm screen (2) to retain any large boulders or plant pieces, etc. Two cylindrical feeders (4) working in opposite directions would feed the ore from the bin (3) onto so called active screens (5). These feeders are the only ones capable to move the sticky, muddy material. The slightly different rpm of their cylinders serve for self-cleaning. Operating only one of the feeders (4) the beneficiation plant would work with half of its capacity.

The active screens (5) consist of longitudinal screen bars, every second of each fixed into a different frame. The two frames move in an oppo site phase with a very high initial acceleration. This way they efficiently break up any mud boulders and only non-ore materials (e.g. hard pieces of waste rock, roots and other plant remains, metal pieces, etc.) remain on them. These would be transported by conveyor (6) to a waste heap.

THE FLOW SHEET OF THE BENEFICIACION PLANT

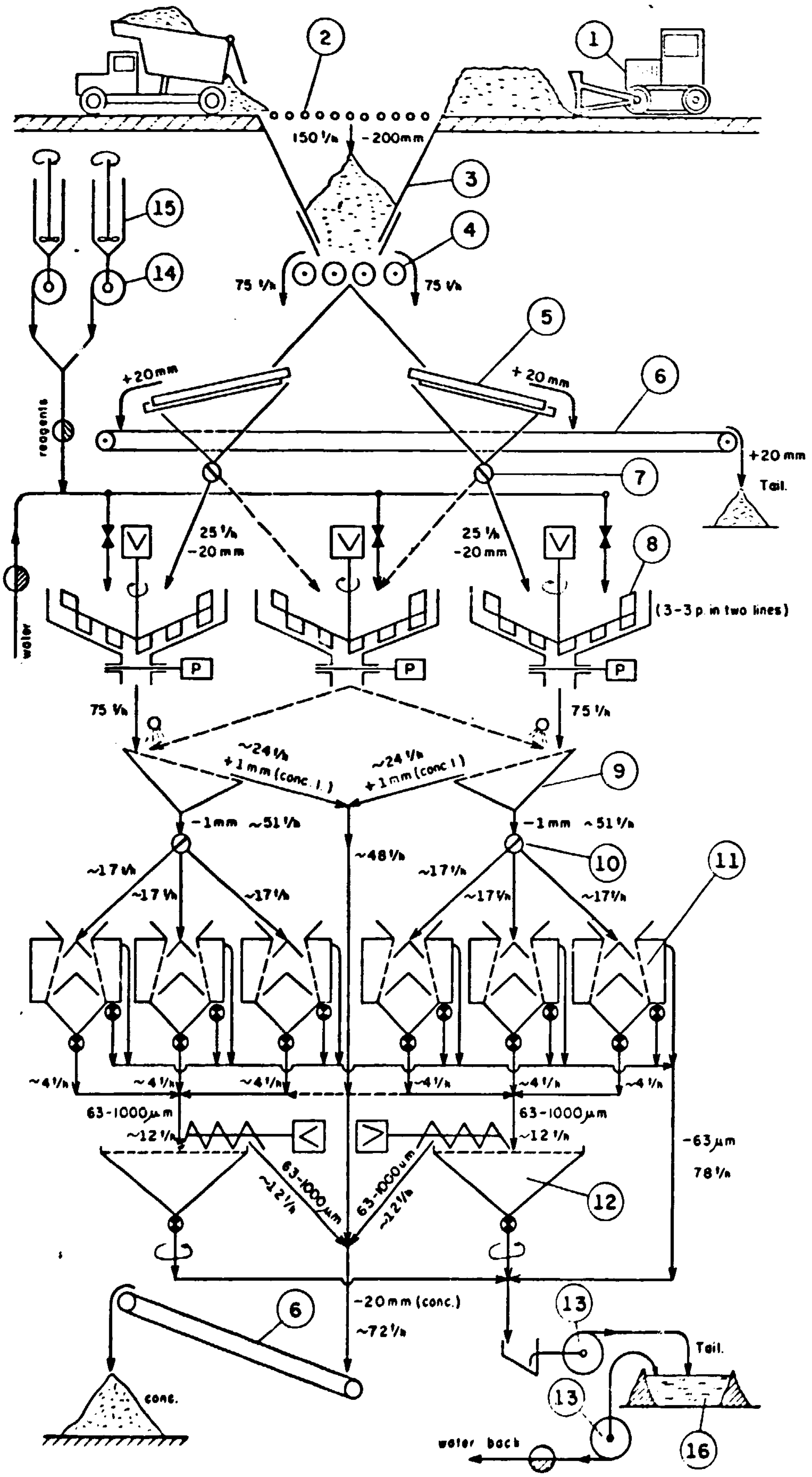
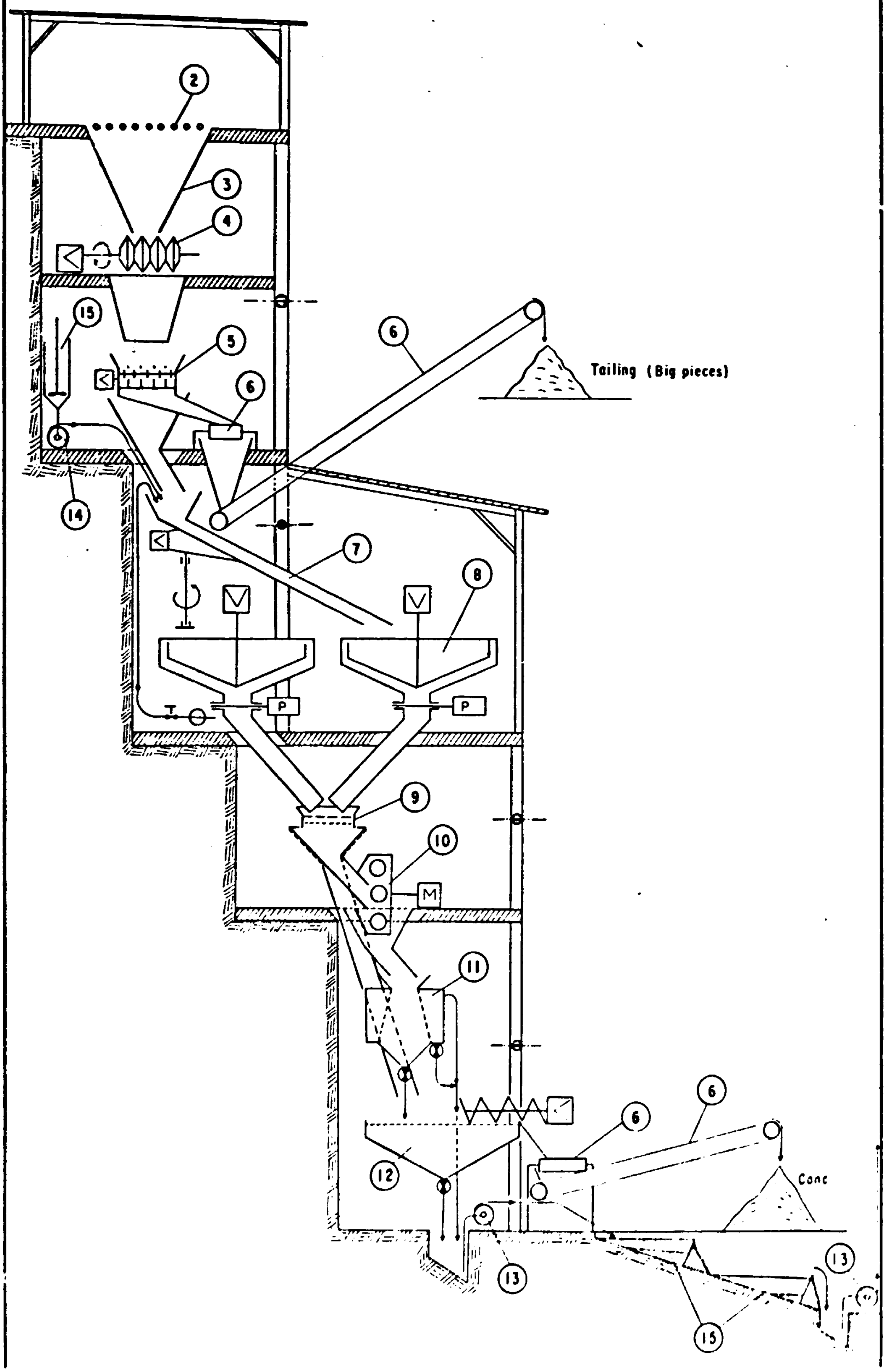


Fig. II/1

COMPILED BY Zs CSILLAG 1988

SKETCH OF THE LOCALIZATION OF THE MACHINERY OF THE BENEFICIATION PLANT



The material passing through the active screens would get through one of the vibrating feeding troughs (7) into the six dia 8 m special, slightly conical soaking tanks (8) provided with very slow-moving variable rpm agitators similar to the ones known as Anker-agitators in the chemical industry and with a dia 400 mm opening to enable the slurry to be emptied within 10 minutes. The latter would be closed and opened by a pneumatically operated gate valve. One charge of a soaking tank would consist of 12.5 t of ore (dry basis) with an adhesive moisture content varying between about 4 and 12.5 t according to the season and of as much of the soaking liquid with the above composition (25 to 33.5 m^3) that the total volume of the slurry be about 41.7 m^3 and its average solids concentration about 300 g/l. The liquid would fill the tank up to a height of about 0.8 m whereas the height of the dense ore slurry would be about 0.3 m. With a filling time of 10 min, an agitation time of 10 min and a discharge time of 10 min as well, the total operating cycle would be about 30 min and the average retention time of the solids about 20 min, which would correspond to the optimum one found during the laboratory tests. The optimum agitation time should be experimentally determined for every raw material type after the start-up of the plant.

The amount of water required for the 300 g/l solids concentration of the slurry found to be the optimum one during the laboratory tests would be supplied by one of the centrifugal pumps (16) through a distributor pipeline. The required amounts of the reagents would be fed by dosage pumps (14) from the preparation tanks (15) into this pipeline, so a ready made soaking liquid would be fed into the soaking tanks (8).

After soaking the slurry would get from the tanks (8) onto the heavy vibrators (9) covered by 15 # screens. These would separate and wash the plus 1 mm fraction of the soaked material expected to amount to a third of the total solids (i.e. about 50 tph dry basis) in order to reduce the load of the cloak sieves (11). The plus 1 mm fraction would get immediately onto the product filters (12) The slurry containing the minus 1 mm fraction would be fed through three branches of the distributors (10) into the so-called cloak sieves (11) to be provided with about 10 m^2 of 250 # screens per unit on the basis of the laboratory tests

Their high sieving capacity is due to the fact that the material to be sieved would not load and so would not close the surface of the screens but would simply drop downward beside them. The fine particles would be carried through the screens by the water passing through the same. Particles being occasionally squeezed into the openings of the screens would be removed by a high-acceleration staggering movement of the same. In order to facilitate this removal the screens would be slightly reclining. The rapidly settling product (expected to amount to a sixth of the original solids, i.e. to about 25 tph) would be removed through cell-feeders at the bottom of the equipment. The water containing the finest solids particles (about 50% of the original solids) would leave at the top through overflow pipes.

The plus 63 µm to minus 1 mm part of the concentrate would be filtered on the two rotary pan filters (12) together with the plus 1 mm fraction, wherefrom the beneficiated product would be discharged by transporting screws onto the conveyor belt carrying it into the stockpile of the alumina plant.

The fine mud would be pumped by one of the large centrifugal pumps (13) to the tailing pond (17). As an effect of the dispersing agents added to it this mud would poorly settle, so the tailing ponds should be made as large as possible and they should consist of at least two stages. Since the daily water consumption would amount to about 7000 m^3 (16 x 440 m^3/h including the washwater fed to the screens and the sieves), a 10 day settling time would require a "free" volume of 70,000 m^3 over and above the volume of the settled mud. The latter (calculating with 0.75 tons of dry material per m^3 volume) would fill about 400,000 m^3 every year. The overflow of the second pond would flow into a sump and could be pumped back to the plant by one of the centrifugal pumps (16). During the start-up of the plant only fresh water would be available, this should be taken from the nearby river and fed into the above-mentioned sump. Later on only the water losses would have to be covered this way (they are expected to amount to a figure between 1000 and 2000 m^3 per day depending on the season). The clarification of the mud slurry could be improved by acidification (using organic acids like formic, acetic, citric acid, etc.) and large molecular anionactive floccculants, however, their use would be too expensive.

The settled mud could be advantegeously used as a raw material for a ceramic industry, e.g. of brick or roof tile manufacturing. However, any detailed investigation of this question would exceed the framework of the present study. The necessary technology for such industries is available.

