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ASSISTANCE TO THE DEVELOPMENT OF THE PHOSPHATE INDUSTRY

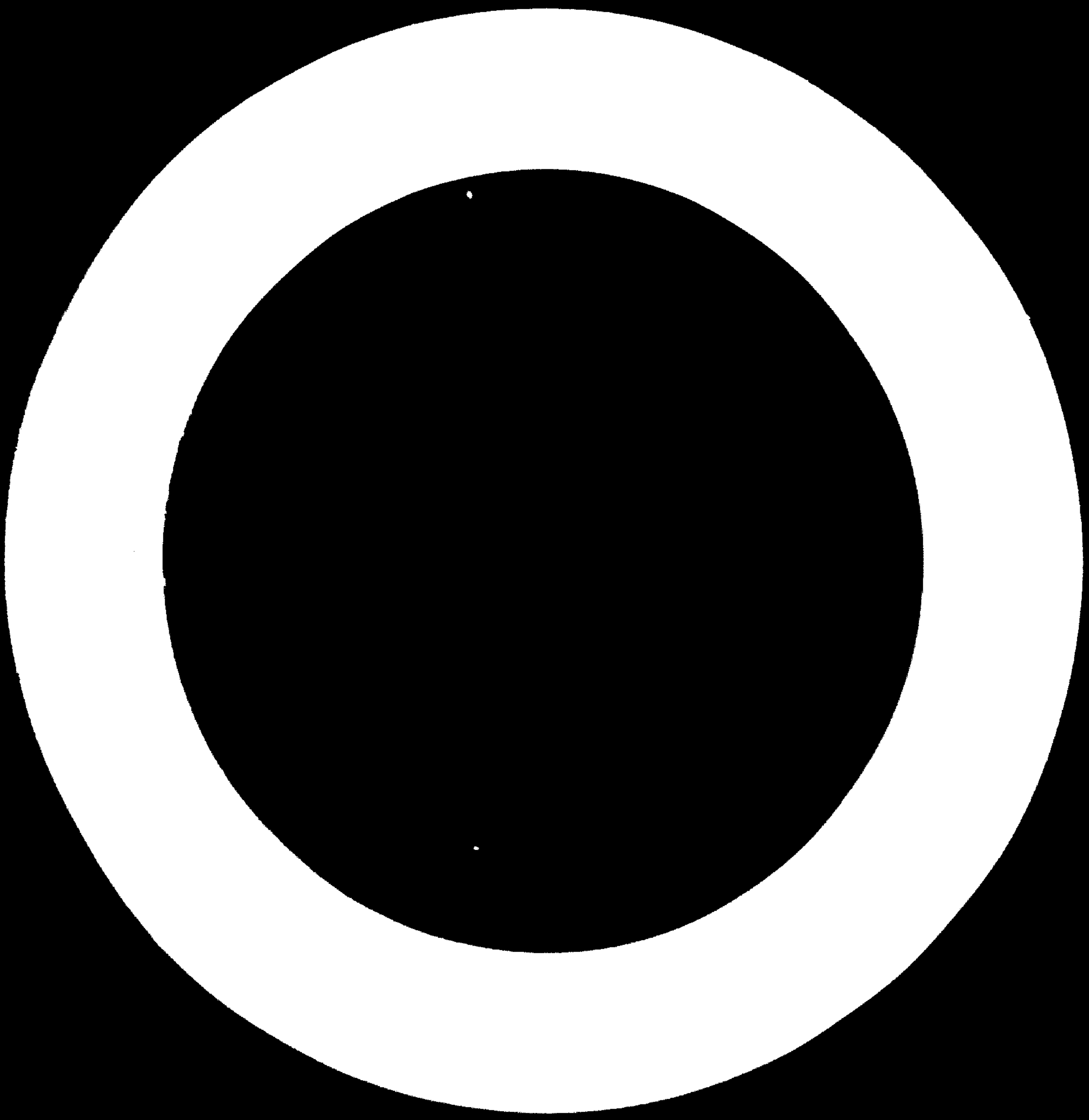
IS/SYR/71/813

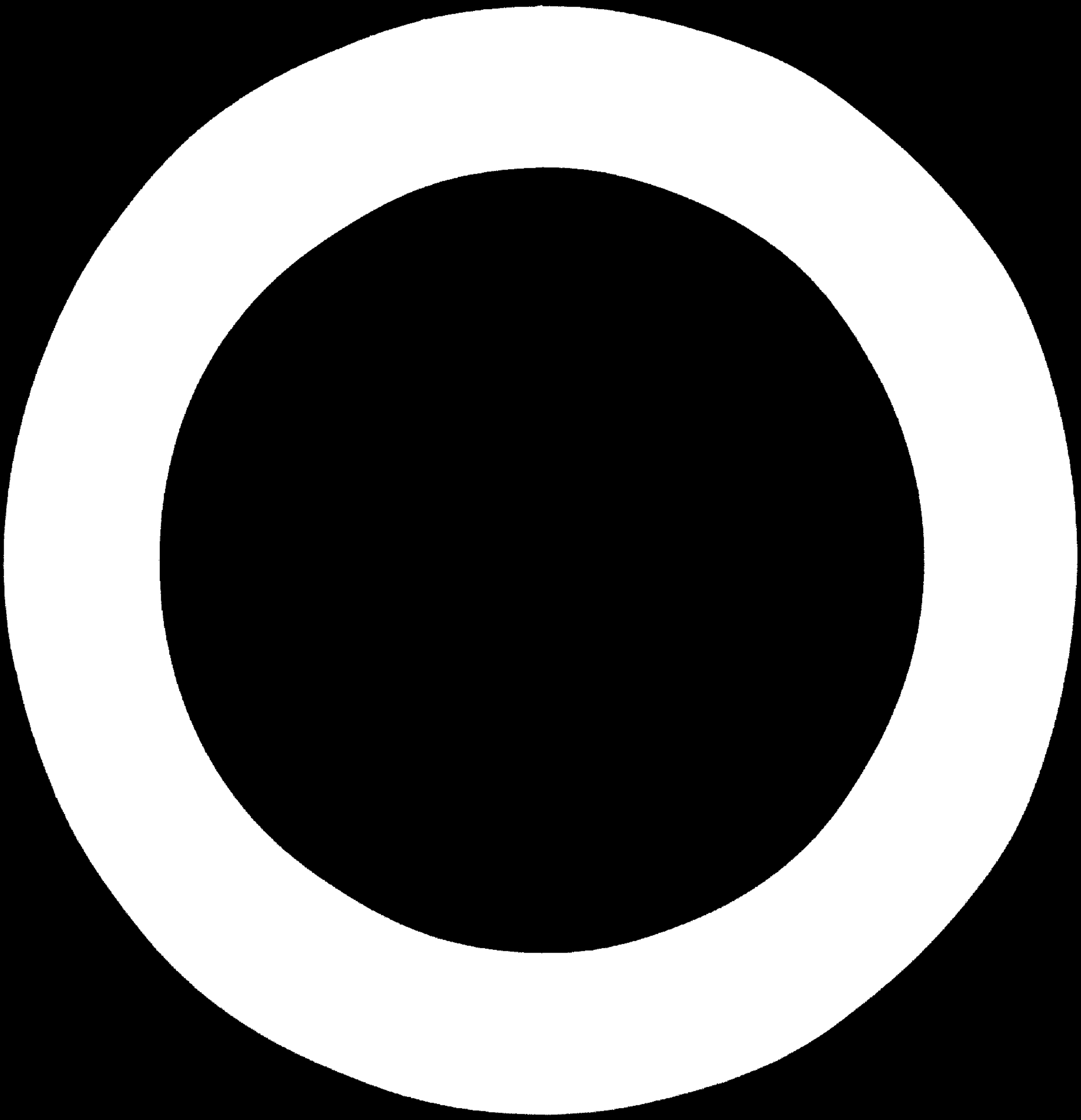
SYRIAN ARAB REPUBLIC

TERMINAL REPORT

Prepared for the Government of the
Syrian Arab Republic by the
United Nations Industrial Development Organization,
executing agency for the
United Nations Development Programme

United Nations Industrial Development Organization





United Nations Development Programme

ASSISTANCE TO THE DEVELOPMENT
OF THE PHOSPHATE INDUSTRY

IS/SYR/71/813

SYRIAN ARAB REPUBLIC

Project Findings and Recommendations

Prepared for the Government of the Syrian Arab Republic
by the United Nations Industrial Development Organization,
executing agency for the United Nations Development Programme

Based on the work of Y.Y. Abdel Azim,
expert in the calcination of phosphate rock

United Nations Industrial Development Organisation
Vienna, 1975

Explanatory notes

Reference to "tons" are to metric tons.

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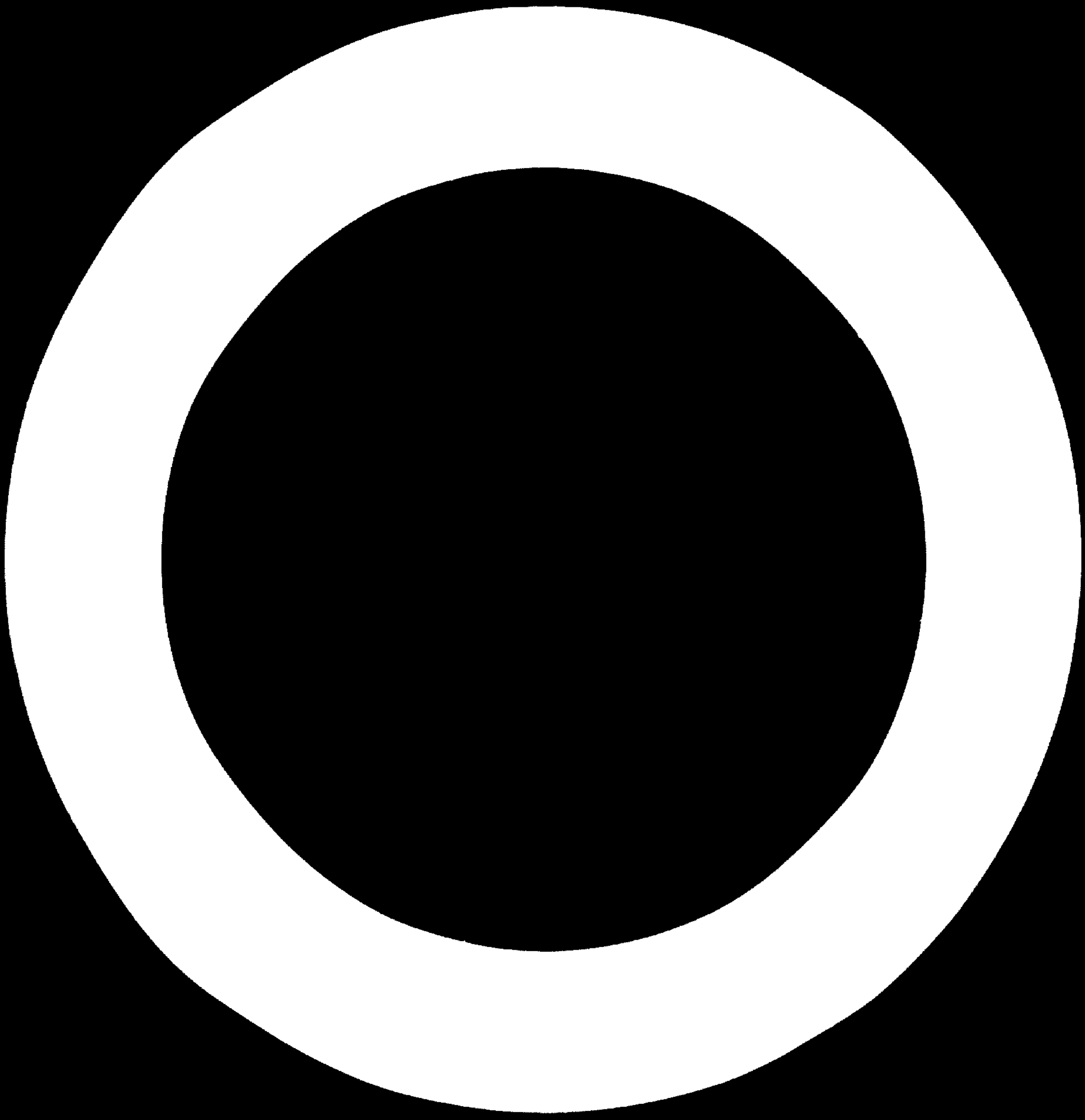


INTRODUCTION

Project background

Phosphate deposits in the Syrian Arab Republic

The main zone of phosphate rock occurence in the Syrian Arab Republic is the Ghadir-el-Hamel area, with five known deposits (see map). Situated in the central part of the country, near the town of Palmyra (Tadmor), it covers an area of approximately 550 km^2, roughly included in a rectangle of 30 x 25 km and extending 45 to 70 km south-west of Palmyra.