III. TECHNOLOGICAL TESTING OF THE BAUXITE AND CONCENTRATE SAMPLES

Table III/1 contains the chemical analyses of the bauxite samples received in 1987 and 1988, respectively. It also contains the analyses of a 1:1:1 blend made of samples CPT-1/87, CPT-2/87 and CPT-3/88 (designated CPT-123) and of two beneficiated samples, CPT-2D and CPT-3D. (The former was made of sample CPT-2/88, the latter from CPT-3/88, as described in the previous chapter.)

Though the samples sent in 1988 are of a somewhat poorer quality than those of 1987, their composition is rather similar. Therefore, the 1988 samples have not been analysed for their minor constituents. The composition of the blend corresponds to the average of its constituents within the usual error margins. Remarkable are the unusually high amounts of bivalent iron found in all the three analysed samples, further the relatively high amounts of CaO and MgO found in them. The organic carbon content of all samples are high, that of CPT-2 especially. During beneficiation the organics content of the samples (especially that of CPT-3) was further enriched. On the other hand, the P_2O_5, V_2O_5 and F contents of the blend are acceptably low, so they can not be so high in any of the individual samples, that they could disturb its processing. Therefore, the individual samples have not been analysed for these components.

The mineralogical composition of the same samples is shown in Table III/2.

Table III/1

Chemical composition of original and beneficiated Ostuacan samples

Component	CPT-1/87	CPT-2/87	CPT-3/87	CPT-1/88	CPT-2/88	CPT-3/88	CPT-123	CPT-2D	CPT-3D
Al_2O_3 %	42.48	41.64	42.00	40.50	41.01	41.04	41.60	42.00	45.20
SiO_2 %	5.06	10.95	12.13	7.14	12.86	12.38	9.50	12.63	9.20
Fe_2O_3 (total) %	24.57	20.03	18.48	24.40	18.23	18.89	20.70	18.70	17.30
TiO_2 %	2.87	2.33	2.13	2.55	2.06	2.03	2.20	2.50	2.00
L.O.I. %	22.22	23.22	22.42	22.85	22.78	22.22	23.40	22.00	24.40
S (total) %	0.09	0.09	0.07	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
FeO %	2.27	1.94	2.05	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Mn_3O_4 %	-	0.08	-	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
CaO %	0.52	0.68	2.01	n.a.	n.a.	n.a.	1.05	1.28	0.96
MgO %	0.66	0.51	0.49	n.a.	n.a.	n.a.	0.55	0.78	0.77
C_{org}	0.22	0.37	n.a.	n.a.	n.a.	0.14	0.24	0.44	0.23
P_2O_5	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	0.20	n.a.	n.a.
V_2O_5	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	0.17	n.a.	n.a.
F	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	0.085	n.a.	n.a.

Table III/2

Mineralogical composition of original and beneficiated Ostuacan samples

Component	CPT-1/87	CPT-2/87	CPT-3/87	CPT-1/88	CPT-2/88	CPT-3/88	CPT-123	CPT-2D	CPT-3D
Al_2O_3 % in									
gibbsite	36.5	31.6	31.4	33.1	29.7	29.6	32.3	33.0	37.9
kaolinite	4.0	6.6	3.9	5.4	8.1	8.9	6.5	5.8	6.2
diaspore	0.4	1.4	0.8	0.4	1.2	0.6	1.0	1.2	0.6
goethite	1.6	2.0	1.9	1.6	2.0	1.9	1.8	2.0	0.5
	42.5	41.6	42.0	40.5	41.0	41.0	41.6	42.0	45.2
SiO_2 % in									
quartz	0.4	3.3	1.7	0.7	3.4	1.9	1.9	5.8	1.9
kaolinite	4.7	7.7	10.5	6.4	9.5	10.5	7.6	6.8	7.3
	5.1	11.0	12.2	7.1	12.9	12.4	9.5	12.6	9.2
Fe_2O_3 % in									
goethite	9.2	6.7	6.7	9.1	6.6	6.4	7.4	6.7	5.9
hematite and maghemite	7.3	7.0	4.7	7.4	5.2	5.4	6.1	6.5	6.4
magnetite	8.1	6.6	7.0	7.9	6.4	7.1	7.2	5.5	5.0
	24.6	20.3	18.4	24.4	18.2	18.9	20.7	18.7	17.3
TiO_2 % in									
anatase	2.9	2.3	2.2	2.6	2.1	2.0	2.2	2.5	2.0
rutile	tr	tr	tr	tr	tr	tr	tr	tr	tr
	2.9	2.3	2.2	2.6	2.1	2.0	2.2	2.5	2.0

The mineralogical composition of the samples supports the conclusions drawn from the chemical ones: the samples received in 1983 are of a little poorer quality (less gibbsite, more kaolinite), however, the characteristic features of the corresponding samples are the same (e.g. the relatively high quartz and diaspore contents of the CPT-2 samples compared to the other ones, etc.).

Digestion test series have been carried out with six of the above samples: CPT-1/87, CPT-2/87, CPT-3/83, CPT-123, CPT-2D and CPT-3D. The parameters and results of these tests are shown in Tables III/3 through III/8. The mineralogical composition of one mud sample from each digestion test series is shown in Table III/9. Each test series consists of six individual digestion tests and their aim is to find the lowest molar ratio at which the extraction yield on alumina is still close to the highest one. The results show that a digestion molar ratio of 1.4 (mol caustic Na_2O per mol Al_2O_3 in solution) can be selected with a still sufficient safety margin.

The extraction yields and Na_2O to SiO_2 weight ratios expected at this molar ratio with the different samples are shown in Table III/10.

A seventh digestion test series has been carried out to find the minimum retention time required for a proper desilication of the liquor phase of the digestion effluent. The results of this test series are compiled in Table III/11. They show that the SiO_2 concentration of the liquor phase does not change after 45 minutes. Since the test series has been carried out with the sample containing the lowest amount of reactive SiO_2, and the desilication reaction is the faster, the more reactive SiO_2 is present, a retention time of 45 minutes will be sufficient for a proper desilication.

Table III/3

Digestion tests

Sample: CPT-1/87

Digestion temperature: 143 °C

Retention time: 2 h

Liquor analyses

Sample	Caustic Na_2O g/l	Al_2O_3 g/l	Molar ratio	Sample (g) per 150 cm^3 digesting liquor
Digesting liquor	140.9	70.0	3.31	-
CPT-1/1	120.7	157.1	1.26	42.6
CPT-1/2	120.7	152.3	1.30	40.0
CPT-1/3	120.2	146.2	1.35	37.7
CPT-1/4	122.5	137.2	1.50	33.4
CPT-1/5	124.3	132.0	1.54	29.7
CPT-1/6	126.7	125.8	1.65	26.4

Mud analyses

Sample	L.O.I. %	Al_2O_3 %	SiO_2 %	Fe_2O_3 %	TiO_2 %	CaO %	MgO %	Na_2O %	η_A %	$\frac{Na_2O}{SiO_2}$
CPT-1/1	7.7	17.5	10.7	49.6	5.4	0.9	1.2	6.0	79.6	0.56
CPT-1/2	7.7	16.4	10.7	50.5	5.6	1.0	1.2	6.1	31.2	0.57
CPT-1/3	8.3	16.3	11.0	50.0	5.3	1.0	1.1	6.1	81.1	0.55
CPT-1/4	6.8	15.0	10.3	52.0	5.6	1.1	1.4	5.6	83.3	0.54
CPT-1/5	7.6	15.5	10.4	52.1	5.3	1.2	1.3	6.1	82.8	0.59
CPT-1/6	7.8	15.5	10.9	50.8	5.2	1.4	1.3	6.3	82.3	0.58

Table III/4

Digestion tests

Sample: CPT-2/87

Digestion temperature: 143 ^{o}C

Retention time: 2 h

Liquor analyses

Sample	Caustic Na_2O g/l	Al_2O_3 g/l	Molar ratio	Sample (g) per 150 cm^3 digesting liquor
Digesting liquor	140.9	70.0	3.31	-
CPT-2/1	112.1	149.2	1.23	44.7
CPT-2/2	112.2	145.4	1.27	42.1
CPT-2/3	114.8	140.0	1.34	39.7
CPT-2/4	116.2	133.7	1.42	35.3
CPT-2/5	118.0	126.8	1.53	31.5
CPT-2/6	118.7	119.4	1.63	28.1

Mud analyses

Sample	L.O.I. %	Al_2O_3 %	SiO_2 %	Fe_2O_3 %	TiO_2 %	CaO %	MgO %	Na_2O %	η_A %	$\frac{Na_2O}{SiO_2}$
CPT-2/1	9.3	20.6	20.8	34.0	3.7	1.1	0.8	7.9	70.4	0.38
CPT-2/2	8.0	20.6	20.8	34.8	3.5	1.1	0.9	8.2	71.1	0.39
CPT-2/3	9.3	19.9	20.6	35.3	3.7	1.2	0.9	8.0	72.5	0.39
CPT-2/4	9.2	19.5	20.6	35.5	3.9	1.2	0.9	8.1	73.2	0.39
CPT-2/5	9.5	19.9	21.0	35.5	3.9	1.2	0.9	8.4	73.0	0.40
CPT-2/6	8.5	19.4	20.8	34.8	4.1	1.3	1.0	8.2	72.8	0.39

Table III/5

Digestion tests

Sample: CPT-3/88

Digestion temperature: 143 ^{o}C

Retention time: 2 h

Liquor analyses

Sample	Caustic Na_2O g/l	Al_2O_3 g/l	Molar ratio	SiO_2 g/l	Sample (g) per 150 ml digesting liquor
Digesting liquor	140.9	70.0	3.31	n.a.	-
CPT-3/1	109.0	142.3	1.26	0.51	50.6
CPT-3/2	115.8	136.1	1.39	0.54	41.4
CPT-3/3	117.6	129.1	1.49	0.47	37.2
CPT-3/4	117.9	121.9	1.59	0.47	33.4
CPT-3/5	123.0	120.5	1.68	0.43	30.0
CPT-3/6	127.7	110.1	1.90	0.50	24.0

Mud analyses

Sample	L.O.I. %	Al_2O_3 %	SiO_2 %	Fe_2O_3 %	TiO_2 %	CaO %	MgO %	Na_2O %	η_A %	$\frac{Na_2O}{SiO_2}$
CPT-3/1	8.3	23.3	20.0	29.9	3.1	3.0	0.8	10.2	64.1	0.51
CPT-3/2	8.2	21.3	20.3	30.1	3.1	3.3	0.8	11.4	67.4	0.56
CPT-3/3	8.1	20.9	21.1	30.5	3.1	3.3	0.9	10.6	68.4	0.50
CPT-3/4	8.5	20.9	21.5	29.6	3.0	3.4	0.9	11.0	67.5	0.51
CPT-3/5	8.6	20.9	20.9	29.6	3.0	3.4	0.9	11.2	67.5	0.54
CPT-3/6	8.5	20.7	21.0	29.6	3.1	3.6	0.9	11.3	67.8	0.54

Table III/6

Digestion tests

Sample: CPT-123

Digestion temperature: 143 °C

Retention time: 2 h

Liquor analyses

Sample	Caustic Na_2O g/l	Al_2O_3 g/l	Molar ratio	SiO_2 g/l	Sample (g) per 150 ml digesting liquor
Digesting liquor	140.9	70.0	3.31	-	-
CPT-123/1	117.1	151.6	1.27	0.59	46.0
CPT-123/2	119.8	145.5	1.35	0.56	41.1
CPT-123/3	122.2	138.3	1.45	0.56	36.7
CPT-123/4	124.2	131.2	1.55	0.57	32.9
CPT-123/5	125.3	127.4	1.62	0.56	29.4
CPT-123/6	130.2	116.1	1.84	0.41	23.5

Mud analyses

Sample	L.O.I. %	Al_2O_3 %	SiO_2 %	Fe_2O_3 %	TiO_2 %	CaO %	MgO %	Na_2O %	η_A %	$\frac{Na_2O}{SiO_2}$
CPT-123/1	8.4	20.2	17.8	37.5	3.9	1.7	0.9	8.6	72.6	0.48
CPT-123/2	8.3	19.4	17.7	37.2	3.9	1.8	1.0	9.2	73.4	0.52
CPT-123/3	8.3	19.0	18.0	37.4	3.9	1.9	1.0	9.0	74.1	0.50
CPT-123/4	7.9	19.0	18.0	37.3	3.9	2.0	1.0	9.5	74.0	0.53
CPT-123/5	8.3	18.9	17.9	37.8	3.9	2.1	1.0	9.1	74.5	0.51
CPT-123/6	8.5	19.0	18.7	37.5	3.6	2.5	1.0	9.7	74.2	0.52

Table III/7

Digestion tests

Sample: CPT-2D

Digestion temperature: 143 °C

Retention time: 2 h

Liquor analyses

Sample	Caustic Na_2O g/l	Al_2O_3 g/l	Molar ratio	SiO_2 g/l	Sample (g) per 150 ml digesting liquor
Digesting liquor	141.4	73.7	3.16	-	-
CPT-2D/1	119.8	149.4	1.31	0.64	41.4
CPT-2D/2	121.5	142.5	1.40	0.57	36.6
CPT-2D/3	119.0	137.3	1.42	0.58	32.4
CPT-2D/4	123.0	134.7	1.50	0.54	30.5
CPT-2D/5	124.3	127.2	1.60	0.48	28.7
CPT-2D/6	125.8	120.8	1.71	0.43	25.0

Mud analyses

Sample	L.O.I. %	Al_2O_3 %	SiO_2 %	Fe_2O_3 %	TiO_2 %	CaO %	MgO %	Na_2O %	η_A %	$\frac{Na_2O}{SiO_2}$
CPT-2D/1	6.4	17.8	24.4	33.8	3.9	2.2	1.4	7.8	76.6	0.32
CPT-2D/2	7.4	17.9	24.3	33.0	4.0	2.2	1.4	8.2	75.9	0.34
CPT-2D/3	6.9	17.5	24.4	33.5	4.0	2.3	1.4	8.0	76.7	0.33
CPT-2D/4	8.9	18.9	24.3	29.6	3.9	2.1	1.1	9.0	71.7	0.37
CPT-2D/5	6.9	17.9	24.0	32.5	4.0	2.4	1.4	8.3	75.5	0.35
CPT-2D/6	7.8	18.1	24.0	31.6	3.9	2.5	1.4	8.6	74.5	0.36

Table III/8

Digestion tests

Sample: CPT-3D

Digestion temperature: 143 ^{o}C

Retention time: 2 h

Liquor analyses

Sample	Caustic Na_2O g/l	Al_2O_3 g/l	Molar ratio	Sample (g) per 150 cm^3 digesting liquor
Digestion liquor	140.9	70.0	3.31	-
CPT-3D/1	123.1	159.5	1.27	42.3
CPT-3D/2	126.0	152.5	1.36	37.7
CPT-3D/3	127.9	142.8	1.47	33.7
CPT-3D/4	129.1	138.8	1.53	31.8
CPT-3D/5	129.1	134.8	1.65	30.1

Mud analyses

Sample	L.O.I. %	Al_2O_3 %	SiO_2 %	Fe_2O_3 %	TiO_2 %	CaO %	MgO %	Na_2O %	η_A %	$\frac{Na_2O}{SiO_2}$
CPT-3D/1	7.2	19.9	20.2	35.9	3.7	1.9	1.6	8.7	78.8	0.43
CPT-3D/2	7.3	19.2	20.3	36.2	3.7	2.0	1.6	9.0	79.7	0.44
CPT-3D/3	7.5	19.2	20.3	35.9	3.7	2.0	1.6	9.2	79.5	0.45
CPT-3D/4	7.3	18.9	20.5	35.7	3.6	2.2	1.7	9.0	79.7	0.44
CPT-3D/5	7.5	18.5	20.2	36.5	3.7	2.3	1.6	8.9	80.6	0.44

Table III/9

Mineralogical composition of selected mud samples

	CPT-1/4	CPT-2/4	CPT-3/3	CPT-123/3	CPT-2D/3	CPT-3D/3
Al_2O_3 % in:						
sodalite	9.3	12.6	15.3	13.2	11.7	13.9
goethite	3.7	3.5	3.0	3.2	3.5	4.0
diaspore	1.0	2.5	0.9	1.3	1.8	0.7
hematite and maghemite	1.0	0.9	0.6	0.9	0.5	0.6
calcium aluminium silicates	tr	tr	1.1	0.4	tr	-
	15.0	19.5	20.9	19.0	17.5	19.2
SiO_2 % in:						
sodalite	9.5	14.8	18.1	14.4	14.0	16.3
quartz	0.8	5.8	3.0	3.6	10.4	4.0
	10.3	20.6	21.1	18.0	24.4	20.3
Fe_2O_3 % in:						
goethite	19.4	11.7	10.2	13.3	11.8	12.0
magnetite	17.1	11.6	11.4	13.0	11.5	14.0
hematite and maghemite	15.5	12.2	8.9	11.4	10.2	9.9
	52.0	35.5	30.5	37.4	33.5	35.9
TiO_2 % in:						
$CaTiO_3$	1.0	1.0	1.1	0.9	1.4	1.1
sodium titanates	4.6	2.9	2.0	3.0	2.6	2.6
	5.6	3.9	3.1	3.9	4.0	3.7
CaO % in:						
calcite	0.4	0.7	0.7	0.6	1.3	1.2
$CaTiO_3$	0.7	0.5	0.8	0.6	1.0	0.8
calcium aluminium silicates	tr	tr	1.8	0.7	tr	tr
	1.1	1.2	3.3	1.9	2.3	2.0

Table III/10

Extraction yields on Al_2O_3 and Na_2O to SiO_2 weight ratios in the digestion residue expected at the processing of the various Ostuacan samples

Sample	Extraction yield on Al_2O_3, %	Na_2O to SiO_2 weight ratio in digestion residue
CPT-1/87	83	0.565
CPT-2/87	73	0.39
CPT-3/88	68	0.53
CPT-123	74	0.51
CPT-2D	76.5	0.33
CPT-3D	79.5	0.45

Table III/11

Digestion/desilication tests

Sample: CPT-1/87

Digestion temperature: 143 °C

Retention time: 15; 30; 45; 60; 90; 120 min

Liquor analyses

Sample	Caustic Na_2O g/l	Al_2O_3 g/l	Molar ratio	SiO_2 g/l	Retention time min
Digesting liquor	144.0	74.1	3.19	-	-
CPT-1I/1	122.3	132.9	1.51	0.67	15
CPT-1I/2	124.4	135.0	1.52	0.54	30
CPT-1I/3	125.0	135.6	1.52	0.50	45
CPT-1I/4	123.9	133.2	1.53	0.50	60
CPT-1I/5	122.5	139.6	1.44	0.50	90
CPT-1I/6	124.9	135.5	1.52	0.51	120

Settling tests have been carried out with the digestion residues of two of the above samples, CPT-1/87 and CPT-2/87. The results are shown in Table III/12. They show that the settling characteristics of the muds are rather similar to those of Hungarian ones.

Table III/12

Red mud settling tests

Time (min)	CPT-1/87 without flocculant cm	CPT-1/87 with a flocculant of 2 g flour per kg red mud cm	CPT-2/87 without flocculant cm	CPT-2/87 with a flocculant of 2 g flour per kg red mud cm
0	29.7	29.4	29.6	29.8
5	29.3	25.0	29.3	24.4
10	28.8	18.6	29.0	17.4
15	28.4	14.2	28.7	15.6
20	27.9	13.7	28.3	14.8
25	27.5	13.0	28.0	14.1
30	27.0	12.5	27.7	13.7
35	26.6	12.1	27.4	13.3
40	26.2	11.8	27.0	13.0
45	25.6	11.4	26.6	12.7
50	25.3	11.2	26.4	12.5
55	24.8	11.0	26.0	12.3
60	24.4	10.9	25.7	12.1
120	19.5	9.9	22.0	11.0
Initial solids concentration, g/l	90.6	93.8	105.5	105.5

IV. MAIN PARAMETERS OF AN ALUMINA PLANT PROCESSING BAUXITES CHARECTERISED BY THE INVESTIGATED SAMPLES

Since the ore is very soft, it would require no grinding, only slurrying in the digestion liquor. However to produce a uniform slurry with a maximum particle size of 0.5 mm low-capacity ball mills should be installed.

The digestion would be carried out in a conventional low-pressure digestion line at a temperature of 140 to 145°C with a retention time of about 45 minutes. The liquor and the slurry would be preheated partly by the flash steam of the digested slurry, partly by boiler steam with a pressure of about 16 bars.

The flashed slurry would be diluted by the overflow of the first stage of the countercurrent mud washing system and decanted in settlers. The underflow of the latter would be washed in three countercurrent decanting stages and the underflow of the last of these would be filtered on rotary drum filters and washed with hot water on their surface. Subsequently it would be disposed to a so-called dry stacking area, where its harmful effect to the environment would be minimal.

The overflow of the settlers containing about 110 g/l caustic Na_2O and 120.6 g/l Al_2O_3 would be subjected to control filtration and cooling and fed to a continuous precipitation system consisting of about 15 precipitator tanks. Here it would be seeded in two stages and further cooled after the first stage. The precipitated slurry would be fed into a three-stage classification system (primary, secondary and tray thickeners), and the underflow of each stage would be subjected to filtration and washing. The coarsest hydrate originating from the first stage would be calcined to product alumina.

The hydrate of the other two stages would be use for seeding (that of the third stage at the beginning of the precipitation line, that of the second stage in the third or fourth precipitator). The seed has to be washed in order to remove its sodium oxalate content which would probably be unusually high because of the high organics content of the raw material.

The overflow of the so-called tray thickeners would be fed into evaporators to remove as much water from it as required by the water balance of the alumina plant. The resulting strong liquor would be used for digesting the ore after making up the caustic soda losses of the process by a 50 % solution of NaOH.

The process flows and the size of the equipment would be more or less independent of the ore quality from the control filtration through the precipitation and hydrate classification to the calcination. The rest of the process flows and equipment would strongly depend on the quality of the ore. Two main variants have been elaborated for the capacity of the plant, one for 100,000 mtpy and the other for 200,000 mtpy alumina.

The main process flows have been calculated in the following way:

Multiplying the Al_2O_3 percentages of the six tested samples (CPT-1/87; CPT-2/87; CPT-3/88; CPT-123; CPT-2D and CPT-3D) shown in Table III/2 with the expected extraction yields indicated in Table III/10 the amount of Al_2O_3 to be dissolved from 1 t of dry ore have been obtained. These are 332.6; 304.0; 279.1; 307.8; 321.3 and 359.3 kg/t in the above order of the samples. Reducing these figures for various processing losses (a part of which are more or less constant, the other part the higher, the lower the quality of the ore) by 5; 6; 7; 6; 6 and 5 relative percents the production of 335.0; 285.8; 259.6; 289.3; 302.0 and 341.3 kg alumina can be expected from every metric ton of dry ore. Dividing 1000 kg/t with these figures 2.99; 3.50; 3.85; 3.46; 3.31 and 2.93 tons of dry ore would be required for the production of 1 t alumina.

By multiplying the expected dry ore consumptions with the so-called mud factors (these can be calculated by dividing the Fe_2O_3 content of the ores by the Fe_2O_3 content of the digestion residues and their value is 0.473; 0.564; 0.623; 0.553; 0.558 and 0.482), the expected amount of the digestion residue (or so-called blow-off mud) can be calculated: 1.42; 1.98; 2.40; 1.91; 1.85 and 1.41 t of dry mud per t of alumina. Adding to these figures 50 % of the expected Al_2O_3 losses (for so-called autoprecipitation in the settlers and mud washers) multiplied by 1.53 (for the $2Al(OH)_3/Al_2O_3$ stoichiometric ratio) the amount of dry red mud to be disposed of would be 1.46; 2.03; 2.46; 1.96; 1.90 and 1.45 t/t.

By multiplying the SiO_2 content of the ore samples by the Na_2O to SiO_2 ratios given in Table III/10 and by 1.32 (conversion factor from Na_2O to 97.5 % purity NaOH) the chemical NaOH losses can be calculated for 1 t of dry ore: 37.7; 56.4; 86.6; 64.0; 55.0 and 54.7 kg/t. Multiplying these figures by the specific dry ore consumptions, adding 1 % of the dry mud weight for the NaOH content of the filtered mud and another 10 kg NaOH for various non-mud-related losses (alumina's Na_2O content; physical losses) the expected specific NaOH consumption can be obtained for 1 t of alumina: 137.3; 227.7; 368.0; 251.0; 211.0 and 184.8 kg/t.

The next job has been to set up the water balance of the process. The amount of water entering the process is made up of five main sources: raw material (L.O.I. plus adhesive moisture); hydrate wash water; red mud wash water; water content of the make-up caustic; unauthorized dilutions to the process liquor (also called false water). The same amount of water has to be removed in order to keep the balance. Water leaves the process with the product hydrate and red mud (both as L.O.I. and adhesive moisture), at the flashing of the digested slurry and in the evaporation plant unit.