The five known phosphate rock deposits are known by the following names:

Khneifiss
Eastern
Lebtar
Central
Hamel

Two of them, the Khneifiss and Eastern deposits, have been the objects of three mining projects (one mining project for Khneifiss and two for Eastern), and are already in production or development. From all available information, the choice of these two deposits for mining development was the best in every aspect. The other phosphate rock deposits may conceal mining possibilities for a more or less distant future.

Khneifiss deposit

The Khneifiss deposit is part of the Ghadir-el-Hamel phosphorite-bearing area in the cental part of the Syrian Arab Republic and is located about 70 km south-west of Palmyra (Tadmor). It is stratigraphically located at the upper part of the Campanian stage, of a limestone series of Upper-Cretaceous age, overlain by Quaternary deposits. It is composed of two commercial beds of grained phosphorites. Phosphate rocks are either hard or soft (friable), with some of interediate hardness. The friable phosphate is predominant and provides the best quality ore. The following figures are estimates of the relative amounts of hard, medium and soft phosphate rocks:

Type	Relative amount (%)	Average P_2O_5 content (%)
Hard	28	23.3
Medium	23	27.7
Soft	49	31.6

The estimated reserves are grouped into "balance" and "non-balance" categories. The latter has a P_2O_5 content lower than 28% or an overburden more than 20-m thick. The balance reserve, with an average thickness of 7.3 m for the two phosphate beds and 28.6% P_2O_5 average content, is in excess of 15.5 million tons. These reserves can be taken by opencast mining methods.

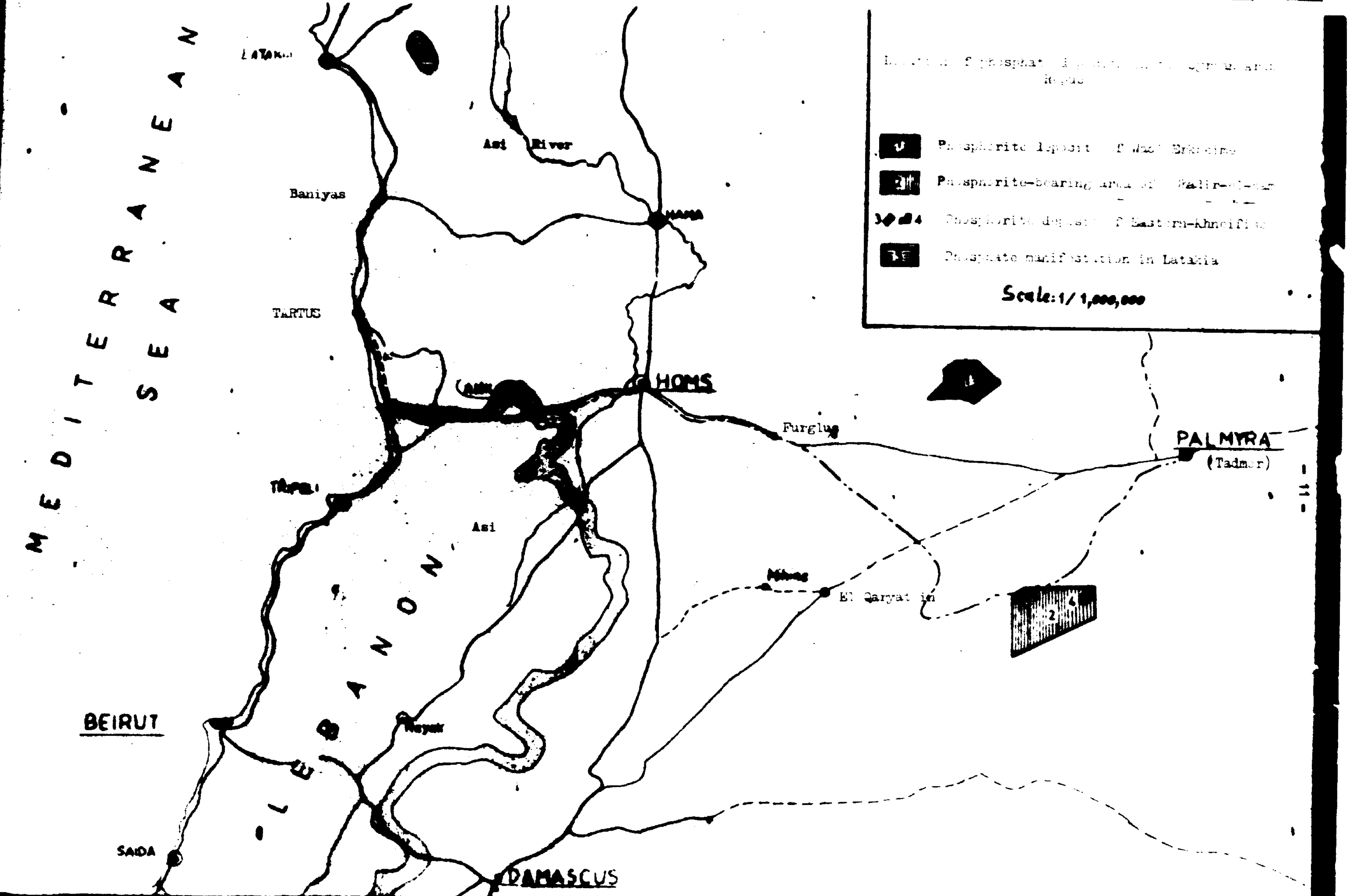
Eastern deposit

The Eastern deposit is part of the Ghadir-el-Hamel phosphorite bearing area, in the central part of Syria, and it is located about 50 km south-southwest of Palmyra. It is stratigraphically located at the upper part of the Campanian stage of a limestone series of Upper-Cretaceous age, overlain by Quaternary deposits. The Eastern deposit was first investigated in 1961/62 by a party of Soviet geologists as part of the work entrusted to Technoexport (Moscow) on the whole area. In the Eastern deposit, as in Khneifiss area, phosphorites can be classified according to their hardness into three types:

Type	Relative amount (%)	Average P_2O_5 content (%)
Hard	Variable	22
Medium-hard	Variable	27
Friable (sand)	Variable	29

In fact, the second and third types are more or less equal since they have the same nature and about the same richness. The only difference is that the third is a phosphorite sand which is completely loose while the second can be loosened by means of excavating tools or machines.

Recently, two zones of the deposit have been the object of two mining projects by Polish and Bulgarian firms. For the first one, called Eastern-A-Biprokop (Design Office of Chemical Raw Material, Poland), estimated


Phosphorite deposit of Wadi Erkheime
Phosphorite-bearing area of Rahir-el-...
Phosphorite deposit of Eastern-Khneifis
Phosphate manifestation in Latakia
Scale: 1/1,000,000
MEDITERRANEAN SEA
LATAKIA
Baniyas
TARTUS
TRIPOLI
BEIRUT
SAIDA
LEBANON
Asi River
HAMA
HOMS
Asi
Rayak
DAMASCUS
Furglus
Mhine
El Qaryatein
PALMYRA
(Tadmor)
1
2
3
4

reserves are 55.0 million tons at an average grade of 24.4% P_2O_5. For the second, called Eastern-B-Niproruda (Research and Design Institute for Mining and Assessing of Ores and Minerals, Bulgaria), they are 18.6 million tons at average grade of 24.7% P_2O_5. These reserves can be taken by opencast mining methods.

Previous studies

There have been six previous reports on this subject: on Syrian phosphate rock (1964, the Khneifiss phosphate deposit (1967), the eastern phosphate deposit (1968), the Eastern-B phosphate deposit (1968) and on both the Khneifiss and the eastern phosphate deposits (1972).[1/] These reports were done in several steps and developed differently over time, which makes it difficult to discover the final solution selected.

More recently, further detailed studies to estimate the reserves of the Eastern-A phosphate deposit have been made by the General Company for Phosphate and Mines (SAR) in connection with the government Geological Research and Mineral Wealth Department, but their final reports are not yet available.

Development of phosphate deposits

As mentioned above, the Khneifiss deposit is in an advanced stage of mining and beneficiation. However, exploitation of the Eastern deposit is still in an embryonic stage. The basis for establishment of mining, beneficiation and calcining methods need to be identified.

Objectives of the project

In view of the situation described above, the Government of the Syrian Arab Republic requested technical assistance in the form of a project with the following objectives:

(a) Determine the most suitable conditions for calcination of phosphate rock from the eastern deposit of the Palmyra Desert;

(b) Determine the suitability of the calcined rock for use in the production of phosphoric acid, superphosphates and nitrophosphate fertilizers;

(c) Assist the Government in identifying possible export markets for the calcined rock.

[1/] See bibliography.

Official arrangements

The Project Document was signed on 8 January 1973, with UNIDO designated as executing agency for the UNDP. The General Company for Phosphate and Mines agreed to provide laboratory facilities and personnel for the necessary tests and chemical analyses.

The Project budget, for the employment of a calcination expert for 12 months, was $US 30,000.

Work on the project did not begin until 17 December 1973, when the expert was briefed at UNIDO and sent to Damascus. (His duty station was changed to Homs on 8 January 1974, which was more convenient to the mines and laboratory facilities.)

Because of the delay experienced in obtaining adequate laboratory facilities (they were not in full action until mid-June 1974), the Project Document was amended on 28 September 1974 by extending the expert's contract 2 months and increasing the budget by $US 5,000.

Other changes in the time schedule and the programme of work were necessary; for example, the study of the suitability of the calcined rock for use in the production of fertilizers and the study of the export market for the calcined rock were cancelled because of the shortage of time.

At the request of the General Company for Phosphate and Mines, a general review of these previous reports was undertaken and prepared as a separate report.

The completion date of the project was 12 February 1975.

I. HEAD-SAMPLE STUDIES

A. Preparation of a technological sample from the Eastern-A phosphate deposit

A technological sample was collected from 10 bore-holes chosen from the network of bore-holes made recently by the Department of Geological Research to estimate the reserves of the Eastern-A phosphate deposit. The distance between the bore-holes was 250 m. The average thickness of the overburden which had to be removed before reaching the phosphate rock deposit was 2.5 m. The samples were collected by a stripping method along the whole thickness of the phosphate beds (hard, medium and soft phosphate rocks). The average depth of the bore-holes was about 10.

The technological sample, weighing about 1 ton was of crushed ore passing a maximum size of 200 mm. Representative hand specimens were selected for mineralogical studies. The technological sample was then subjected to steps of crushing to pass a 19.0-mm sieve and then riffle cut by coning and quartering to obtain representative samples for complete chemical and screen analyses and beneficiation tests. Figure I is a flow chart of the sample treatment procedure.

B. Mineralogical study

The specimens were mainly composed of friable phosphorites. Phosphorite is an impure calcium phosphate rock usually containing calcite, chalcedony, quartz, clay, iron oxides and various impurities, including uranium. The phosphate mineraloid may be replaced to various degrees by calcite or have calcite cores or inclusions. In addition, the phosphate mineraloid has in its structure an additional small fraction of carbonate either as a part of the mineral lattice or as a dispersed component; however, it is not believed to be present in the form of calcite.

In the case of the sample of Eastern-A phosphate rock reserved for mineralogical studies, the phosphorites present were mainly carbonaceous. Siliceous and mixed-matrix phosphorites were also present but minor. The typical rock was a light brownish-grey in colour. The matrix was primarily calcite, the major cementing material, and a little quartz sand.

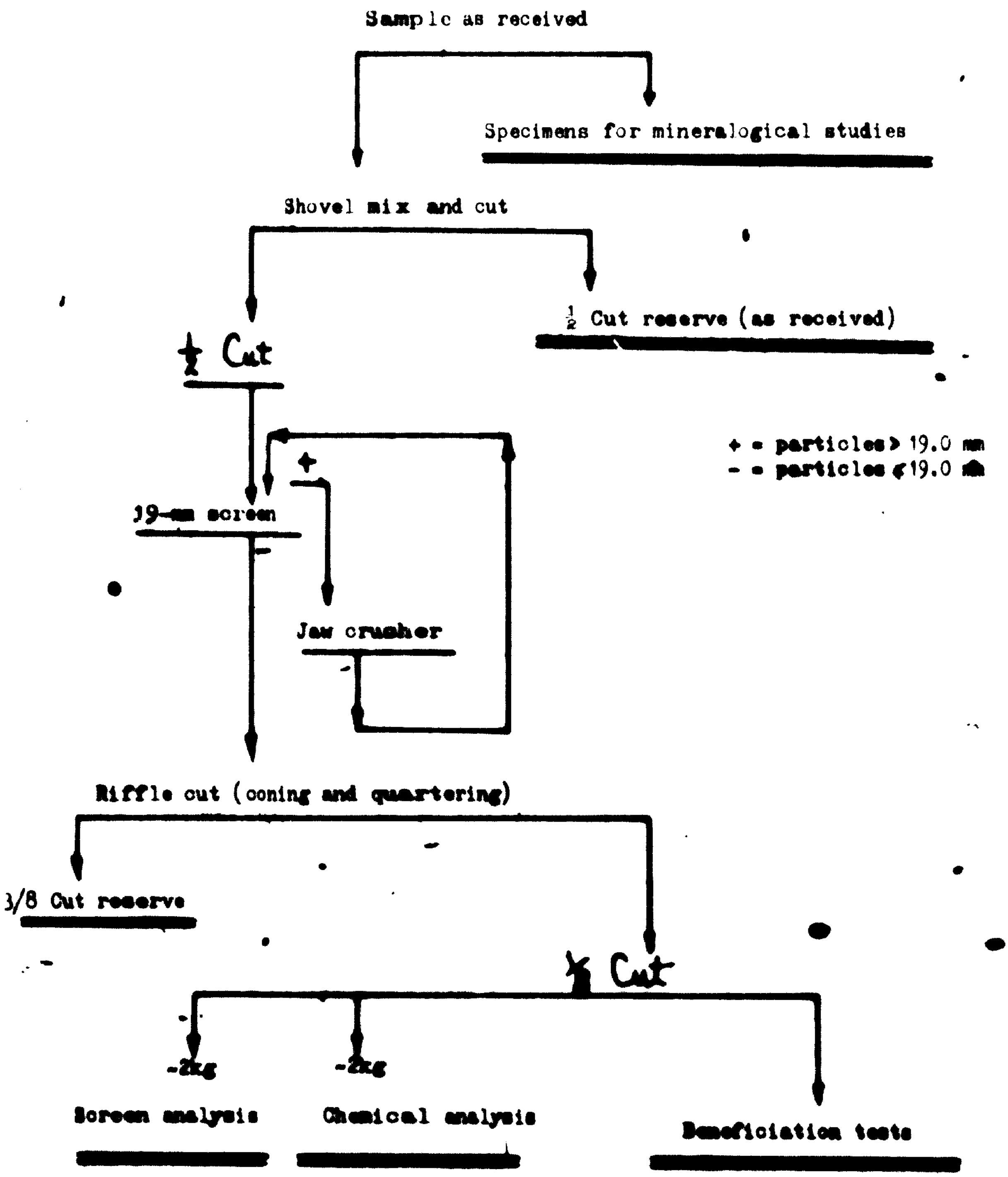


Figure I. Sample treatment procedure

Chalcedony, clay and limonite were rare. The approximate composition of the phosphorite is as follows:

Component	Proportion (volume %)
Phosphate mineraloid	60-65
Calcite	30-35
Siliceous minerals	5-10

C. Chemical analysis

The representative sample, crushed to 19.0 mm and weighing about 2 kg, (see figure I for the sample treatment procedure) was ground and powdered to pass a 200-mesh screen. It was then quartered to 100 g and subjected to chemical analysis, the results of which are shown in table 1.

Table 1. Chemical composition of the technological sample

Constituent	Proportion (wt %)
P_2O_5	24.41
SiO_2	7.76
CaO	43.41
MgO	1.68
Fe_2O_3	0.23
Al_2O_3	0.56
F	2.65
Cl	0.12
CO_2	9.64
SO_3	1.08
Insoluble residue	7.07
Loss on ignition	12.86

The ore is seen to be of low grade, as it only contains 24.41% P_2O_5, which is equivalent to 53.31% bone phosphate of lime (BPL), $Ca_3(PO_4)_2$.
In agreement with the mineralogical investigation, the gangue minerals were found to be mainly carbonates (essentially calcite) and to a lesser extent detrital quartz.

D. Crushability and dry-screen analyses

The components of the ore are markedly different in hardness. The crushing of such ore is rather critical and must be controlled to avoid overproduction of fines. The study of the crushing characteristics of the head sample was given a great deal of care.

Primary crushing

A laboratory jaw crusher was used in a closed circuit for the primary crushing of the technological sample to pass a 19.0-mm sieve (see figure I). A representative sample of the crushed material (about 2 kg) was collected for screen analysis by coning and quartering. A set of 11 sieves ranging from 19.0 mm to 0.075 mm was used. Each size fraction was weighed and analysed chemically for P_2O_5, insoluble residue and loss on ignition. The results are shown in table 2 and represented graphically in figures II, III and IV. From the results shown in the table, it can be said that screening 19-mm crushed ore selectively concentrates phosphorite.

The carbonate content (indicated by the loss on ignition) and the phosphorite frequency (represented by the P_2O_5 content) show an inverse relation to one another: the fractions with highest phosphate content are in the range -0.75 + 0.120 mm, and the fractions with lowest carbonate content are in the same range.

It was possible to classify the size fractions into three grade categories as shown in table 3.

Table 2. Grain-size distribution and chemical analysis of ore after primary crushing (19.0 mm)

Size fraction[a] (mm)	Proportion of total (wt%)	Chemical analysis % (wt%)		
		P_2O_5	Loss on ignition	Insoluble residue
- 19.00 + 9.50	17.48	21.24	16.79	9.12
- 9.50 + 4.75	9.95	16.73	22.31	6.81
- 4.75 + 2.50	2.50	15.89	21.06	11.26
- 2.50 + 1.50	2.27	18.61	19.08	10.66
- 1.50 + 0.75	8.70	24.23	13.44	9.02
- 0.75 + 0.385	12.36	29.05	8.60	8.63
- 0.385 + 0.200	29.70	30.52	7.14	8.11
- 0.200 + 0.120	3.37	29.26	8.90	3.43
- 0.12 + 0.075	10.81	25.31	13.95	3.87
- 0.075	2.86	18.35	21.29	5.49
Total	100.00			
Head sample (calculated)		24.68	13.48	7.54

[a] "+" refers to particles larger than, "-" to particles smaller than or equal to, the size indicated.

Table 3. Classification of size fractions after primary crushing (19.0 mm)

Grade category	Size fraction (mm)	Proportion of total (wt%)	Chemical analysis (wt%)			P_2O_5 Recovery (%)
			P_2O_5	Loss on ignition	Insoluble residue	
First poor fraction	-19.00 + 0.75	40.90	19.14	19.28	8.10	31.5
Rich fraction	-0.75 + 0.075	56.24	29.12	8.88	7.00	66.4
Second poor fraction	-0.075	2.86	18.35	21.29	5.50	2.1
Total		100.00				100.0
Head sample, (calculated)			24.73	13.48	7.54	

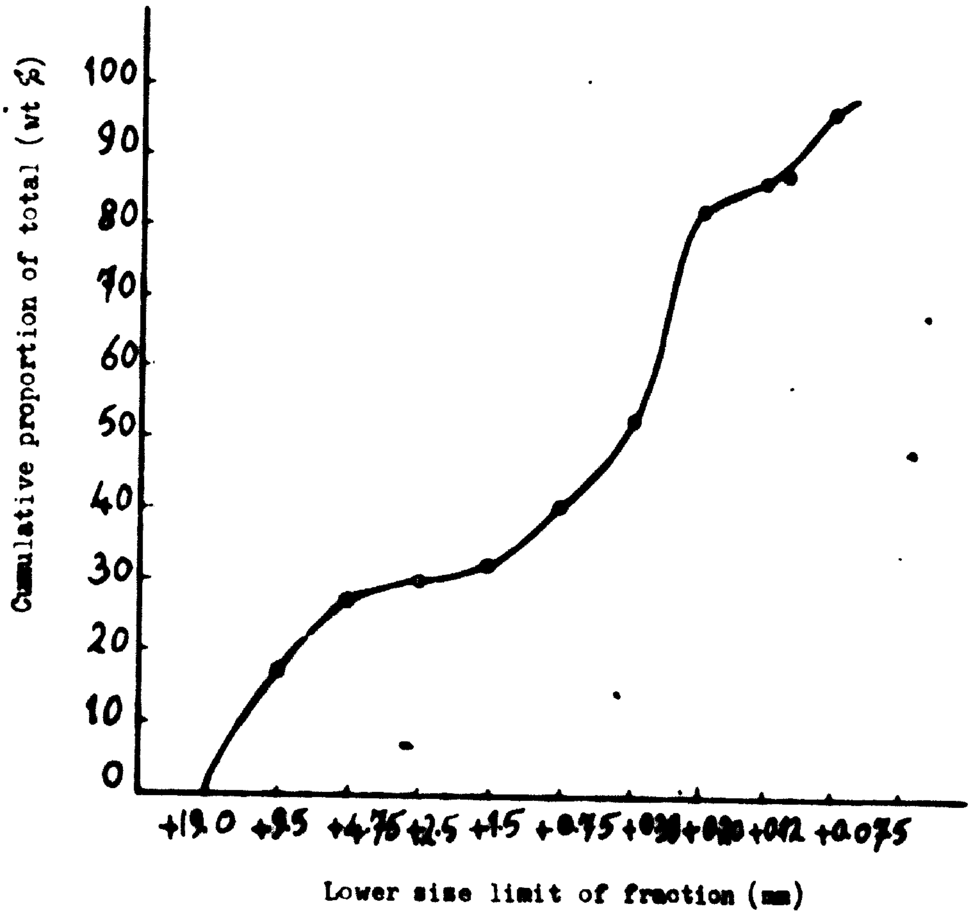


Figure II. Grain-size distribution after primary crushing to a maximum size of 19.0 mm

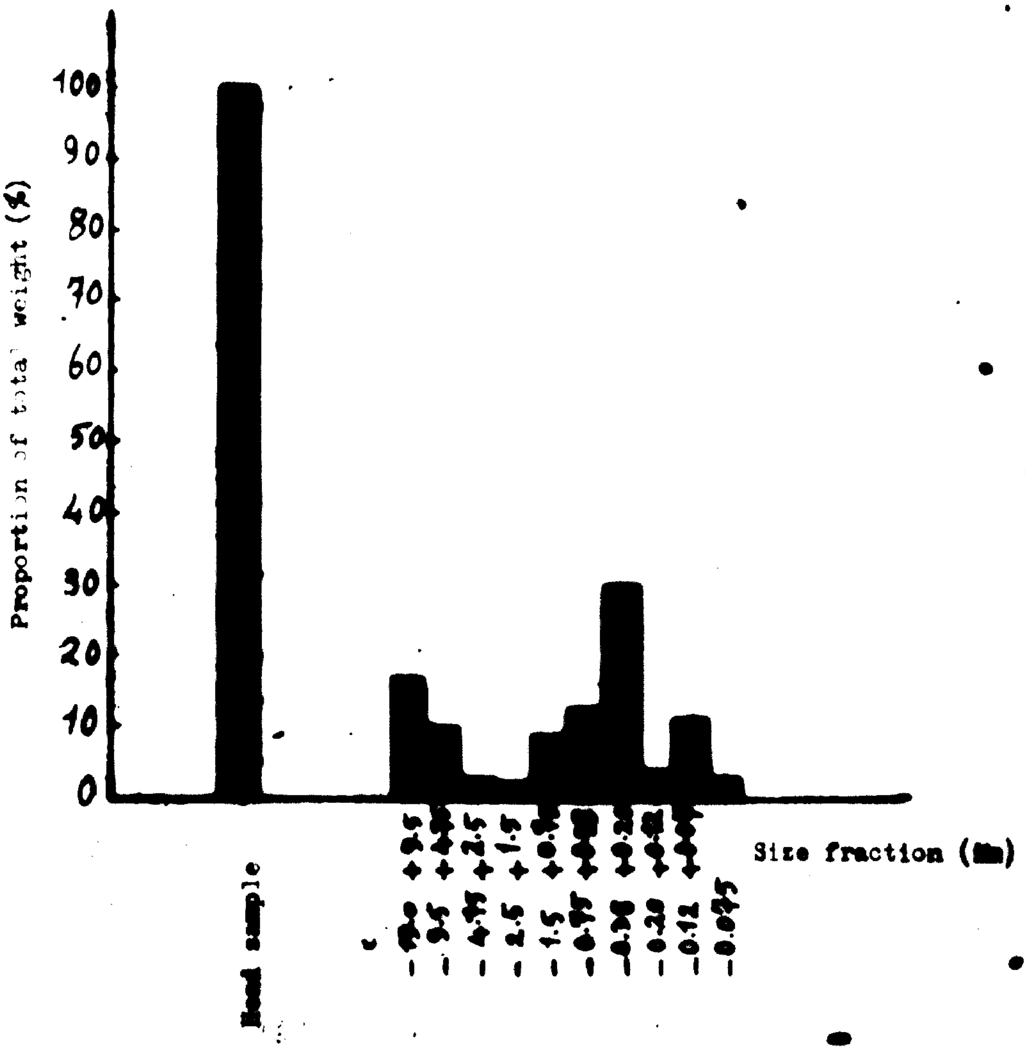


Figure III. Relative weight of the fractions obtained by screening after primary crushing (19.0 mm)