Based on the geologist's estimate (average moisture of the ore 25 % during the four driest months of the year; 50 % during the most humid months of the year; 40 % during the four intermediate months) an average adhesive moisture content of 40 % (wet basis) has been taken into consideration for the raw material in the variants with unbeneficiated feed. The adhesive moisture content of the beneficiated materials would probably be higher, about 50 % as an average.

A 50 % NaOH solution contains 1.225 kg H_2O for every kg NaOH (including the water content of the NaOH itself). 2 tons of water have been taken into consideration for the washing of every ton of red mud (dry basis), 0.9 ton for hydrate washing and a total of 1 ton of unauthorized dilutions per ton of alumina.

The L.O.I. of the red mud is the sum of the L.O.I. content of the blow-off mud and of 34.6 % of the autoprecipitation. An average of 45 % adhesive moisture has been taken into consideration for the red mud and one of 12 % for the product hydrate.

Table IV/1 shows the water balance of the variants.

Table IV/1

Water balance of the variants in kg/t alumina

Material flow	Ore quality corresponding to sample CPT-1/87	CPT-2/87	CPT-3/88	CPT-123	CPT-2D	CPT-3D
In: Ore: L.O.I.	664	813	856	810	728	715
adh.m.	1993	2333	2576	2307	3310	2930
Make-up caustic	168	279	451	307	258	226
Red mud wash water	2920	4060	4920	3920	3800	2900
Hydrate wash water	900	900	900	900	900	900
Unauthorized dilutions	1000	1000	1000	1000	1000	1000
Total	7645	9385	10694	9244	9996	8671
Out: Red mud: L.O.I.	111	199	217	175	145	120
adh.m.	1195	1661	2013	1604	1555	1186
Hydrate: L.O.I.	529	529	529	529	529	529
adh.m.	208	208	208	208	208	208
To be evaporated	5602	6788	7727	6728	7559	6628
Total	7645	9385	10694	9244	9996	8671

The penultimate line of the table shows the total amount of water to be evaporated. A part of them would be evaporated in the three flash stages of the digestion line, the rest has to be evaporated in evaporators set up specially for this purpose. Our calculations have shown that the first part (which would amount to about 6 % of the total water equivalent of the digestor feed) would be 986; 957; 940; 960; 965 and 982 kg/t, whereas the other part 4616; 5831; 6787; 5768; 6594 and 5646 kg/t.

The amount of heat required for the production of one metric ton of alumina is shown in Table IV/2.

Table IV/2

Amount of heat required for manufacturing one metric ton of alumina

	CPT-1/87	CPT-2/87	CPT-3/88	CPT-123	CPT-2D	CPT-3D
Digestion, Mcal/t	750	736	727	738	740	748
Evaporation, Mcal/t	692	875	1018	865	989	847
Minor consumers in the wet section, Mcal/t	123	152	173	148	145	123
Total wet section, Mcal/t	1565	1763	1918	1751	1874	1718
Primary heat for wet section, Mcal/t	1739	1959	2131	1946	2082	1902
Calcining, Mcal/t	800	800	800	800	800	800
Total primary heat, Mcal/t	2539	2759	2931	2746	2882	2709

The primary heat consumptions have been calculated with the assumption of a boiler plant with a thermal efficiency of 90 %.

All the characteristic consumption figures calculated on the basis of tests carried out with sample CPT-123 are in a very good agreement with the arithmetical averages of the respective figures of the individual variants (CPT-1/87; CPT-2/87 and CPT-3/88). This proves that the specific consumption figures for other combinations of the possible raw materials can also be calculated in a similar way.

Since the estimated amount of ore reserves permits only a 100,000 tpy plant to be fed by CPT-1/87 ore quality, and two logical combinations offer themselves for a 200,000 tpy plant (CPT-1/87 plus either CPT-2/87 or CPT-3D), the characteristic parameters of these three variants (called A, B and C) are compiled in Tables IV/3, IV/4 and IV/5.

Table IV/3

MATERIAL CONSUMPTION FIGURES OF VARIANTS A, B AND C

		A	B	C
Alumina production, tpy		100,000	200,000	200,000
Ore consumption (dry basis) tpy	CPT-1/87	299,000	299,000	299,000
	CPT-2/87	-	350,000	-
	CPT-3D	-	-	293,000
	TOTAL	299,000	649,000	592,000
Available alumina, Kg/t dry ore		352.6	326.4	356.0
Processing losses, % of available Al_2O_3		5	5.5	5
Alumina produced, Kg/t dry ore		335.0	308.2	338.2
Ore consumption (dry basis), t/t		2.99	3.245	2.96
Mud factor, t/t dry ore		0.473	0.522	0.477
Digestion residue, t/t alumina (dry basis)		1.42	1.70	1.41
Last washer/filtered mud, t/t alumina (dry basis)		1.46	1.75	1.45
Chemical NaOH losses, Kg/t dry ore		37.7	47.8	46.1
Total NaOH consumption, Kg/t alumina (Expressed as 97.5% purity NaOH, but actually in the form of a 50% solution)		137.3	182.9	161.0
Burnt lime, Kg/t alumina		10	10	10
Synthetic flocculant, Kg/t alumina		0.06	0.07	0.06
Industrial water (excluding beneficiation), m^3/t		7	7	7
Power (excluding beneficiation) Kwh/t		370	335	335

Table IV/4

WATER BALANCE OF VARIANTS A, B AND C IN KG/T ALUMINA

MATERIAL FLOW			A	B	C
In:	Ore:	L.O.I.	664	739	690
		Adh. m.	1993	2167	2462
		Make-up caustic	168	224	197
		Red mud wash water	2920	3500	2900
		Hydrate wash water	900	900	900
		Unautjorized dilutions	1000	1000	1000
		Total	7645	8530	8149
Out:	Red mud:	L.O.I.	111	158	115
		Adh. m.	1195	1432	1191
	Hydrate:	L.O.I.	529	529	529
		Adh. m.	208	208	208
	To be evaporated:	By flashing	986	970	984
		In evaporators	4616	5233	5122
	Total		7645	8530	8149

Table IV/5

AMOUNT OF HEAT REQUERED FOR MANUFACTURING ONE METRIC

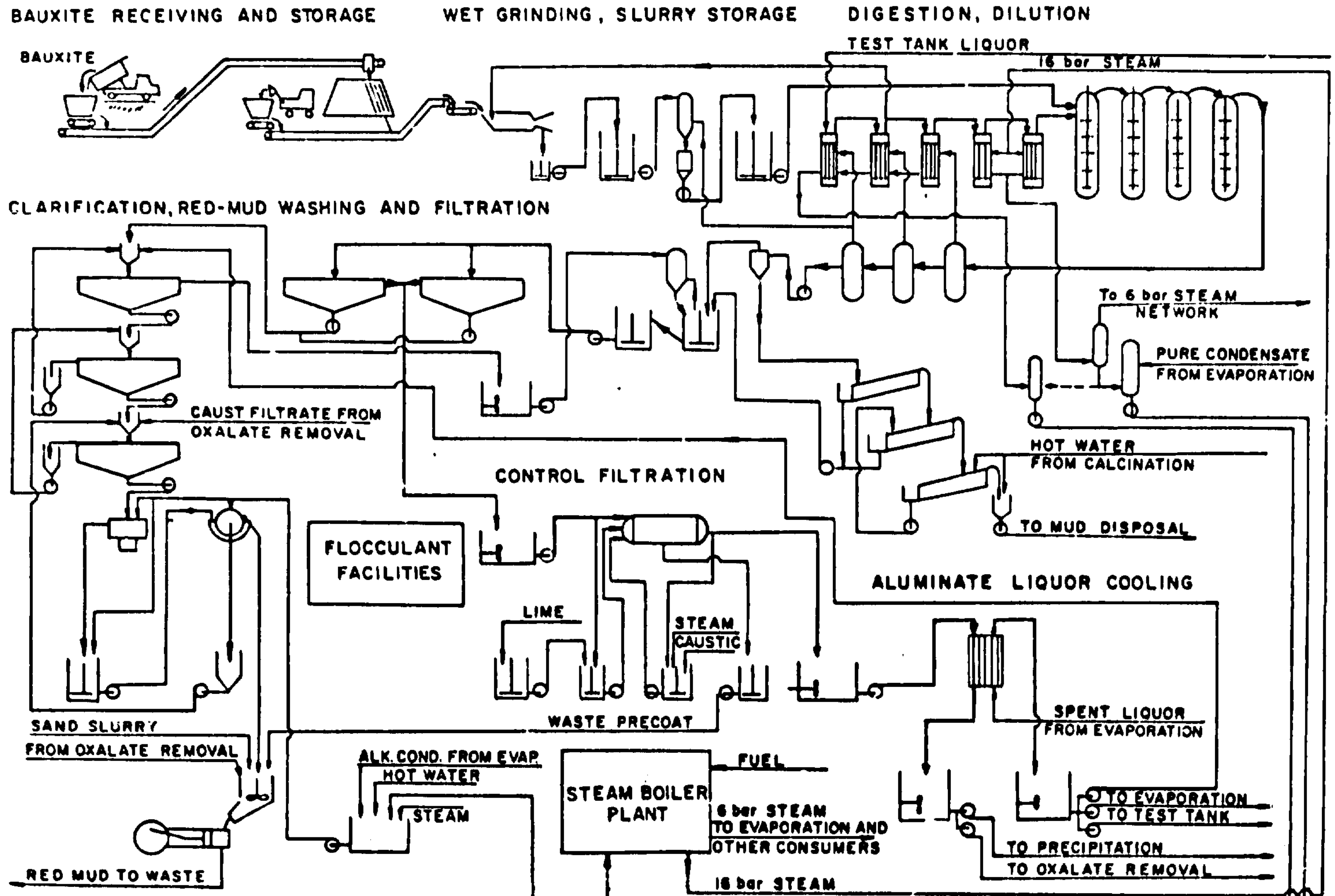
TON OF ALUMINA IN M CAL

	A	B	C
Digestion	750	745	749
Evaporation	692	785	768
Minor consumers in the wet section	123	138	123
Total wet section	1565	1666	1640
Primary heat for wet section	1739	1851	1822
Calcinig	800	800	800
Total primary heat	2539	2651	2622

Figs. IV/1 and IV/2 show the layout and the flow-sheeet of the alumina plant.

CONCEPTUAL TECHNOLOGICAL PROCESS FLOW SHEET

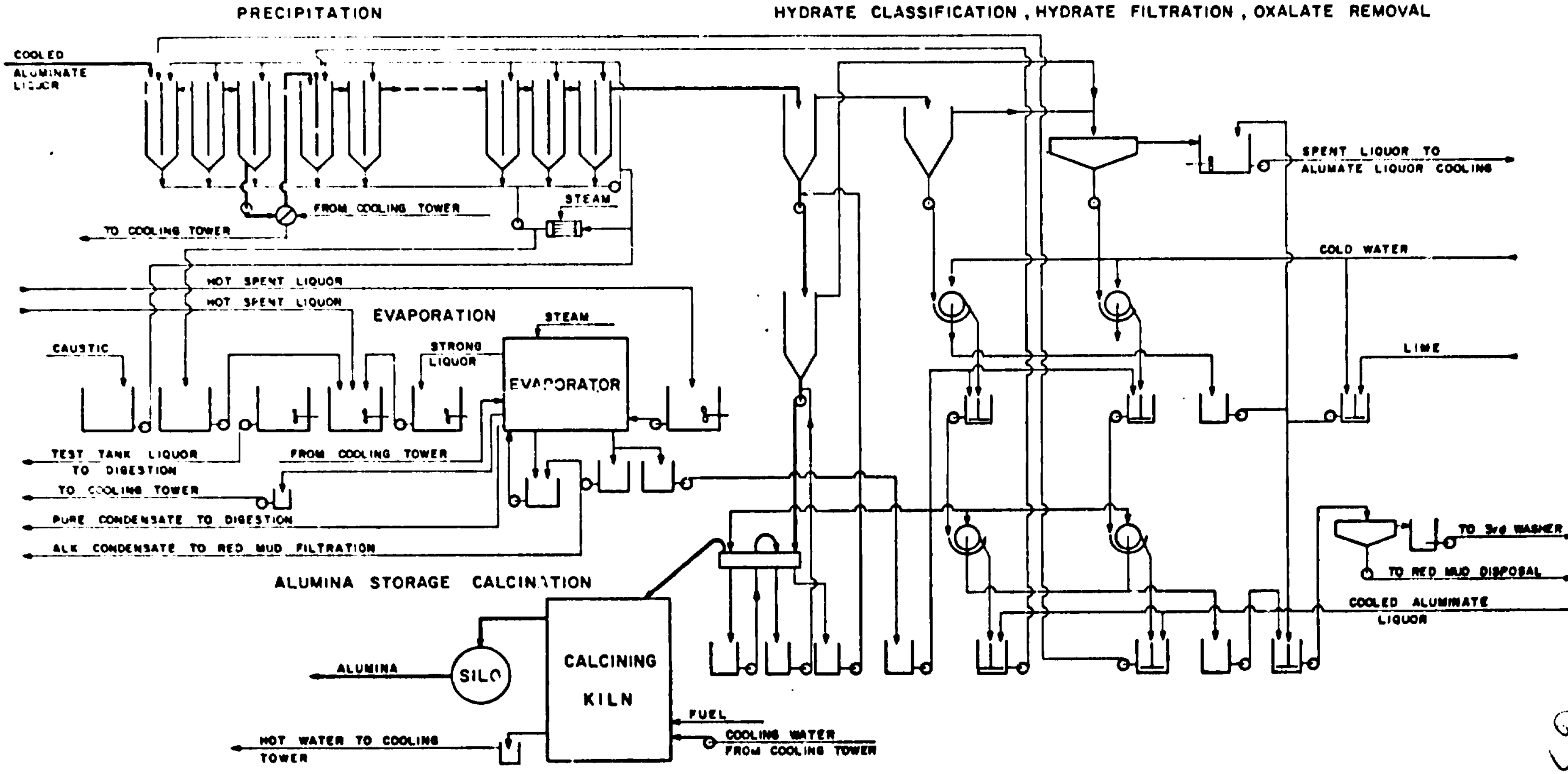
FIG. IV/1
SHEET 1



Compiled by Gy. Toth 1988

FIG. IV/1
Sheet 2

CONCEPTUAL TECHNOLOGICAL PROCESS FLOW-SHEET



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LAYOUT OF ALUMINA PLANT

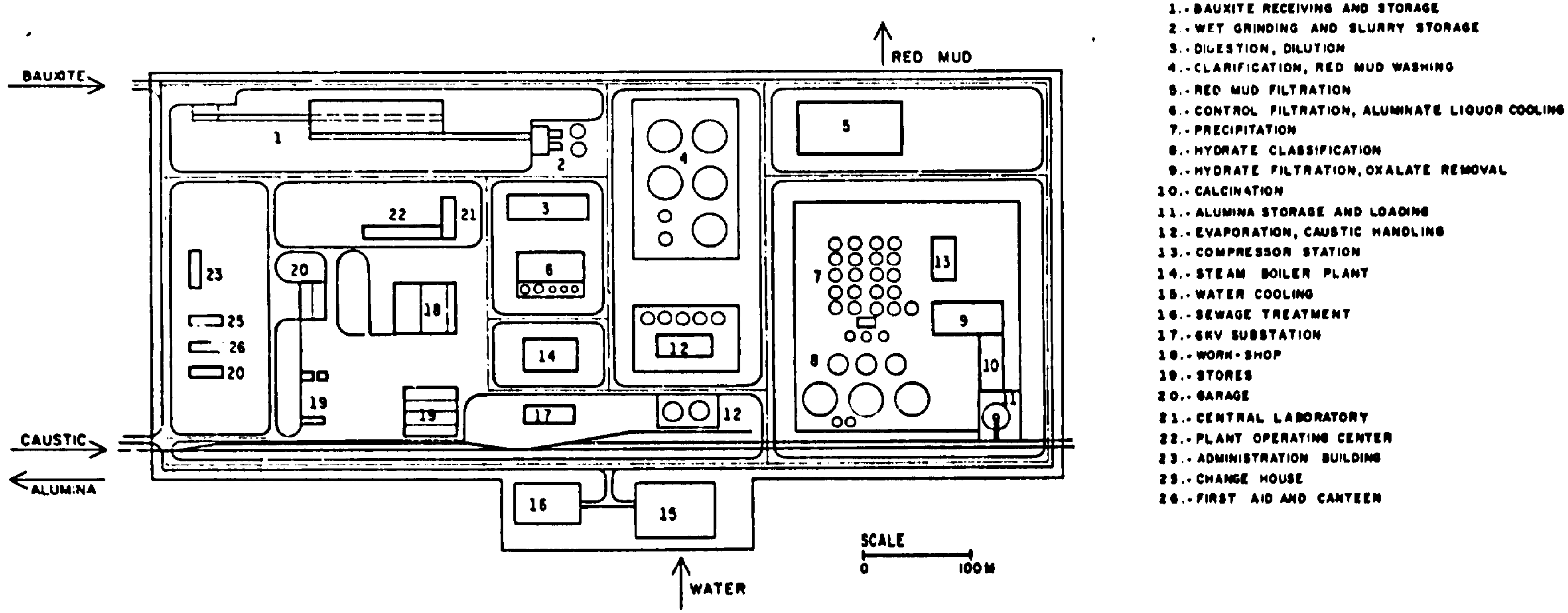

FIG. IV/2

V. BASIC EQUIPMENT REQUIRED FOR THE MANUFACTURING OF 100,000 AND 200,000 TPY ALUMINA FROM THE OSTUACAN BAUXITIC LATERITES

V/A MINING AND BAUXITE TRANSPORTATION

EQUIPMENT DAILY PROD.	VARIANT A 2000 t			VARIANT B 4333 t			VARIANT C 6000 t			TECHNICAL DATA AND/OR MACHINE TYPE
Numbers	TOT	OP	SP	TOT	OP	SP	TOT	OP	SP	
Front end loader	3	2	1	3	2	1	4	3	1	CAT 950 C=150 t/h
Loader and tractor	2	1	1	2	1	1	2	1	1	CAT 988 C=350 or 1400 m^2/h
Dumper truck	19	17	2	40	36	4	55	50	5	CAT 769 C=30 t
Roadgrader	2	1	1	2	1	1	2	1	1	CAT 120 G
Mechanical drilling machine	3	2	1	3	2	1	4	3	1	C=30 m depth, coreing with water flushing

V/B BAUXITE BENEFICIATION PLANT (ONLY FOR VARIANT C)

SERIAL NUMBER	EQUIPMENT	NUMBER TOT	OP	SP	TECHNICAL DATA
1	Bulldozer	1	1	-	CAT D8L
2	Screen	1	1	-	Slot 200 mm Size. 6x5 m
3	Receiving bin	1	1	-	$V = 60\ m^3$
4	Cylindrical feeder	2	2	-	Two rolls, Ø.6 m, L= 1 m
5	Active screen	2	2	-	Slot 20 mm, Size 1.5 x 4 m
6	Belt conveyor	4	4	-	Width 1 m
7	Vibrating feeding trough	2	2	-	Dia. 400 mm
8	Soaking tank	6	6	-	Dia. 8 m, agitator: Anker type RPM 30/min
9	Heavy vibrator	2	2		15 #, Size 1.5 x 4 m
10	Slurry distributor	2	2	-	Three outputs
11	Cloak sieve	6	6	-	Sieve area 10 m^2, 250 #
12	Rotary pan filter	2	2	-	Filter area 20 m^2
13	Centrifugal pump	2	1	1	$Q = 10\ 000\ m^3/h$
14	Dosage pump	4	2	2	Q = 5 - 100 l/min
15	Preparation tank	4	2	2	$V = 2\ m^3$
16	Centrifugal pump	2	1	1	$Q = 10\ 000\ m^3/h$
17	Tailing pond	2 *	2	-	$V = 600\ 000\ m^3$

* One new pond has to be added every year

V/C ALUMINA PLANT

When sizing the main equipment an average operating factor of 91 % was taken into consideration. This means that every equipment has to operate 8000 hours per year with the calculated load (or a little more with a slightly reduced one).

Raw material handling would require a capacity of 62; 135 and 135 t/h for Variant A, B and C. However, during the rainy season the required capacities would be 75; 162.5 and 148 t/h because of the higher moisture content of the ore. (The seasonal change of the moisture content of the beneficiated ore would probably be insignificant).

Digestion would require a capacity to handle 200 m^3/h bauxite slurry for Variant A and 400 m^3/h for Variants B and C. (The differences between the latter variants are just within error margins). Grinding and slurry handling would have a slurry flow amounting to one third of these figures. The 45 min retention time at the digestion could be provided by three digestors of 50 and 100 m^3 volume, respectively, plus 1 spare in each case. The digested slurry should be flashed in three stages, with about 4 and 8 tons of flash steam per stage, respectively. Recuperation preheaters would have heating surface areas of about 200 and 400 m^2 per stage, respectively. The required surface area of the heaters heated by fresh steam would also be about 200 and 400 m^2.

Calculating with a 20 kg/m^2h mud load for settlers and 40 kg/m^2h for washers, 5 pcs of equipment with areas of 460; 1100 and 910 m^2 (i.e. diameters between 24 and 38 m) would be required. Because of the high sand content of some samples (first of all CPT-2 types) a desanding system capable of handling at least 10 % of the solids (2,5 and 1 t/h, respectively) should be installed before the settlers and a smaller one (for, say, 1; 2.5 and 2 t/h, respectively) before the mud filters.

Calculating with a 100 kg/m^{2}h filtration load (dry basis) the plant would require mud filters with 182.5; 456.5 and 365.5 m^2 total filtration areas. It would be advisable to install 50 m^2 filters (4 operating + 1 spare) for 100,000 tpy and 100 m^2 filters (5 operating + 1 spare and 4 operating + 1 spare) for 200,000 tpy.

The pregnant liquor flow would be about 225 and 450 m^3/h, respectively. Therefore, control filters with 225 and 450 m^2 total filtration areas should be installed. For pregnant liquor cooling 800 and 1600 m^2 total plate heat exchanger surface areas would be required. 15 Precipitators with 1000 and 2000 m^3 volumes each would be required and one and two sets of primary, secondary and tray thickeners for hydrate classification, respectively. (Dia. 6 m for primaries, 12 m for secondaries and 30 m for trays). One spare set should be provided in both cases.

A 15 m^2 horizontal pan filter should be provided for product hydrate filtration for 100,000 tpy and a 30 m^2 one for 200,000 tpy capacity. Drum filters with a total surface area of 100 and 200 m^2 would be required for filtering and washing the coarse seed hydrate and ones with a surface area of 50 and 100 m^2 for the fine one. (At least 2 pcs in each case, i.e. 2x25 + 2x50 and 2x50 + 4 x 50 m^2, resp.). A 325 and a 650 t/d capacity stationary kiln would be required for calcination,respectively.

58, 131 and 129 t/h capacity evaporator sets would be required for average conditions. However, to cope with the higher moisture content of the ore during the wet season, 70: 158 and 141 t/h capacities should be installed for the three variants.