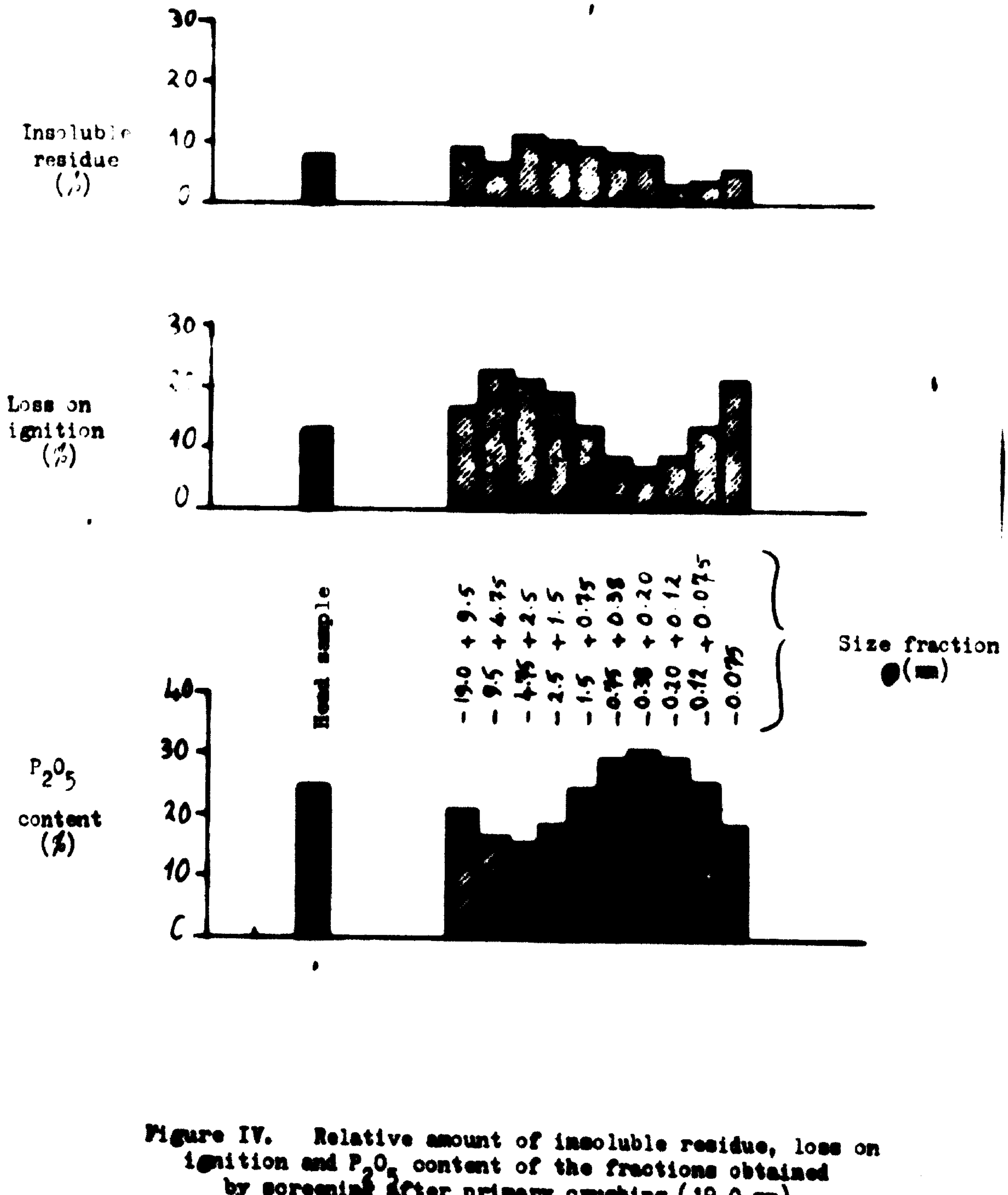


Figure IV. Relative amount of insoluble residue, loss on ignition and P_2O_5 content of the fractions obtained by screening after primary crushing (19.0 mm)

From these results, it is clear that by primary crushing the ore to 19.0 mm and dry screening, a concentrate (the rich fraction) permitting recovery of 66.4% of the P_2O_5 could be obtained.

Secondary crushing

First test

The portion of the primary crushed ore (19.0 mm) was screened with a 9.5-mm sieve, the oversize was subjected to further steps of crushing and screening to pass the 9.5-mm sieve, and the final products were mixed and homogenized. A laboratory jaw crusher was used in a closed circuit. A representative sample was collected for screen analysis by coning and quartering. A set of sieves ranging from 9.5 mm to 0.075 mm was used. Each size fraction was weighed and analysed chemically for P_2O_5, insoluble residue and loss on ignition. The results are tabulated in table 4 and represented graphically in figures V, VI and VII.

Table 4. Grain-size distribution and chemical analysis of ore after secondary crushing (9.5 mm)

Size fraction[a] (mm)	Proportion of total (wt%)	Chemical analysis (wt%)		
		P_2O_5	Loss on ignition	Insoluble residue
- 9.50 + 4.75	18.89	17.70	20.79	7.12
- 4.75 + 2.50	4.70	17.80	20.02	9.56
- 2.50 + 1.50	3.20	18.95	19.11	9.14
- 1.50 + 0.75	9.73	24.33	13.67	3.73
- 0.75 + 0.385	13.38	28.62	8.79	8.88
- 0.385 + 0.200	31.20	30.38	7.41	8.26
- 0.200 + 0.120	3.44	29.64	8.83	6.66
- 0.120 + 0.075	13.94	24.23	15.32	4.05
- 0.075	1.52	17.55	22.05	5.60
Total	100.00			
Head sample (calculated)		25.06	13.04	7.53

[a] See foot-note to table 2.

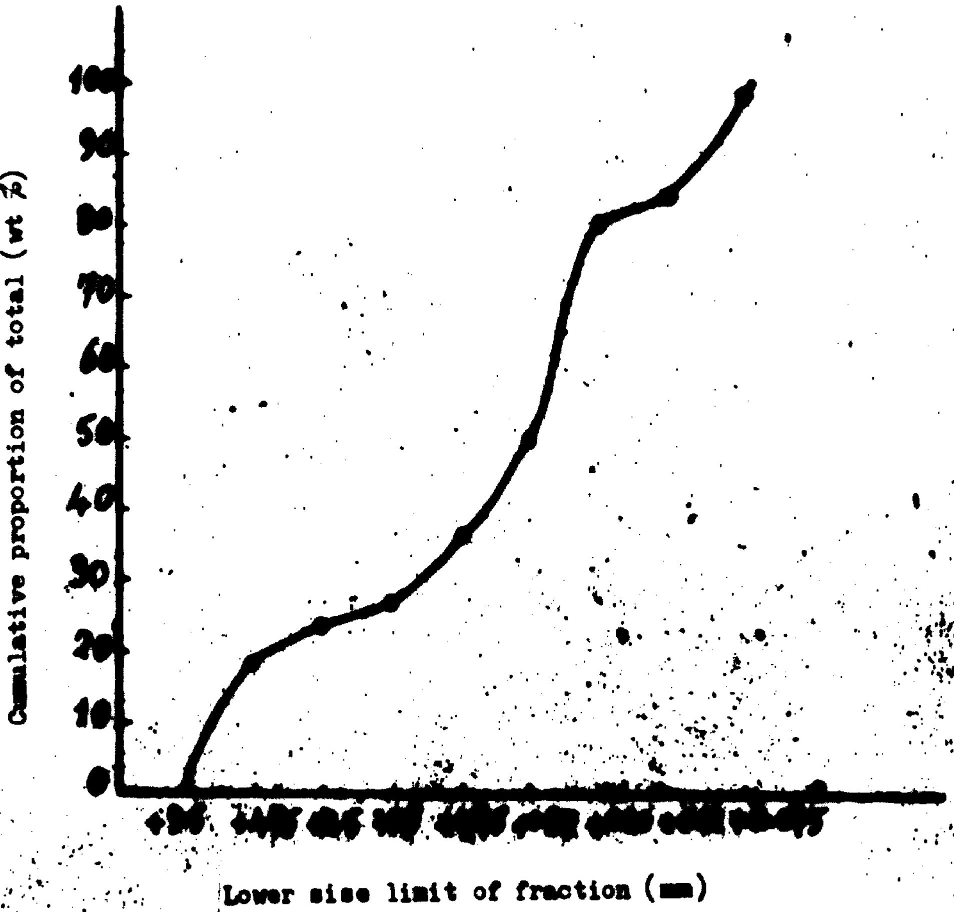


Figure V. Grain-size distribution after secondary crushing to a maximum size of 9.5 mm

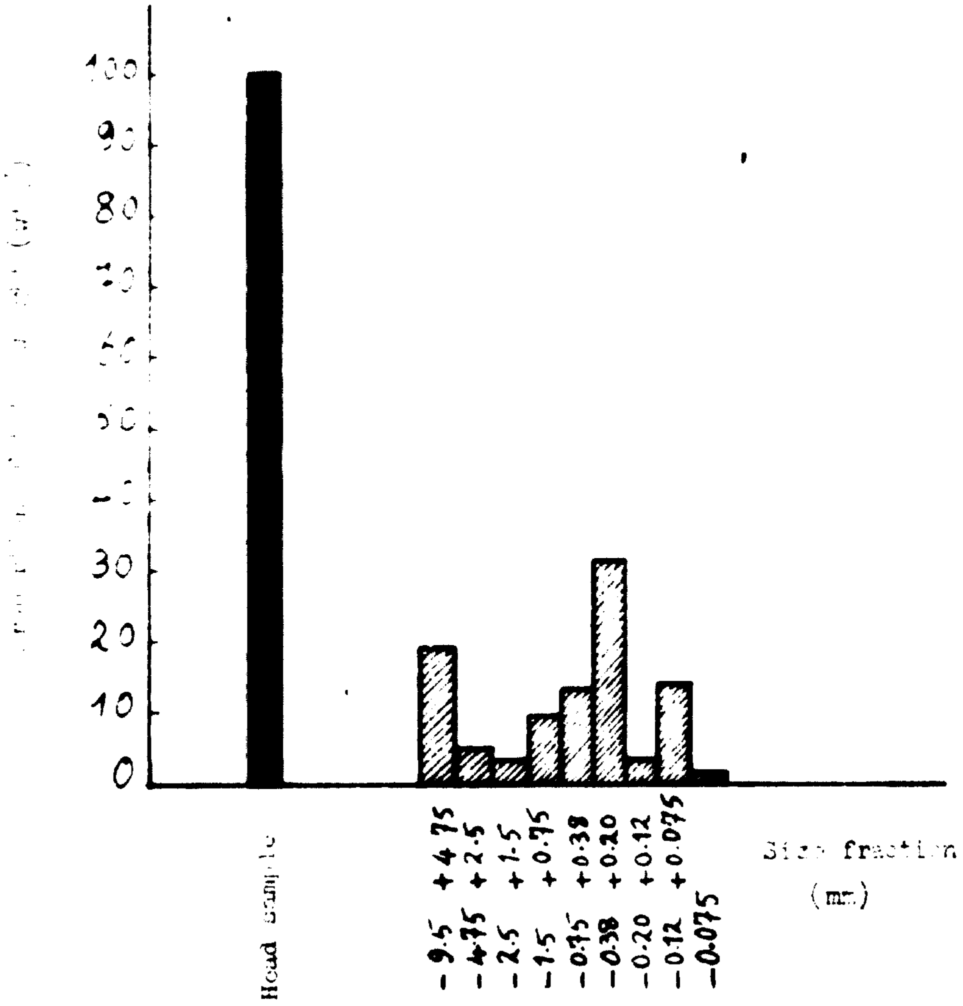


Figure VI. Relative weight of the fractions obtained by screening after secondary crushing (9.5 mm)

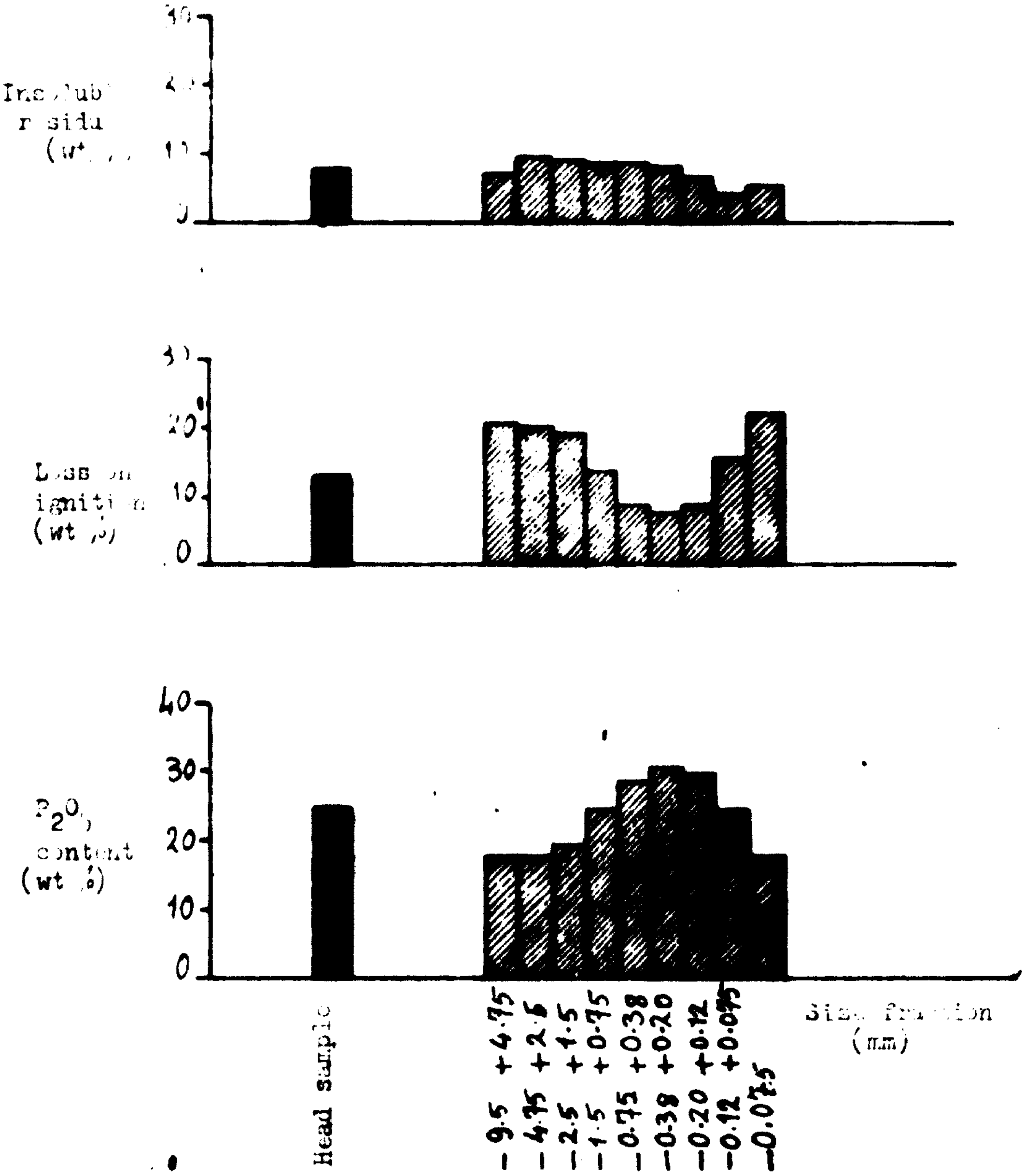


Figure VII. Relative amount of insoluble residue, loss on ignition and P_2O_5 content of the fractions obtained by screening after secondary crushing (9.5 mm)

It can be seen that the highest phosphate content is within the fractions - 0.75 + 0.120 mm, which also shows the lowest carbonate content.

A particle-size classification of the ore thus treated would make it possible to obtain the ore grades shown in table 5.

Table 5. Classification of size fractions after secondary crushing (9.5 mm)

Grade Category	Size fraction (mm)	Proportion of total (wt%)	Chemical analysis (wt %)			P_2O_5 Recovery (%)
			P_2O_5	Loss on ignition	Insoluble residue	
First poor fraction	- 9.50 + 0.75	36.52	19.55	18.62	7.96	28.45
Rich fraction	- 0.75 + 0.120	48.02	29.82	7.87	8.26	57.11
Second poor fraction	- 0.120	15.46	23.47	15.91	4.18	14.44
Total		100.00				100.00
Head sample (calculated)			25.06	13.04	7.52	

These results show that by secondary crushing the ore to 9.5 mm and dry screening, a concentrate (the rich fraction) permitting recovery of 57.11% of the P_2O_5 could be obtained.

Second test

Another portion of the primary crushed ore was screened with a 4.75-mm sieve, the oversize was subjected to further steps of crushing and screening to pass the 4.75 mm sieve, and the final products were mixed and homogenized. Again, a laboratory jaw crusher was used in a closed circuit. A representative sample was collected for screen analysis by coning and quartering. A nest of 9 sieves ranging from 4.75 mm to 0.075 mm was used. Each size fraction was weighed and chemically analysed for P_2O_5, loss of ignition and insoluble residue. The results are tabulated in table 6 and graphically represented in figures VIII, IX and X.

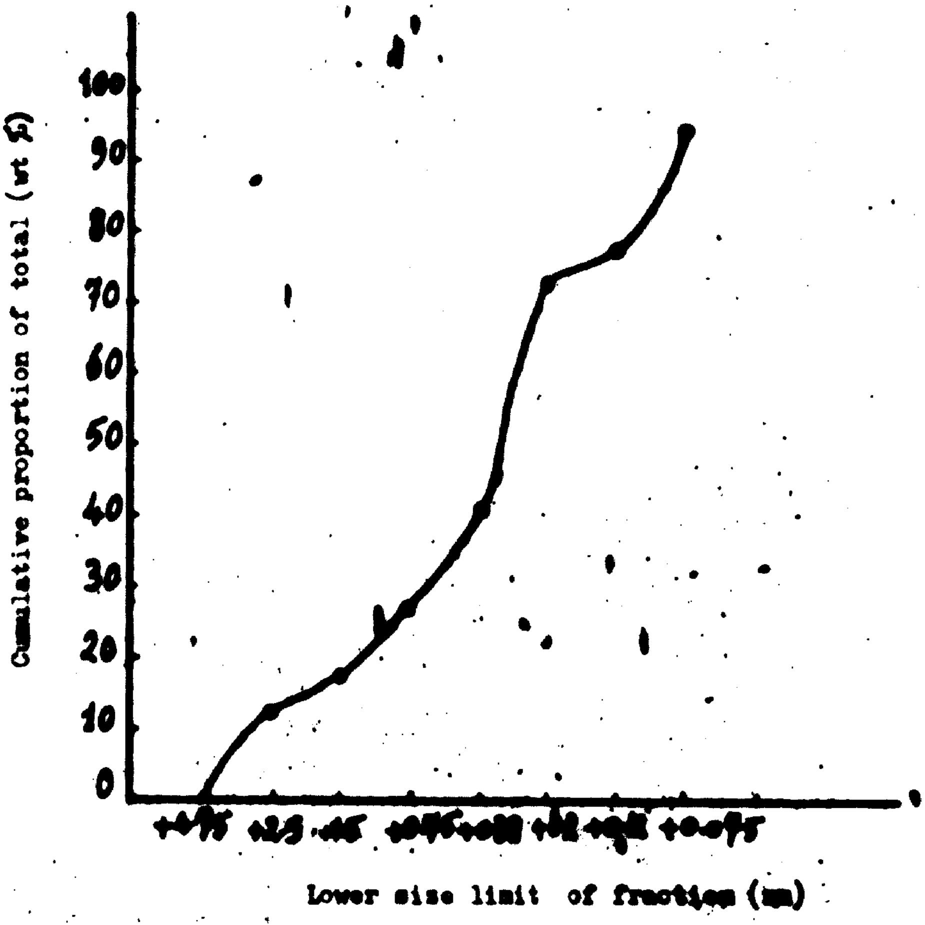


Figure VIII. Grain-size distribution after secondary crushing to a maximum size of 4.75 mm

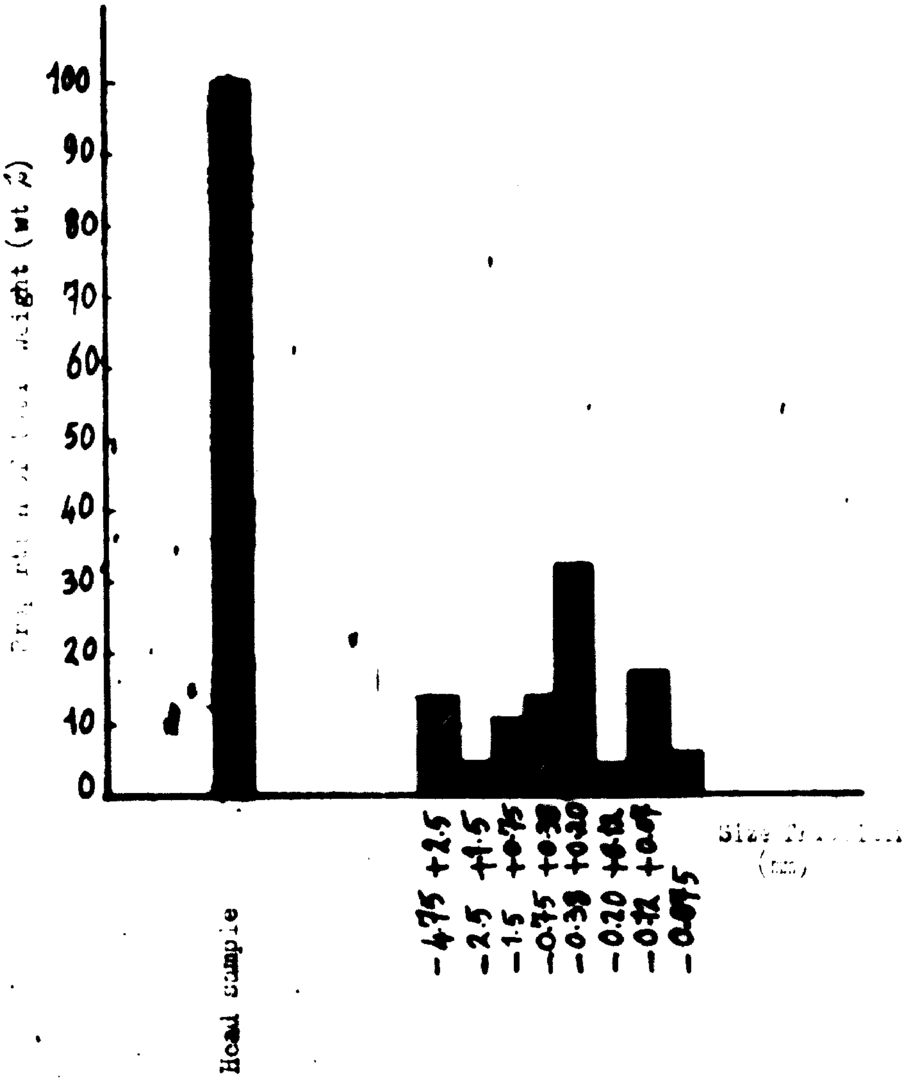


Figure IX. Relative weight of the fractions obtained by screening after secondary crushing (4.75 mm)