ALUMINA PLANT EQUIPMENT LIST

AREA EQUIPMENT	100 000 TPY NUMBER TOT	OP	SP	TECHNICAL DATA	200 000 TPY NUMBER TOT	OP	SP	TECHNICAL DATA
BAUXITE RECEIVING AND STORAGE								
1. Receiving hopper with apron belt	1	1	0	Cap: 315 t/h	2	2	0	Cap: 350 t/h
2. Belt conveyor to stockpile	1	1	0	Cap: 315 t/h	1	1	0	Cap: 700 t/h
3. Hopper with apron belt	1	1	0	Cap: 85 t/h	2	2	0	Cap: 85 t/h
4. Belt conveyor to grinding	1	1	0	Cap: 85 t/h	1	1	0	Cap: 170 t/h
5. Front end loader	1	1	0	Cap: 1,5 m^3	2	2	0	Cap: 1,5 m^3
WET GRINDING AND SLURRY STORAGE								
1. Reversible belt conveyor	1	1	0	Cap: 85 t/h	1	1	0	Cap: 170 t/h
2. Weighing belt	2	1	1	Cap: 85 t/h	2	1	1	Cap: 170 t/h
3. Ball mill (incl. el. motor, gearbox, lubrication system, conical screen)	2	1	1	2 m dia-2 m length	2	1	1	3 m dia-2 m length
4. Mill slurry tank	2	1	1	3.15 m dia-3.15 m height V=25 m^3, flat bottom, mech. agitation	2	1	1	4 m dia-4 m height V=50 m^3, flat bottom, mech. agitation

ALUMINA PLANT EQUIPMENT LIST

AREA EQUIPMENT	100 000 TPY				200 000 TPY			
	NUMBER TOT	OP	SP	TECHNICAL DATA	NUMBER TOT	OP	SP	TECHNICAL DATA
5. Mill slurry pump	2	1	1	Q= 70 m^3/h	2	1	1	Q=140 m^3/h
6. Slurry storage tank	2	2	0	6.3 m dia-6.3 m height V=200 m^3, flat bottom mech. agitation	2	2	0	6.3 m dia-14 m height V=400 m^3, flat bottom mech. agitation
7. Slurry heating condenser	2	1	1	1.4 m dia-4 m height barometric type	2	1	1	2 m dia-5 m height barometric type
8. Heated slurry tank	2	1	1	4 m dia-3 m s.s V=40 m^3, 90° cone bottom	2	1	1	5 m dia-4 m s.s V=90 m^3, 90° cone bottom
9. Slurry pump	4	2	2	Q=70 m^3/h	4	2	2	Q=140 m^3/h
10. Sump tank	1	-	-	2 m dia-2 m depth V=6.3 m^3, mech. agitation	1	-	-	2 m dia-2 m depth V=6.3 m^3, mech. agitation
11. Sump pump	1	-	-	Q=30 m^3/h	1	-	-	Q=30 m^3/h
12. EOT crane	1	-	-	Cap: 5 t	1	-	-	Cap: 5 t

ALUMINA PLANT EQUIPMENT LIST

AREA EQUIPMENT	100 000 TPY NUMBER TOT	OP	SP	TECHNICAL DATA	200 000 TPY NUMBER TOT	OP	SP	TECHNICAL DATA
DIGESTION, DILUTION								
1. Liquor heaters	5	5	0	A = 200 m^2, design pr.=20 bar, shell and tube type	5	5	0	A=400 m^2, design pr.=20 bar shell and tube type
2. Digester	4	3	1	V=50m^3, design pr.=12 bar agitated	4	3	1	V=100 m^3, design pr.=12 bar agitated
3. Flash tank	3	3	0	V=50 m^3, design pr=10 bar	3	3	0	V=100 m^3, design pr.=10 bar
4. Flashed slurry pump	2	1	1	Q=200 m^3/h	2	1	1	Q=400 m^3/h
5. Hydrocyclone	2	1	1	0.9 m dia	2	1	1	1.2 m dia
6. Condenser	1	1	0	1.4 m dia - 4 m height barometric type	1	1	0	2.0 dia - 5 m height barometric type
7. Diluting tank	2	2	0	6.3 m dia - 6.3 m height V=200 m^3, flat bottom mech. agitation	2	2	-	7 m dia - 11 m height V=400 m^3, flat bottom mech. agitation
8. Diluted slurry pump	2	1	1	Q=300 m^3/h	2	1	1	Q=600 m^3/h
9. Good condensate flash tank	1	1	0	V=2 m^3, design pr.=8 bar	1	1	0	V=4 m^3, design pr.=8 bar
10. Good condensate receiver	1	1	0	V=40 m^3, design pr.=8 bar	1	1	0	V=80 m^3, design pr.=8 bar
11. Good condensate pump	2	1	1	Q=30 m^3/h	2	1	1	Q=60 m^3/h
12. Alkaline condensate receiver	1	1	0	V=20 m^3, design pr.=8 bar	1	1	0	V=40 m^3, design pr.=8 bar
13. Alkaline condensate pump	2	1	1	Q=20 m^3/h	2	1	1	Q=40 m^3/h
14. Rake classifier	3	3	0	1.2 m width-8.4 m length	3	3	0	2.4 m width-8.4 m length
15. Sand tank	1	1	0	3 m dia-2 m s.s. V=17 m^3, 90°cone bottom	1	1	0	3 m dia-2 m s.s, V=17 m^3, 90°cone bottom
16. Sand disposal pump	2	1	1	Q=10 m^3/h	2	1	1	Q=20 m^3/h

ALUMINA PLANT EQUIPMENT LIST

AREA EQUIPMENT	100 000 TPY				200 000 TPY			
	NUMBER			TECHNICAL DATA	NUMBER			TECHNICAL DATA
	TOT	OP	SP		TOT	OP	SP	
17. Interstage pump	4	2	2	$Q=30\ m^3/h$	4	2	2	$Q=60\ m^3/h$
18. Sump tank	2	-	-	2 m dia-2 m depth $V=6.3\ m^3$, mech.agitation	2	-	-	2 m dia-2 m depth $V=6.3\ m^3$, mech.agitation
19. Sump pump	2	-	-	$Q=30\ m^3/h$	2	-	-	$Q=30\ m^3/h$
CLARIFICATION, RED MUD WASHING AND FILTRATION								
1. Settler	2	2	0	24 m dia-4.5 m s.s. covered, cable torque rake	2	2	0	38 m dia-4.5 m s.s. covered, cable torque rake
2. Settler overflow tank	2	2	0	6.3 m dia-6.3 m height $V=200\ m^3$, flat bottom mech. agitation	2	2	0	10 m dia-6 m height $V=500\ m^3$, flat bottom side agitation
3. Settler overflow pump	2	1	1	$Q=250\ m^3/h$	2	1	1	$Q=500\ m^3/h$
4. Settler undeflow pump	4	2	2	$Q=30\ m^3/h$	4	2	2	$Q=60\ m^3/h$
5. Washer feed tank	3	3	0	3 m dia-3.2 m height $V=15\ m^3$, 90° cone bottom	3	3	0	3 m dia-3.2 m height $V=15\ m^3$, 90° cone bottom
6. Washer	3	3	0	24 m dia-4.5 m s.s covered, cable troque rake	3	3	0	38 m dia-4.5 m s.s covered, cable torque rake
7. First washer overflow tank	1	1	0	6.3 m dia-6.3 m height $V=200\ m^3$, flat bottom	1	1	0	10 m dia-6 m height $V=500\ m^3$, flat bottom
8. First washer overflow pump	2	1	1	$Q=80\ m^3/h$	2	1	1	$Q=180 m^3/h$
9. 2nd and 3rd washer overflow tank	2	2	0	2 m dia-6 m height $V=15\ m^3$, 90° cone bottom	2	2	0	2 m dia-6 m height $V=15\ m^3$, 90° cone bottom
10. 2nd and 3rd washer overflow pump	4	2	2	$Q=80\ m^3/h$	4	2	2	$Q=180\ m^3/h$

ALUMINA PLANT EQUIPMENT LIST

AREA EQUIPMENT	100 000 TPY NUMBER TOT	OP	SP	TECHNICAL DATA	200 000 TPY NUMBER TOT	OP	SP	TECHNICAL DATA
11. Washer underflow pump	6	3	3	$Q=60\ m^3/h$	6	3	3	$Q=120\ m^3/h$
12. Flocculant preparing and dosing unit	1	-	-	As per the supplier	1	-	-	As per the supplier
13. Vibrating screens	3	2	1	$A=0.4\ m^2$	3	2	1	$A=0.8\ m^2$
14. Red mud slurry tank	2	2	0	5 m dia-5 m height $V=100\ m^3$, flat bottom mech. agitation	2	2	0	6.3 m dia-6.3 m height $V=200\ m^3$, flat bottom mech. agitation
15. Red mud slurry pump	3	2	1	$Q=100\ m^3/h$	3	2	1	$Q=200\ m^3/h$
16. Vacuum drum filter, incl. filtrate receivers	5	4	1	$A=50\ m^2$, roller discharge	6	5	1	$A=100\ m^2$, roller discharge
17. Screw conveyor	5	4	1	Cap: 15 t/h	6	5	1	Cap: 25 t/h
18. Reactor agitator	3	2	1	As per the supplier	3	2	1	As per the supplier
19. H.P. piston pump	2	1	1	$Q=60\ m^3/h$; p=50 bar	2	1	1	$Q=130\ m^3/h$; p=50 bar
20. Filtrate tank	1	1	0	4 m dia-4 m height $V=50\ m^3$, flat bottom	1	1	0	5 m dia-4 m height $V=80\ m^3$, flat bottom
21. Filtrate pump	2	1	1	$Q=80\ m^3/h$	2	1	1	$Q=180\ m^3/h$
22. Wash Water tank	1	1	0	6.3 m dia-6.3 m height $V=200\ m^3$, flat bottom	1	1	0	10 m dia-6 m height $V=500\ m^3$, flat bottom
23. Wash water pump	2	1	1	$Q=80\ m^3/h$	2	1	1	$Q=180\ m^3/h$
24. Condenser	1	1	0	1.4 m dia-4 m height barometric type	1	1	0	2 m dia-5 m height barometric type
25. Hot well tank	1	1	0	3.15 m dia-3.15 m height $V=25\ m^3$, flat bottom	1	1	0	4 m dia-4 m height $V=50\ m^3$, flat bottom
26. Cooling water pump	2	1	1	$Q=80\ m^3/h$	2	1	1	$Q=160\ m^3/h$

ALUMINA PLANT EQUIPMENT LIST

AREA EQUIPMENT	100 000 TPY NUMBER TOT	 OP	 SP	 TECHNICAL DATA	200 000 TPY NUMBER TOT	 OP	 SP	 TECHNICAL DATA
27. Vacuum pump	3	2	1	Q=4000 m^3/h, p=380 Hgmm	3	2	1	Q=10000 m^3/h, p=380 Hgmm
28. Air blower	2	1	1	Q=1000 m^3/h, p=0.3 barg	2	1	1	Q=2500 m^3/h, p = 0.3 barg
29. Sump tank	4	-	-	2 m dia-2 m depth V=6.3 m^3, mech.agitation	4	-	-	2 m dia-2 m depth V=6.3 m^3, mech.agitation
30. Sump pump	4	-	-	Q=30 m^3/h	4	-	-	Q=30 m^3/h
31. EOT crane	1	-	-	Cap: 5 t	1	-	-	Cap: 5 t
32. Mobil H.P. Water pump	1	-	-	Q=6 m^3/h; p=140 bar	1	-	-	Q=6 m^3/h; p=140 bar
CONTROL FILTRATION, ALUMINATE LIQUOR COOLING								
1. Press filters	3	2	1	A=125 m^2	3	2	1	A=250 m^2
2. Repulpers	3	2	1	0.6 m width-6 m length	3	2	1	0.6 m width-6 m length
3. Filtrate tank	2	2	0	6.3 m dia-6.3 m height V=200 m^3, flat bottom, mech. agitation	2	2	0	10 m dia-6 m height V=500 m^3, flat bottom, side agitation
4. Filtrate pump	2	1	1	Q=250 m^3/h	2	1	1	Q=500 m^3/h
5. Lime slurry preparing tank	1	1	0	3.15 m dia-3.15 m height V=25 m^3, flat bottom, mech. agitation	1	1	0	4 m dia-4 m height V=50 m^3, flat bottom, mech. agitation
6. Lime slurry pump	2	1	1	Q=15 m^3/h	2	1	1	Q=30 m^3/h
7. Precoat preparing tank	2	2	0	3.15 m dia-3.15 m height V=25 m^3, flat bottom, mech. agitation	2	2	0	4 m dia-4 m height V=50 m^3, flat bottom mech. agitation
8. Precoat pump	2	1	1	Q=100 m^3/h	2	1	1	Q=200 m^3/h

ALUMINA PLANT EQUIPMENT LIST

AREA / EQUIPMENT	100 000 TPY NUMBER TOT	OP	SP	TECHNICAL DATA	200 000 TPY NUMBER TOT	OP	SP	TECHNICAL DATA
9. Waste precoat tank	1	1	0	3.15 m dia-3.15 m height $V=25\ m^3$, flat bottom mech. agitation	1	1	0	4 m dia-4 m height $V=50\ m^3$, flat bottom, mech. agitation
10. Waste precoat pump	2	1	1	$Q=15\ m^3/h$	2	1	1	$Q=30\ m^3/h$
11. Caustic tank	1	1	0	3.15 m dia-3.15 m height $V=25\ m^3$, flat bottom, mech. agitation	1	1	0	4 m dia-4 m height $V=50\ m^3$, flat bottom, mech. agitation
12. Caustic circulating pump	1	1	0	$Q=50\ m^3/h$	1	1	0	$Q=100\ m^3/h$
13. Plate heat exchanger	3	2	1	$A=400\ m^2$	5	4	1	$A=400\ m^2$
14. Cooled aluminate liquor tank	1	1	0	6.3 m dia-6.3 m height $V=200\ m^3$, flat bottom	1	1	0	10 m dia-6 m height $V=500\ m^3$, flat bottom
15. Cooled aluminate liquor pump to seed slurrying	3	2	1	$Q=80\ m^3/h$	3	2	1	$Q=160\ m^3/h$
16. Precipitation feed pump	2	1	1	$Q=150\ m^3/h$	2	1	1	$Q=300\ m^3/h$
17. Hot spent liquor tank	1	1	0	6.3 m dia-6.3 m height $V=200\ m^3$, flat bottom	1	1	0	10 m dia-6 m height $V=500\ m^3$, flat bottom
18. Hot spent liquor pump to evaporation	2	1	1	$Q=200\ m^3/h$	2	1	1	$Q=400\ m^3/h$
19. Hot spent liquor pump to clarification	2	1	1	$Q=25\ m^3/h$	2	1	1	$Q=50\ m^3/h$
20. Sump tank	2	-	-	2 m dia-2 m depth $V=6.3\ m^3$, mech.agitation	2	-	-	2 m dia-2 m depth $V=6.3\ m^3$, mech.agitation
21. Sump pump	2	-	-	$Q=30\ m^3/h$	2	-	-	$Q=30\ m^3/h$
22. EOT crane	1	-	-	Cap: 1 t	1	-	-	Cap: 1 t