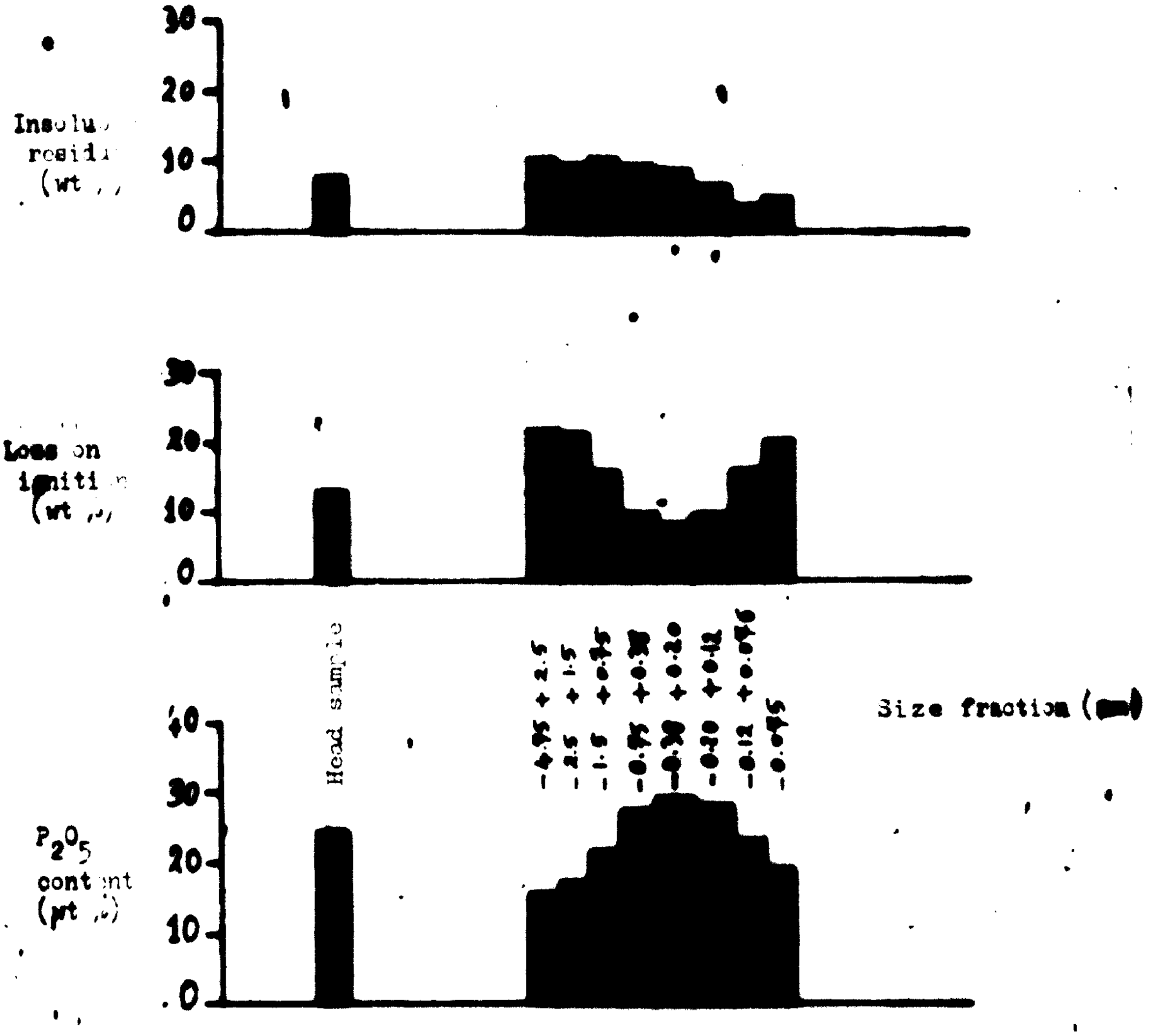


Figure X. Relative amount of insoluble residue, loss on ignition and P_2O_5 content of the fractions obtained by screening after secondary crushing (4.75 mm)

Table 6. Grain-size distribution and chemical analysis of rock after secondary crushing (4.75 mm)

Size fraction^a mm		Proportion of total (wt%)	Chemical analysis (wt%)		
			P_2O_5	Loss on ignition	Insoluble residue
- 4.75	+ 2.50	12.99	15.47	21.76	10.08
- 2.50	+ 1.00	4.29	17.11	20.86	9.24
- 1.00	+ 0.75	10.52	21.22	15.51	9.[illegible]
- 0.75	+ 0.355	13.36	[illegible]	4.56	9.3[illegible]
- 0.355	+ 0.200	32.15	29.00	8.00	8.53
- 0.200	+ 0.120	4.[illegible]	23.14	9.78	5.43
- 0.120	+ 0.075	17.04	23.29	15.00	3.66
- 0.075		5.64	19.31	20.1[illegible]	4.63
Total		100.00			
Head sample (calculated)			24.19	13.45	7.85

a See foot-note to table 2.

The highest phosphate content is within the fraction - 0.75 + 0.120 mm, which also shows the lowest carbonate content.

It was possible to classify the size fraction here also, into three grade categories as shown in table 7.

Table 7. Classification of size fractions after secondary crushing (4.75 mm)

Grade Category	Size fraction (mm)	Proportion of total (wt%)	Chemical analysis (wt%)			P_2O_5 Recovery (%)
			P_2O_5	Loss on ignition	Insoluble residue	
First poor fraction	- 4.75 + 0.75	27.80	18.0	19.3	9.8	20.7
Rich fraction	- 0.75 + 0.12	49.52	28.4	8.6	8.5	58.5
Second poor	- 0.120	22.68	22.2	17.3	4.0	20.8
Total		100.00				100.0
Head sample (calculated)			24.19	13.45	7.85	

From the results, it is clear that by secondary crushing the ore to 4.75 mm and dry screening, a concentrate (the rich fraction) permitting recovery of [illegible] of the P_2O_5 could be obtained.

Conclusions

A comparison of all the preceding results of primary and secondary crushing of the ore shows that the ore used in this work was moderately hard. In the crushing stages to the coarser sizes forced feeding was possible without much harm and with production of a reasonable amount of fines. Closer control of the conditions becomes essential in the stages of secondary crushing. The general behaviour of the distribution of phosphorites, carbonates and silica contents of the ore in primary crushing is more or less the same as in secondary crushing. It can be concluded that satisfactory upgrading of the ore to a reasonable recovery rate is not possible by the simple treatment of crushing and selective screening.

E. Washing, wet screening and desliming

Washing tests

First test series

A series of six tests was made to investigate the effect of washing on the chlorine content of the ore. The tests were conducted by mixing 100-g lots of crushed ore (4.75 mm) with 100 cm^3 of Homs water for periods of 5, 10, 15, 20, 25 and 30 min. The washed products were filtered, dried and chemically analysed. The results are given in table 8.

Table 8. Effect of washing with Homs water on the chemical analysis of secondary crushed ore (4.75 mm)

Test No.	Duration of washing (min)	Weight residue (g)	P_2O_5 (wt%)	Cl (wt%)	Loss on ignition (wt%)	Insoluble residue (wt%)
1	5	98.5	25.00	0.023	13.00	7.67
2	10	98.4	25.53	0.020	12.51	8.07
3	15	98.1	25.23	0.023	12.92	7.87
4	20	97.8	24.84	0.020	13.39	8.04
5	25	97.8	25.56	0.020	12.91	7.71
6	30	97.7	24.84	0.023	12.97	8.19

Second test series

A series of 12 tests was conducted to investigate the effect of washing on the chlorine content of the [illegible] water [illegible] in the [illegible]. The tests were carried out by mixing [illegible] of crushed ore (4.75 mm) with 100 cm^3 of water from the El Bardah, El [illegible]wanah and El Bassirah and from Homs for periods of 5, 15 and 30 min. The washed products were filtered, dried and analysed chemically. The results are shown in table 4.

Table 4. Effect of washing with different waters on the chemical analysis of secondary crushed ore (4.75 min)

Source of water	Test No.	Duration of washing (min)	Weight of residue (g)	P$_2$O$_5$ (wt%)	Cl (wt%)	Loss on ignition (wt%)	Insoluble residue (wt%)
El Bardah	1	5	98.[illegible]	25.42	0.01[illegible]	12.94	7.63
	2	15	97.9	24.37	0.013	12.83	5.9[illegible]
	3	30	97.[illegible]	24.33	0.0[illegible]	13.09	7.30
El [illegible]wanah	4	5	98.5	25.35	0.028	13.11	8.47
	5	15	97.9	25.66	0.026	12.74	8.21
	6	30	97.8	25.48	0.026	12.98	8.54
El Bassirah	7	5	98.5	25.30	0.030	13.20	7.16
	8	15	97.7	25.76	0.030	12.63	7.75
	9	30	97.5	25.84	0.026	13.09	7.79
Homs	10	5	98.5	25.00	0.023	13.00	7.67
	11	15	98.1	25.23	0.023	12.92	7.87
	12	30	97.7	24.84	0.023	12.97	8.19

Conclusions

A comparison of the preceding results shows that washing for 5 minutes is quite sufficient to reduce the chlorine content to values largely lower than the acceptable value in the industries using any of the four kinds of water mentioned above. However, El Bardah water seems to be the most effective water to use for this purpose.

Primary washing followed by wet screening tests

To study the effect of washing and wet screening on the upgrading of the ore, two series of tests were run. In the first series, representative samples of the primary crushed ore (19.0 mm) were used and in the second, representative samples of the secondary crushed ore (4.75 mm).

Series of tests on -19.0-mm crushed ore

A series of five tests was conducted to investigate the effect of washing and wet screening on the upgrading of the ore.

In test No. 1, the crushed ore was washed for 5 min and wet screened on a 4.75-mm sieve. A slight upgrading was obtained: 27.06% P_2O_5 at 81.4% P_2O_5 recovery.

In test No. 2, the crushed ore was washed for 5 min, wet screened on a 4.75-mm sieve and the undersize deslimed to 0.060 mm. A concentrate of grain size -4.75 + 0.060 mm was obtained having 28.32% P_2O_5 at 72.6% P_2O_5 recovery.

In test No. 3, the crushed ore was also washed for 5 min and wet screened on a 4.75-mm sieve, but the undersize was deslimed to 0.120 mm. The concentrate of grain size -4.75 + 0.120 mm thus obtained had 28.11% P_2O_5 at 66.6% P_2O_5 recovery.

In test No. 4, the crushed ore was washed for 5 min and wet screened on a 2.5-mm sieve. The undersize was deslimed to 0.120 mm. A concentrate of grain size -2.5 + 0.120 mm was collected with 29.16% P_2O_5 at 63.7% P_2O_5 recovery.

In test No. 5, the crushed ore was washed for 5 minutes and wet screened on a 2.5-mm sieve. The undersize was deslimed to 0.60 mm. A concentrate of grain size -2.5 + 0.060 mm was obtained having 28.87% P_2O_5 at 70.0% P_2O_5 recovery.

The results are tabulated in table 10. It is seen that by washing, wet screening and desliming the -19.0-mm crushed ore, a moderate improvement in the grade of the ore can be obtained.

Table 10. Effect of washing, wet screening and desliming on the chemical analysis of primary crushed ore (19.0 mm)

Test No.	Size fraction (mm)	Proportion of total sample (wt%)	Chemical analysis (wt%)			P_2O_5 Recovery (%)
			P_2O_5	Loss on ignition	Insoluble residue	
1	- 19.0 + 4.75	25.30	18.31	20.83	5.3	18.6
	- 4.75+	74.70	27.06	11.01	7.58	81.4
	Feed (calculated)	100.00	24.84	13.48	7.02	100.0
2	- 19.0 + 4.75	25.40	18.87	19.37	5.24	19.1
	- 4.75+ 0.060	64.30	28.32	9.87	8.25	72.6
	- 0.060	10.30	20.41	19.51	5.17	8.3
	Feed(calculated)	100.00	24.99	13.25	7.16	100.0
3	- 19.0 + 4.75	25.90	18.73	21.26	5.12	19.5
	- 4.75+ 0.12	58.50	28.11	9.53	8.66	66.6
	- 0.12	15.60	22.19	16.79	5.50	13.9
	Feed (calculated)	100.00	24.75	13.66	7.23	100.0
4	- 19.0 + 2.5	30.30	18.49	20.24	9.45	22.5
	- 2.5 + 0.12	54.20	29.16	8.74	7.75	63.7
	- 0.12	15.50	22.24	16.90	5.81	13.8
	Feed (calculated)	100.00	24.84	13.42	7.96	100.0
5	- 19.0 + 2.5	29.30	18.85	18.77	9.81	22.1
	2.5 + 0.060	60.40	28.87	9.10	8.12	70.0
	- 0.060	10.30	19.16	20.11	5.20	7.9
	Feed(calculated)	100.00	24.92	13.05	8.30	100.0

Series of tests on -4.75-mm crushed ore

A series of six tests was conducted to investigate the effect of washing and wet screening on the upgrading of the -4.75-mm crushed ore.

In test No. 1, the crushed ore was washed for 5 min and wet screened on a 2.5-mm sieve. A very slight upgrading was detected.