ALUMINA PLANT EQUIPMENT LIST

AREA / EQUIPMENT	100 000 TPY NUMBER TOT	OP	SP	100 000 TPY TECHNICAL DATA	200 000 TPY NUMBER TOT	OP	SP	200 000 TPY TECHNICAL DATA
PRECIPITATION, HYDRATE CLASSIFICATION								
1. Precipitator	17	15	2	8 m dia-23 m height $V=1000\ m^3$, cone bottom, air agitation	17	15	2	10 m dia-33 m height $V=2000\ m^3$, cone bottom, air agitation
2. Interstage transfer pump	2	1	1	$Q=300\ m^3/h$	2	1	1	$Q=580\ m^3/h$
3. Interstage cooler	2	1	1	$A=70\ m^2$, spiral heat exchanger type	2	1	1	$A=130\ m^2$, spiral heat exchanger type
4. Cooling water tank	1	1	0	3.15 m dia-3.15 m height $V=25\ m^3$, flat bottom	1	1	0	3.15 m dia-3.15 m height $V=25\ m^3$, flat bottom
5. Cooling water pump	2	1	0	$Q=60\ m^3/h$	2	1	1	$Q=120\ m^3/h$
6. Transfer pump	2	1	1	$Q=300\ m^3/h$	2	1	1	$Q=580\ m^3/h$
7. Precipitator discharge pump	2	1	1	$Q=300\ m^3/h$	2	1	1	$Q=580\ m^3/h$
8. Caustic heater	1	1	0	$A=200\ m^2$, design pr=10 bar shell and tube type	1	1	0	$A=400\ m^2$, design pr=10 bar shell and tube type
9. Caustic circulating pump	1	1	0	$Q=100\ m^3/h$	1	1	0	$Q=200\ m^3/h$
10 Primary thickener (P.T.)	2	1	1	6 m dia-15 m s.s. 60° cone bottom	3	2	1	6 m dia-15 m s.s. 60° cone bottom
11. P.T. underflow pump	4	1	3	$Q=30\ m^3/h$	6	2	4	$Q=30\ m^3/h$
12. Product hydrate storage tank	1	1	0	6 m dia-15 m s.s. 60° cone bottom	2	2	0	6 m dia-15 m s.s. 60° cone bottom
13. Product hydrate charge pump	2	1	1	$Q=30\ m^3/h$	2	1	1	$Q=60\ m^3/h$
14. Secondary thickener (S.T.)	2	1	1	12 m dia-4 m s.s. 60° cone bottom	3	2	1	12 m dia-4 m s.s. 60° cone bottom

ALUMINA PLANT EQUIPMENT LIST

AREA / EQUIPMENT	100 000 TPY				200 000 TPY			
	NUMBER			TECHNICAL DATA	NUMBER			TECHNICAL DATA
	TOT	OP	SP		TOT	OP	SP	
15. S.T. underflow pump	4	1	3	Q=60 m^3/h	6	2	4	Q=60 m^3/h
16. Tertiary thickener (T.T.)	2	1	1	30 m dia-4 m s.s. covered, 1:3 slope bottom, cable torque rake	3	2	1	30 m dia-4 m s.s. covered, 1:3 slope bottom, cable torque rake
17. T.T. underflow pump	4	1	3	Q=40 m^3/h	6	2	4	Q=40 m^3/h
18. Spent liquor tank	2	2	0	6.3 m dia-6.3 m height V=200 m^3, flat bottom	2	2	0	10 m dia-6 m height V=500 m^3, flat bottom
19. Spent liquor pump	2	1	1	Q=250 m^3/h	2	1	1	Q=450 m^3/h
20. Sump tank	5	-	-	2 m dia-2 m depth V=6.3 m^3; mech. agitation	5	-	-	2 m dia-2 m depth V=6.3 m^3; mech. agitation
21. Sump pump	5	-	-	Q=30 m^3/h	5	-	-	Q=30 m^3/h
22. Elevator	1	-	-	Cap: 1 t	1	-	-	Cap: 1 t
HYDRATE FILTRATION								
OXALATE REMOVAL								
1. Pan filter for product hydrate filtration, incl. filtrate receivers	1	1	0	A=15 m^2	1	1	0	A=30 m^2
2. Wash water and filtrate tank	4	4	0	3.15 m dia-3.15 m height V=25 m^3, flat bottom	4	4	0	4 m dia-4 m height V=50 m^3, flat bottom
3. Wash water and filtrate pump	8	4	4	Q=30 m^3/h	8	4	4	Q=60 m^3/h
4. Vacuum drum filter for S.T. seed filtration, incl. filtrate receivers	2	2	0	A=50 m^2	4	4	0	A=50 m^2
5. S.T. seed slurry tank	2	2	0	3.15 m dia-3.15 m height V=25 m^3, flat bottom, mech. agitation	2	2	0	4 m dia-4 m height V=50 m^3, flat bottom, mech. agitation

ALUMINA PLANT EQUIPMENT LIST

AREA EQUIPMENT	100 000 TPY				200 000 TPY			
	NUMBER			TECHNICAL DATA	NUMBER			TECHNICAL DATA
	TOT	OP	SP		TOT	OP	SP	
6. S.T. seed slurry charge pump	4	2	2	Q=60 m^3/h	4	2	2	Q=120 m^3/h
7. Vacuum drum filter for T.T. seed filtration, incl. filtrate receivers	2	2	0	A=25 m^2	2	2	0	A=50 m^2
8. T.T. seed slurry tank	2	2	0	3.15 m dia-3.15 m height V=25 m^3, flat bottom, mech. agitation	2	2	0	4 m dia-4 m height V=50 m^3, flat bottom, mech. agitation
9. T.T. seed slurry charge pump	4	2	2	Q=40 m^3/h	4	2	2	Q=80 m^3/h
10. First filtrates tank	1	1	0	3.15 m dia-3.15 m height V=25 m^3, flat bottom	1	1	0	4 m dia-4 m height V=50 m^3, flat bottom
11. First filtrates pump	2	1	1	Q=80 m^3/h	2	1	1	Q=160 m^3/h
12. Second filtrates tank	1	1	0	3.15 m dia-3.15 m height V=25 m^3, flat bottom	1	1	0	4 m dia-4 m height V=50 m^3, flat bottom
13. Second filtrates pump	2	1	1	Q=35 m^3/h	2	1	1	Q=70 m^3/h
14. Condenser	1	1	0	1.4 m dia-4 m height barometric type	1	1	0	2 m dia-5 m height barometric type
15. Hot well tank	1	1	0	2 m dia-2 m height V=6.3 m^3, flat bottom	1	1	0	3.15 m dia-2 m height V=16 m^3, flat bottom
16. Cooling water pump	2	1	1	Q=100 m^3/h	2	1	1	Q=200 m^3/h
17. Vacuum pump	2	1	1	Q=8000 m^3/h, p=380 Hgmm	3	2	1	Q=8000 m^3/h, p=380 Hgmm
18. Air blower	2	1	1	Q=2000 m^3/h, p=0.5 barg	3	2	1	Q=2000 m^3/h, p=0.5 barg
19. Lime milk preparing tank	2	2	0	3.15 m dia-3.15 m height V=25 m^3, flat bottom, mech. agitation	2	2	0	3.15 m dia-3.15 m height V=25 m^3, flat bottom, mech. agitation
20. Lime milk pump	2	1	1	Q=10 m^3/h	2	1	1	Q=20 m^3/h

ALUMINA PLANT EQUIPMENT LIST

AREA EQUIPMENT	100 000 TPY NUMBER TOT	OP	SP	TECHNICAL DATA	200 000 TPY NUMBER TOT	OP	SP	TECHNICAL DATA
21. Causticising tank	1	1	0	4 m dia-4 m height $V=50\ m^3$, flat bottom, mech. agitation	1	1	0	5 m dia-5 m height $V=100\ m^3$, flat bottom mech. agitation
22. Causticised filtrate pump	1	1	0	$Q=50\ m^3/h$	1	1	0	$Q=100\ m^3/h$
23. Settler	1	1	0	8 m dia-4m s.s. covered, cable torque rake	1	1	0	12 m dia-4 m s.s. covered, cable torque rake
24. Settler overflow tank	1	1	0	4 m dia-4 m height $V=50\ m^3$, flat bottom	1	1	0	5 m dia-5 m height $V=100\ m^3$, flat bottom
25. Settler overflow pump	1	1	0	$Q=40\ m^3/h$	1	1	0	$Q=80\ m^3/h$
26. Settler underflow pump	2	1	1	$Q=10\ m^3/h$	2	1	1	$Q=20\ m^3/h$
27. Sump tank	3	-	-	2 m dia-2 m depth $V=6.3\ m^3$, mech. agitation	3	-	-	2 m dia-2 m depth $V=6.3\ m^3$, mech. agitation
28. Sump pump	3	-	-	$Q=30\ m^3/h$	3	-	-	$Q=30\ m^3/h$
29. EOT crane	1	-	-	Cap: 1 t	1	-	-	Cap: 1 t
CALCINATION, ALUMINA STORAGE AND LOADING								
1. Calcining kiln (incl. hydrate preheater, alumina cooler, dust collector, feed, discharge and other auxiliary equipment)	1	1	0	Cap: 325 t/day Al_2O_3, fluid type	1	1	0	Cap: 650 t/day Al_2O_3, fluid type
2. Cooling water tank	1	1	0	2 m dia-2 m height $V=6.3\ m^3$, flat bottom	1	1	0	2 m dia-2 m height $V=6.3\ m^3$, flat bottom
3. Cooling water pump	4	2	2	$Q=30\ m^3/h$	4	2	2	$Q=60\ m^3/h$
4. Alumina transporting system to silo	1	1	0	Cap: 14 t/h	1	1	0	Cap: 28 t/h

ALUMINA PLANT EQUIPMENT LIST

AREA EQUIPMENT	100 000 TPY NUMBER TOT	 OP	 SP	 TECHNICAL DATA	200 000 TPY NUMBER TOT	 OP	 SP	 TECHNICAL DATA
5. Alumina silo (incl. air slides at bottom)	1	1	0	Cap: 3000 t	1	1	0	Cap: 6000 t
6. Silo discharge and railtanker loading facilities	1	1	0	Cap: 60 t/h	1	1	0	Cap: 120 t/h
EVAPORATION, CAUSTIC HANDLING								
1. Evaporator unit (incl. pumps, control system and other auxiliary equipment)	1	1	0	Cap: 70 t/h evaporated water, counter-current, falling film type	1	1	0	Cap: 160 t/h evaporated water, counter-current, falling film type
2. Spent liquor tank	2	2	0	10 m día-6 m height $V=500\ m^3$, flat bottom, side agitation	2	2	0	11.3 m día-10.5 m height $V=1000\ m^3$, flat bottom, side agitation
3. Strong liquor tank	1	1	0	10 m día-6 m height $V=500\ m^3$, flat bottom, side agitation	1	1	0	11.3 m día-10.5 m height $V=1000\ m^3$, flat bottom, side agitation
4. Strong liquor pump	2	1	1	$Q=30\ m^3/h$	2	1	1	$Q=60\ m^3/h$
5. Test tank	2	2	0	6.3 m día-6.3 m height $V=200\ m^3$, flat bottom, mech. agitation	2	2	0	10 m día-6 m height $V=500\ m^3$, flat bottom, side agitation
6. Test tank liquor pump	2	1	1	$Q=160\ m^3/h$	2	1	1	$Q=320\ m^3/h$
7. Wash water tank	1	1	0	6.3 m día-6.3 m height $V=200\ m^3$, flat bottom, mech. agitation	1	1	0	10 m día-6 m height $V=500\ m^3$, flat bottom, side agitation
8. Alk. condensate tank	1	1	0	6.3 m día-6.3 m height $V=200\ m^3$, flat bottom	1	1	0	10 m día-6 m height $V=500\ m^3$, flat bottom
9. Alk. condensate pump to red-mud filtration	2	1	1	$Q=80\ m^3/h$	2	1	1	$Q=180\ m^3/h$