In test No. 2, the crushed ore, after washing for 5 min and wet screening on the 2.5-mm sieve, was deslimed to 0.060 mm. A concentrate of grain size

Table 14. Effect of washing, wet screening and desliming on the chemical analysis of secondary crushed ore (4.75 mm)

Test No.	Size fraction (mm)	Proportion of total sample (wt %)	Chemical analysis (wt %)			P_2O_5 recovery (%)
			P_2O_5	Loss on ignition	Insoluble residue	
1	- 4.75 + 2.5	15.8	15.69	22.62	6.92	10.2
	- 2.5	84.2	25.99	12.15	7.73	89.8
	Feed (calculated)	100.0	24.35	13.80	7.59	100.0
2	- 4.75 + 2.5	16.2	15.92	21.81	7.71	10.3
	- 2.5 + 0.060	68.6	28.16	10.14	8.40	77.9
	- 0.060	15.2	19.34	19.50	6.18	11.8
	Feed (calculated)	100.0	24.83	13.44	7.95	100.0
3	- 4.75 + 1.5	20.2	16.30	20.66	11.24	13.3
	- 1.5 + 0.060	64.8	28.37	9.61	[illegible]	74.8
	- 0.060	15.0	[illegible]	20.01	6.02	11.9
	Feed (calculated)	100.0	24.61	13.43	8.56	100.0
4	- 4.75 + 1.5	21.2	17.83	19.70	11.53	15.3
	- 1.5 + 0.12	54.7	28.62	9.03	9.29	63.5
	- 0.12	24.1	21.76	18.01	4.51	21.2
	Feed (calculated)	100.0	24.67	13.44	8.60	100.0
5	- 4.75 + 1	23.7	17.80	20.91	10.29	17.0
	- 1 + 0.12	52.9	29.13	8.96	9.15	62.4
	- 0.12	23.4	21.76	18.20	4.65	20.6
	Feed (calculated)	100.0	24.70	13.93	8.35	100.0
6	- 4.75 + 1	23.0	17.70	21.36	9.68	16.2
	- 1 + 0.060	62.7	29.44	9.30	8.15	73.4
	- 0.060	14.3	18.44	20.06	8.39	10.4
	Feed (calculated)	100.0	25.15	13.60	8.52	100.0

-1.5 + 0.060 mm was obtained having 28.16% P_2O_5 at 71.9% P_2O_5 recovery.

In test No. 3, the crushed ore was washed for 5 min and screened on a 1.5-mm sieve. The undersize was deslimed to 0.060 mm. A concentrate of grain size -1.5 + 0.060 mm was obtained having 28.37% P_2O_5 at 74.8% P_2O_5 recovery.

In test No. 4, the crushed ore, after washing and wet screening on the 1.5-mm sieve, was deslimed to 0.120 mm. A concentrate of grain size -1.5 + 0.12 mm was collected with 23.62% P_2O_5 at 63.5% P_2O_5 recovery.

In test No. 5, the crushed ore was washed for 5 min and wet screened on a 1-mm sieve. The concentrate, of grain size -1 + 0.12 mm, had 19.13% P_2O_5 at 62.4% P_2O_5 recovery.

In test No. 6, the crushed ore, after washing and wet screening on the 1-mm sieve, was deslimed to 0.060 mm. A concentrate of grain size -1 + 0.060 mm was collected having 29.44% P_2O_5 at 73.4% P_2O_5 recovery.

The detailed results of this series are shown in table 11.

Conclusions

From the results of washing, wet screening and desliming of the crushed ore (both 19.0 mm and 4.75 mm), it can be concluded that satisfactory upgrading at reasonable recovery is not possible by this means.

II. ORE-CALCINATION STUDIES

The chemical and mineralogical studies of the Eastern-A technological sample showed that the gangue minerals are primarily calcite (30-35%) with subordinate silica (5-10%) and only minor amounts of clay. The major beneficiation problems will be those of the separation of the phosphate mineral from calcite, silica in various forms and, possibly, clay.

Beneficiation processes that might be applicable to this type of ore include calcination followed by quenching-hydration, wet scrubbing, washing and desliming. The calcite is converted to lime in the calcination. During the scrubbing operation, the lime is slaked ($CaO + H_2O \rightarrow Ca(OH)_2$) and becomes very fine (milky), thus simplifying the subsequent classification steps. The wet process may have the advantage of not requiring a fine grind. In addition, it is conceivable that water in the system can attack the lime in places not readily accessible to the action of dry grinding and thus effect a more complete separation of the lime from the phosphate nodules and permit virtually complete elimination of free lime. The result would be a higher grade phosphate concentrate for acidulation, requiring lower acid consumption in superphosphate production.

Calcination was carried out in crucible furnace using porcelain dishes of 200-cm^3 capacity. The calcined sample was quenched with an equal amount of water immediately after removal from the furnace. The quenched and hydrated ore was scrubbed in a 1-liter capacity glass beaker and washed by decantation till the washings were slime-free. The hydration and washing processes of the calcined phosphate ore are very important for the efficiency of the whole operation of concentration and for the final enrichment of the phosphate ore.

The primary objective of the calcination studies was the determination of the time and temperature conditions that would give maximum loss on ignition. A large number of calcination and washing tests were carried out to determine these and the other optimum parameters with the highest efficiency. A list of all parameters studied follows:

Grain size of the crude ore
Temperature of calcination
Duration of calcination

Duration of quenching-hydration

Source of wash water

Temperature of wash water

Addition of ammonium chloride and hydrochloric acid to wash water

Dry grinding, scrubbing and separation conditions

Effect of grain size

To investigate the effect of grain size, ore crushed to different sizes (-19.0, -9.5, -4.75, -2.5, and -1.20 mm) were calcined under the following conditions:

Temperature of calcination, 900°C

Duration of calcination, 1 h

Duration of quenching-hydration 30 min

Washing with hot water, until washings were slime-free

The results are shown in table 12.

Table 12. Effect of grain size on the chemical analysis of calcined ore

Feed-ore grain size	Product	Proportion of total (wt %)	Chemical analysis (wt %)				P_2O_5 recovery (%)
			P_2O_5	Cl	Loss on ignition	Insoluble residue	
- 19.0 mm	Concentrate	81.30	32.07	0.04	1.97	9.33	95.4
	Tailings	18.70	6.76	-	20.57	4.04	4.6
	Feed (calculated)	100.00	27.33	-	5.44	8.33	100.0
- 9.5 mm	Concentrate	78.67	32.70	0.04	1.87	9.23	93.8
	Tailings	21.33	7.88	-	18.98	3.92	6.2
	Feed (calculated)	100.00	27.40	-	5.51	8.10	100.0
- 4.75 mm	Concentrate	80.07	32.85	0.04	2.01	8.79	94.5
	Tailings	19.93	7.70	-	19.62	3.84	5.5
	Feed (calculated)	100.00	27.83	-	5.52	7.79	100.0
- 2.5 mm	Concentrate	74.5	31.63	0.07	1.88	10.68	90.0
	Tailings	25.5	10.27	-	17.07	3.95	10.0
	Feed (calculated)	100.0	26.18	-	5.75	8.95	100.0
- 1.20 mm	Concentrate	72.7	31.73	0.03	1.62	10.39	89.0
	Tailings	27.3	10.46	-	16.92	4.14	11.0
	Feed (calculated)	100.0	25.91	-	5.79	8.68	100.0

It appears that calcination of ore crushed to 4.75 mm gives the best grade of concentrate at high P_2O_5 recovery.

Effect of temperature of calcination

Theoretically, calcium carbonate ($CaCO_3$) decomposes into CaO and CO_2 at 898°C, but the actual temperature required for calcination may vary somewhat in accordance with the material being calcined and the properties desired in the calcined product.

A series of calcination experiments at 850°, 900°, 950° and 1000°C was performed, using the following conditions:

Grain size of the crude ore, -4.75 mm
Duration of calcination, 1 h
Duration of quenching-hydration 30 min
Washing with hot water, until washings were slime-free

The results are shown in table 13.

Table 13. Effect of temperature of calcination on the chemical analysis of calcined ore

Temperature of calcination (°C)	Product	Proportion of total (wt %)	Chemical analysis (wt %)				P_2O_5 recovery (%)
			P_2O_5	Cl	Loss on ignition	Insoluble residue	
850	Concentrate	86.11	28.26	0.04	6.94	8.64	90.85
	Tailings	13.89	17.68	-	15.19	5.30	9.15
	Feed (calculated)	100.00	26.78	-	8.08	8.17	100.00
900	Concentrate	80.07	32.85	0.04	2.01	8.79	94.50
	Tailings	19.93	7.70	-	19.62	3.84	5.50
	Feed (calculated)	100.00	27.83	-	5.52	7.79	100.00
950	Concentrate	78.82	32.19	0.04	1.64	10.50	92.76
	Tailings	21.18	9.35	-	18.11	3.58	7.24
	Feed (calculated)	100.00	27.35	-	5.12	9.02	100.00
1000	Concentrate	82.91	32.55	0.03	1.03	10.89	95.47
	Tailings	17.09	7.52	-	18.06	3.59	4.53
	Feed (calculated)	100.00	28.26	-	3.93	9.61	100.00

It is clear that a temperature of 900°C may be considered sufficient for the decomposition of the carbonates in this ore.

Effect of duration calcination

The time required for calcination is also a function of temperature and grain size. For the purpose of this investigation, calcination tests of the crude ore crushed to 4.75 mm were carried out at 900°C for periods in the range 30 - 90 min. The results are in table 14.

Table 14. Effect of duration of calcination on the chemical analysis of calcined ore

Calcination time (min)	Products	Proportion of total (wt %)	Chemical analysis (wt %)				P_2O_5 recovery (%)
			P_2O_5	Cl	Loss on ignition	Insoluble residue	
30	Concentrate	87.86	30.23	0.03	4.43	8.43	95.50
	Tailings	12.14	10.28	-	18.76	4.69	4.50
	Feed (calculated)	100.00	27.80	-	6.16	7.97	100.00
45	Concentrate	82.08	32.32	0.04	2.35	8.75	95.12
	Tailings	17.92	7.92	-	19.50	3.86	4.88
	Feed (calculated)	100.00	27.89	-	5.42	7.87	100.00
60	Concentrate	80.07	32.85	0.04	2.01	8.79	94.50
	Tailings	19.93	7.70	-	19.62	3.84	5.50
	Feed (calculated)	100.00	27.83	-	5.52	7.79	100.00
75	Concentrate	77.84	32.81	0.03	1.85	9.11	92.84
	Tailings	22.16	8.90	-	18.25	4.03	7.16
	Feed (calculated)	100.00	27.51	-	5.48	7.98	100.00
90	Concentrate	76.83	32.19	0.03	1.90	10.37	91.02
	Tailings	23.17	10.56	-	17.56	3.67	8.98
	Feed (calculated)	100.00	27.17	-	5.52	8.81	100.00

A gradual decrease in the ignition loss is observed as the calcination time is increased from 30 min to 60 min. After that, the decrease in loss is insignificant. Calcination at 900°C for 60 min gives the best grade of concentrate ([illegible] P_2O_5 at 94.50% P_2O_5 recovery).

Effect of duration of quenching-hydration

The quenching-hydration and washing of the calcined ore are two of the principal stages of the technological flow chart that finally determine the efficiency of the whole process of enrichment by calcination. The calcium oxide formed in the calcination process because of removal of CO_2 must be hydrated. The hydroxide so obtained forms a suspension of slaked lime. Only by elimination of the slaked lime by powerful stirring followed by good washing can one use all the advantages offered by the calcination process. In order to accomplish the hydration, it is necessary to determine the optimum time of contact of the calcined ore with water. A series of tests was made for that purpose, and the results are given in table 15.

Table 15. Effect of duration of quenching-hydration on the chemical analysis of calcined ore

Quenching hydration time (min)	Product	Proportion of total (wt %)	Chemical analysis (wt %)				P_2O_5 recovery (%)
			P_2O_5	Cl	Loss on ignition	Insoluble residue	
10	Concentrate	81.53	31.35	0.04	3.18	8.75	93.11
	Tailing	18.47	10.28	-	16.70	3.59	6.89
	Feed (calculated)	100.00	27.45	-	5.67	7.79	100.00
20	Concentrate	80.52	32.42	0.04	1.87	8.43	94.70
	Tailing	19.48	7.49	-	17.99	3.75	5.30
	Feed (calculated)	100.00	27.56	-	5.00	7.51	100.00
30	Concentrate	80.07	32.85	0.04	2.01	8.79	94.50
	Tailings	19.93	7.70	-	19.62	3.84	5.50
	Feed (calculated)	100.00	27.83	-	5.52	7.79	100.00
40	Concentrate	80.34	33.18	0.03	1.35	9.04	94.40
	Tailings	19.66	8.06	-	17.22	4.18	5.60
	Feed (calculated)	100.00	28.23	-	4.46	8.08	100.00
50	Concentrate	78.57	32.62	0.04	1.64	9.56	93.91
	Tailings	21.43	7.78	-	18.98	4.14	6.09
	Feed (calculated)	100.00	27.28	-	5.34	8.39	100.00
60	Concentrate	76.87	32.88	0.03	1.50	10.47	90.15
	Tailings	23.13	11.96	-	15.22	4.24	9.85
	Feed (calculated)	100.00	28.03	-	4.67	9.02	100.00

The concentrate of the highest quality, 33.18% P_2O_5 at 94.4% P_2O_5 recovery is obtained with a quenching-hydration and wet scrubbing of 40 min; however, 30 min is quite sufficient from the economic point of view.