ALUMINA PLANT EQUIPMENT LIST

AREA EQUIPMENT	100 000 TPY NUMBER TOT	OP	SP	100 000 TPY TECHNICAL DATA	200 000 TPY NUMBER TOT	OP	SP	200 000 TPY TECHNICAL DATA
10. Alk condensate pump to hydrate filtration and oxalate removal	2	1	1	Q=35 m^3/h	2	1	1	Q=70 m^3/h
11. Hot well tank	1	1	0	4 m dia-4 m height V=50 m^3, flat bottom	1	1	0	5 m dia-4 m height V=80 m^3, flat bottom
12. Booster pump	2	1	1	Q=50 m^3/h	2	1	1	Q=100 m^3/h
13. Cleaning water pump	1	-	-	Q=40 m^3/h	1	-	-	Q=80 m^3/h
14. Caustic unloading and transfering pump	2	-	-	Q=100 m^3/h	2	-	-	Q=200 m^3/h
15. Caustic storage tank	2	2	0	11.3 m dia-10.5 m height V=1000 m^3, flat bottom	2	2	0	16 m dia-10 m height V=2000 m^3, flat bottom
16. Make-up caustic pump	2	1	1	Q=7 m^3/h	2	1	1	Q=15 m^3/h
17. Acid receiving tank	1	1	0	5 m dia-5 m height V=100 m^3, flat bottom	1	1	0	5 m dia-5 m height V=100 m^3, flat bottom
18. Acid transfer pump	1	1	0	Q=10 m^3/h	1	1	0	Q=10 m^3/h
19. Diluted acid tank	1	1	0	5 m dia-5 m height V=100 m^3, flat bottom	1	1	0	5 m dia-5 m height V=100 m^3, flat bottom
20. Diluted acid pump	1	1	0	Q=60 m^3/h	1	1	0	Q=120 m^3/h
21. Mobile acid tank and pump	1	-	-	As per the supplier	1	-	-	As per the supplier
22. Sump tank	3	-	-	2 m dia-2 m depth V=6.3 m^3; mech. agitation	3	-	-	2 m dia-2 m depth V=6.3 m^3; mech. agitation
23. Sump pump	3	-	-	Q=30 m^3/h	3	-	-	Q=30 m^3/h
24. EOT crane	1	-	-	Cap: 2 t	1	-	-	Cap: 3 t

ALUMINA PLANT EQUIPMENT LIST

AREA / EQUIPMENT	100 000 TPY NUMBER TOT	OP	SP	TECHNICAL DATA	200 000 TPY NUMBER TOT	OP	SP	TECHNICAL DATA
COMPRESSOR STATION								
1. Turbocompressor (incl. electric motor, gearbox, coolers, lubricating system and control equipment)	2	1	1	Cap: 4000 m^3/h; p =7 barg	3	2	1	Cap: 4000 m^3/h; p=7 barg
2. Air strainer	2	1	1	Cap: 4000 m^3/h	3	2	1	Cap: 4000 m^3/h
3. Air dryer (for instrument air)	2	1	1	Cap: 200 m^3/h	2	1	1	Cap: 250 m^3/h
4. Air tank (for instrument air)	3	3	0	V=5 m^3, p=7 barg	3	3	0	V=7 m^3, p=7 barg
5. EOT crane	1	-	-	Cap: 8 t	1	-	-	Cap: 10 t
STEAM BOILER PLANT								
Complete industrial steam boiler plant								
1. Steam boiler	3	2	1	Cap: 20 t/h steam p=16 bar; t = 210°C	3	2	1	Cap: 45 t/h steam p=16 bar; t = 210°C
WATER COOLING, SEWAGE TREATMENT								
1. Water intake pump	2	1	1	Q=100 m^3/h	2	1	1	Q=200 m^3/h
2. Cooling tower	1	1	0	Cap: 1500 m^3/h	1	1	0	Cap: 3000 m^3/h
3. Circulating pump	3	2	1	Q=750 m^3/h	6	4	2	Q=750 m^3/h
4. Cooled water pump	3	2	1	Q=750 m^3/h	6	4	2	Q=750 m^3/h
5. Warm water transfer pump	3	2	1	Q=650 m^3/h	6	4	2	Q=650 m^3/h
6. Sewage treatment facilities (incl. sewage purification basin, settler, sewage pump and other auxiliary equipment)	1	1	0	Cap:10 m^3/h	1	1	0	Cap:10 m^3/h

EQUIPMENT LIST

(WORKSHOP, MOTOR VEHICLES AND LABORATORY)

EQUIPMENT	NUMBER	TECHNICAL DATA
WORKSHOP		
1. Vertical boring and turning machine	1	1,450 mm dia x 865 mm
2. Engine lathe	3	650-400 mm dia x 5,000-1,500 mm
3. Universal instrument lathe	2	250-200 mm dia x 600-350 mm
4. Turret lathe	1	400 mm dia x 40 mm
5. Horizontal drilling-milling machine	1	100 mm dia - 1,120 x 1,250 mm
6. Universal milling machine	2	400-250 mm x 1,650 - 1,000 mm
7. Vertical milling machine	1	250 mm x 1,000 mm
8. Parallel planing machine	1	800 mm x 3,000 mm
9. Shaping machine	1	710 mm x 450 mm
10. Hydraulic slotting machine	1	800 mm dia x 120-500 mm
11. Multiple profiling machine	1	
12. Universal cylindrical grinding machine	2	500 mm x 1,500 mm
13. Universal tool grinding machine	1	
14. Radial drilling machine	1	75 mm dia x 2,000 mm
15. Upright drilling machine	1	50 mm dia
16. Portable radial drilling machine	2	25 mm dia
17. Framed saw	2	270 mm x 250 mm
18. Hydraulic press	1	
19. Hydraulic horizontal combination die	1	
20. Vulcanizer set	1	for 1,400 mm widht belts
21. Laying out bench	3	1,600 mm x 1,000 mm
22. Manual operated pipe bending machine	2	up to 50 mm dia pipe
23. Hand pump for hydraulic test	2	
24. Smith's carth., twin	1	2,000 mm - 1,000 mm
25. Air fo.,.ng hammer	1	
26. Carbide speeder	1	
27. Coiler	1	
28. Impregnating unit (vacuum)	1	
29. Control device of rotary	1	
30. Table drilling machine	4	
31. Table grinding machine	3	
32. Pedestal grinding machine	2	

EQUIPMENT LIST

(WORKSHOP, MOTOR VEHICLES AND LABORATORY)

EQUIPMENT	NUMBER	TECHNICAL DATA
33. Welding machine	3	
34. Portable welding machine	8	
35. Overhead crane	1	Cap. = 8 t
36. Overhead crane	1	Cap. = 4 t
37. Semi-gantry crane	1	Cap. = 20/5 t
38. Vice benches, supporting grates, cases		
MOTOR VEHICLE		
1. Autocrane	1	Cap. = 10 t
2. Autocrane	1	Cap. = 6.3 t
3. Truck	3	Cap. = 3.5 t
4. Dumper	1	Cap. = 6 m^3
5. Electric truck with platform	2	Cap. = 2 t
6. Electric truck with platform	2	Cap. = 3 t
7. Electric truck with turnable fork	2	Cap. = 2 t
8. Electric truck with fork	1	Cap. = 2 t
9. Truck trailer	2	Cap. = 1 t
10. Truck trailer	2	Cap. = 2 t
LABORATORY FOR PROCESS CONTROL		
1. Bauxite analyser using the neutron activation principle for the quick determination of Al_2O_3 and SiO_2	1	
2. X-ray diffractometer equipment with vacuum spectrograph	1	
3. Atomic absorption spectrophotometer	1	
4. Alumina thermoquant unit (measuring on the thermometric principle for quick series-analyses of aluminate liquor)	1	
5. Spectrophotometer	1	
6. Photoelectric equipment for determining grain-size distribution	1	minus 60 microns
7. Vibrating machine for sieves	2	
8. Set of sieves for determining grain-size distribution	4	plus 60 microns

EQUIPMENT LIST

(WORKSHOP, MOTOR VEHICLES AND LABORATORY)

EQUIPMENT	NUMBER	TECHNICAL DATA
9. Fine-crushing equipment	1	Cap. = 175 kg/h Input grain-size: 50 mm Product grain-size: 2 mm
10. Laboratory fine-crusher equipment	1	Cap. = 1 kg/h
11. Fine-grinding (powdering) equipment with steel lining		Product size: minus 60 microns
a. with two places	1	Cap. = 1 kg/h total
b. with six places	1	Cap. = 10-20 g/each
12. Centrifuge (steel)	1	6 x 400 - 500 ml units
13. Drying oven	5	Controllable in temperature range of + 25 °C to 350 °C
14. Ignition furnace	2	Controllable up to 1,200 °C
15. Analytical scale	4	Max.load: 200 g, Sensitivity: 0.1 mg
16. Analytical quick scale	3	Max.load: 500 g, Sensitivity: 0.1 mg
17. Physical balance	2	Max.load: 5 kg
18. PH-meters with platinium normal calomel resp. F^- and Cl^--ion sensitive electrodes	2	
19. Laboratory bomb digester with oil bath	1	6 bombs of 250 ml or 300 ml capacity, temperature control up to 260 °C
20. Sorptometer for the determination of surface area (PERKIN-ELNER 212D)	1	Surface area can be determined from about 0.1 m^2/g up to about 1,000 m^2/g
21. Dewar vessel	2	

Conclusions and Recommendations

- The technological tests and subsequent calculations have shown that the better qualities ("good" quality represented by sample CPT-1 and "submarginal" quality represented by sample CPT-2) of the OSTUACAN bauxitic laterite can be processed to alumina according to the Bayer process with acceptable specific consumption figure.

- The "low quality" material (represented by sample CPT-3) can also be processed, however only after beneficiation.

- The actual cost of production will be only calculated in the Pre-feasibility Study to be prepared by the Consejo de Recursos Minerales, however it can be stated even in the present stage that the whole project will stand or fall by the question whether sufficient amounts of "good" quality material can be found for a 20 year supply of a 100,000 tpy minimum size alumina plant. This would be require 6 million tons of "probable" good quality ore (dry basis), at least half of it falling into "proven" category. Should the answer to this question "yes" the use of "submarginal" and "beneficiated low qality" material could make possible to construct the alumina plant for a more economical size of about 200,000 tpy.

- It is recommended that the Consejo de Recursos Minerales prepare the Pre-feasibility Study based partly on the data included in the present Report, partly on the other written materials enumerated in the Introduction prepared by the Hungarian experts. This Study would show whether any of the varriants taken into consideration could be financially viable under Mexican economic condition.

- The prospection and surveying of the bauxite area are urget tasks as presently only a franction of the required amount of bauxite corresponds to the proven and probable categories and the total area is not explored even on a reconnaissance level. This would require contour maps (at least 1:5,000) of the productive areas and of their vicinity. The use of high capacity mechanized core-drilling equipment are needed.

- Beneficiation experiments carried out up to now, are not regarded as completed ones. They should be continued in the ALUTERV-FKI laboratory to learn which layer(s) of the laterite profile can be beneficiated with more efficiency than the composite samples. For this, individaual samples are needed.

- In order to improve the economy of the alumina industy it is suggested to investigate the question of aluminium sulfate manufacturing. Even though the present contract (UD/UC/Mex/86/203) does not cover this topic, it has to be mentioned, that aluminium sulfate is manufactured in Hungary (and over most of the world) on an alumina hydrate basis; the manufacturing process is widely known. It is important to raise this question, the more so, since if the quantity of the Mexican bauxite reserves were not sufficient to cover the requirements of an alumina industry, it would be necessary to investgate the possibility of manufacturing aluminium sulfate directly from bauxite from the points of view of the technology and the economy as well. According to our informations such a technology has already developed is the GDR, at least at a laboratory level. The requirement of working out this topic has to be formulated in a separate contract.

- For the future, the ALUTERV-FKI intents to continue its colaboration with the Consejo de Recursos Minerales in the same frame as it was done up to now, further ALUTERV-FKI is ready to receive Mexican expert(s) to introduce him (them) to the alumina industy and demonstrate the most significant technological and beneficiation tests were carried out for this Study.