Effect of source of wash water

In this investigation heated water from four different sources was tested: Homs and the three wells near the phosphate deposits that were mentioned earlier. The other parameters were fixed as follows:

Grain size of crude ore, -1.25 mm
Temperature of calcination, 900°C
Duration of calcination, 1 h
Duration of quenching-hydration, 30 min

The results are in table 16.

Table16. Effect of using heated water from various sources for washing on the chemical analysis of calcined ore

Source of water	Product	Proportion of total (wt %)	Chemical analysis				P_2O_5 recovery (%)
			P_2O_5	Cl	Loss on ignition	Insoluble residue	
El Bardah	Concentrate	80.86	32.77	0.07	2.39	8.76	94.6
	Tailings	19.14	7.96	-	18.95	3.57	5.4
	Feed (calculated)	100.00	28.01	-	5.55	7.76	100.0
El Sowanah	Concentrate	80.81	32.55	0.07	2.44	9.88	94.7
	Tailings	19.19	7.68	-	18.81	3.78	5.3
	Feed (calculated)	100.00	27.77	-	5.57	8.70	100.0
El Bassirah	Concentrate	80.86	32.24	0.08	2.70	8.80	94.8
	Tailings	19.14	7.57	-	18.59	3.70	5.2
	Feed (calculated)	100.00	27.50	-	5.73	7.81	100.0
Homs	Concentrate	80.07	32.85	0.04	2.01	8.79	94.5
	Tailings	19.93	7.70	-	19.62	3.84	5.5
	Feed (calculated)	100.00	27.83	-	5.52	7.79	100.0

It is clear that all four kinds of water are suitable for washing the calcined ore; the differences in the results are insignificant.

Effect of temperature of washing water

In order to determine the influence of wash-water temperature calcination tests were carried out under the same conditions as in the preceding experiment, except the wash water was not heated. The results are tabulated in table 17.

Table 17. Effect of using cold water from various sources for washing on the chemical analysis of calcined ore

Source of water	Product	Proportion of total (wt %)	Chemical analysis				P_2O_5 recovery (%)
			P_2O_5	Cl	Loss on ignition	Insoluble residue	
El Bardah	Concentrate	82.38	32.24	0.07	2.59	9.03	94.41
	Tailings	17.62	8.93	-	13.59	3.97	5.59
		100.00	28.32	-	5.40	8.12	100.00
El Sowanah	Concentrate	85.31	31.42	0.08	3.29	9.38	95.37
	Tailings	14.69	8.85	-	18.89	4.04	4.63
		100.00	28.10	-	5.57	8.59	100.00
El Bassirah	Concentrate	83.94	31.81	0.08	2.56	8.77	95.52
	Tailings	16.06	7.78	-	19.16	3.58	4.48
		100.00	27.95	-	5.21	7.93	100.00
Homs	Concentrate	80.75	32.60	0.06	2.23	9.47	94.91
	Tailings	19.25	7.37	-	19.43	3.53	5.09
		100.00	27.73	-	5.53	8.31	100.00

A comparison of tables 16 and 17 shows that the temperature of washing water has no significant effect on the grade of concentrate obtained, although the use of hot water does slightly increase the P_2O_5 content of the concentrate.

Effect of additions of ammonium chloride and hydrochloric acid to the wash water

The role of ammonium chloride in the washing process is to convert the calcium and magnesium hydroxides to their soluble chlorides so that they can be easily removed. Addition of hydrochloric acid to the washing water increases the solubility and the dispersion of calcium and magnesium hydroxides, i.e., the role of hydrochloric acid would be not only to convert calcium and magnesium hydroxides to their soluble chlorides, but also to increase the solubility and dispersion of calcium and magnesium compounds.

A series of tests was made in which different amounts of ammonium chloride and hydrochloric acid were added to the wash water under the following conditions:

Grain size of crude ore, -4.75 mm
Temperature of calcination, 900 C°
Duration of calcination 1 h
Duration of quenching-hydration 30 min

The results are shown in table 18.

Table 18. Effect of additions of NH_4Cl and HCl to wash water on the chemical analysis of calcined ore

Addition and amount (kg/ton of ore)	Product	Proportion of total (wt %)	Chemical analysis				P_2O_5 recovery (%)
			P_2O_5	Cl	Loss on ignition	Insoluble residue	
NH_4Cl, ten	Concentrate	82.90	32.52	0.07	2.57	8.45	95.19
	Tailings	17.10	7.98	-	19.12	3.78	4.81
		100.00	28.31	-	5.39	7.64	100.00
NH_4Cl, twenty	Concentrate	81.89	32.57	0.10	2.22	9.85	94.74
	Tailings	18.11	8.19	-	19.25	3.62	5.26
		100.00	28.15	-	5.29	8.71	100.00
HCl, two	Concentrate	82.71	31.95	0.07	3.11	9.32	95.00
	Tailings	17.29	8.04	-	18.40	3.98	5.00
		100.00	27.81	-	5.75	8.38	100.00
HCl, five	Concentrate	82.71	32.72	0.09	2.64	8.88	95.04
	Tailings	17.29	8.21	-	18.97	3.75	4.96
		100.00	28.47	-	5.45	7.98	100.00

Effect of dry grinding and separation

Lime produced by calcination under optimum conditions is softer than the other mineral gangue constituents and should be broken up, preferably by a carefully selected grinding or scrubbing operation. Air classification may then be used to remove the fine lime fraction.

Various methods of accomplishing this process were considered. The one adopted (because of its flexibility and compatibility with laboratory-size samples) involved the use of a 2-liter porcelain-ball mill with ball-to-ore weight ratio of 1:1. Two grinding times were tried: 15 min and 30 min. The products were screened on a 0.075-mm sieve.

Calcination was carried out under these conditions:

Grain size of **crude** ore, -4.75 mm
Temperature of **calcination**, 900°C
Duration of calcination, 1 h

The results are shown in table 19.

Table 19. **Effect** of **dry grinding and separation** on **the chemical analysis** of **calcined** ore

Time of Grinding time (min)	Product	Proportion of total (wt %)	Chemical analysis				P_2O_5 recovery (%)
			P_2O_5	Cl	Loss on ignition	Insoluble residue	
15	Concentrate	85.38	29.95	0.05	3.60	8.64	91.45
	Tailings	14.62	16.35	-	9.40	5.15	8.55
		100.00	27.96	-	4.44	8.12	100.00
30	Concentrate	83.24	30.18	0.05	3.20	9.12	90.45
	Tailings	16.76	15.87	-	8.86	5.34	9.55
		100.00	27.77	-	4.14	8.48	100.00

It would appear from the data obtained that dry grinding and screening of the calcined ore fail to give high quality concentrate.

Conclusions

On the basis of all the results of crude-ore calcination tests reported above, the optimum parameters are as follows:

Grain size of crude ore, -4.75 mm
Temperature of calcination, 900°C
Duration of calcination, 1 h
Duration of quenching-hydration, 30 min
Washing with water, until slime-free

The concentrate under these conditions obtained is of reasonably high grade: 32.85% P_2O_5 (equivalent to 71.77% BPL) at 94.50% P_2O_5 recovery.

III. PRECONCENTRATE CALCINATION STUDIES

Attempts were made to obtain higher quality concentrates by using wet or dry screening before calcination. The results of these tests are reported in this chapter.

Effect of primary washing and wet screening followed by calcination and secondary washing

It was noticed from the results of the primary washing and wet screening tests (tables 10 and 11) that a considerable increase in P_2O_5 content was obtained in the deslimed products. The increase varied from 2 to 5 percentage points on different size fraction samples. Such results give the possibility of combining both primary washing and screening the crude ore followed by calcination and secondary washing.

A series of calcination tests with various preconcentrated ores was conducted as follows:

(a) In test No. 1, a representative sample of the -19-mm crushed ore was washed and wet screened on a 4.75-mm sieve and the -4.75-mm fraction was calcined;

(b) In test No. 2, a representative sample of the -19-mm crushed ore was washed and wet screened on a 4.75-mm sieve and the undersize was deslimed to 0.060 mm. The size fraction -4.75 + 0.060 mm was calcined;

(c) In test No. 3, a representative sample of -4.75-mm crushed ore was washed and wet screened on a 2.5-mm sieve. The fraction -2.5 mm was calcined;

In test No. 4, a representative sample of -4.75 mm crushed ore was washed and wet screened on a 2.5-mm sieve and the undersize was deslimed to 0.060 mm. The fraction -2.5 + 0.060 mm was calcined.

The calcination parameters were fixed as follows:

Temperature of calcination, 900°C
Duration of calcination, 1 h
Duration of quenching-hydration, 30 min
Secondary washing, until slime-free

The results are shown in table 20.

[illegible] washing, wet screening and [illegible] wet [illegible] calcination and washing on the chemical analysis of processed ore

Test No.	Product	[illegible] (wt %)	Chemical analysis					Recovery (%)
			[illegible]	CaO	Loss on ignition	Insoluble residue	[illegible]	
1	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	4.05	-	4.[illegible]
		[illegible]	[illegible]	[illegible].84	[illegible]	[illegible]	-	100.0
2	Concentrate	91.71	34.15	49.92	1.74	[illegible]	0.01	98.6
	Tailings	8.29	[illegible].51	62.79	[illegible]	2.29	-	1.4
		100.00	[illegible]	[illegible]	3.43	3.13	-	100.0
3	Concentrate	84.79	33.36	50.12	1.61	8.29	[illegible]	95.4
	Tailings	15.21	[illegible]	60.42	13.06	4.51	-	4.6
		100.00	29.62	51.67	4.09	7.71	-	100.0
4	Concentrate	89.75	34.46	50.[illegible]	1.45	3.52	0.01	99.1
	Tailings	10.25	2.30	65.09	24.05	0.69	-	0.9
		100.00	31.21	52.06	3.76	7.71	-	100.0

Effect of dry screening followed by calcination and washing

A series of calcination tests were conducted at varied preconcentrate ore as follows:

(a) In test No. 1, a representative sample of the -19.0-mm crushed ore was dry screened on a 4.75-mm sieve and the -4.75-mm fraction was calcined;

(b) In test No. 2, a representative sample of the -19.0-mm crushed ore was dry screened on a 4.75-mm sieve and the -4.75-mm fraction was dry screened on a 0.060-mm sieve. The -4.75 + 0.060-mm fraction was calcined;

(c) In test No. 3, a representative sample of the ore crushed to 4.75 mm was dry screened on a 2.5 mm sieve. The -2.5-mm fraction was calcined;

(d) In test No. 4, a representative sample of -4.75-mm crushed ore was dry screened on a 2.5-mm sieve and the -2.5-mm fraction was dry screened on a 0.060-mm sieve. The -2.5 + 0.060-mm fraction was calcined.

The calcination parameters were fixed as in the preceding series of tests. The results are shown in table 21.

Table 21. Effect of dry sizing followed by calcination and washing

Test No.	Product	Proportion of total (wt %)	Chemical analysis					P_2O_5 recovery (%)
			P_2O_5	CaO	Loss on ignition	Insoluble residue	[illegible]	
1	Concentrate	86.36	33.14	49.[illegible]	2.09	8.[illegible]	[illegible]	96.[illegible]
	Tailings	13.64	7.50	[illegible]	27.59	3.[illegible]	-	3.44
		100.00	29.63	[illegible].36	5.56	8.[illegible]	-	100.00
2	Concentrate	90.28	33.31	49.[illegible]	1.98	8.6[illegible]	[illegible]	98.88
	Tailings	9.72	3.52	[illegible]	31.56	3.[illegible]	-	1.12
		100.00	30.41	[illegible]	4.34	8.[illegible]	-	[illegible]
3	Concentrate	83.33	33.14	49.99	1.96	8.[illegible]	[illegible]	95.47
	Tailings	16.67	7.88	[illegible]	24.27	3.[illegible]	-	4.[illegible]
		100.00	28.92	[illegible]	5.67	7.[illegible]	-	100.00
4	Concentrate	88.50	33.19	49.[illegible]	1.87	9.1[illegible]	[illegible]	98.[illegible]
	Tailings	11.50	3.90	5[illegible].40	29.07	3.1[illegible]	-	1.4[illegible]
		100.00	29.81	[illegible]	4.99	8.4[illegible]	-	100.00

Conclusions

On the basis of the data in tables 20 and 21 it can be stated that primary washing and wet screening followed by calcination and secondary washing gives better results than dry screening followed by calcination and washing.
The best results were shown by test No. 2 in table 20 (an increase of the P_2O_5 content of the concentrate to 34.15% equivalent to 74.61% BPL at 98.6% P_2O_5 recovery) and test No. 4 in table 20 (increase of the P_2O_5 content of the concentrate to 34.46% equivalent to 75.29% BPL at 99.10% P_2O_5 recovery). However, these recoveries should be corrected because the actual recoveries of the preconcentrates fed to the calcination tests were 72.6% and 77.9% respectively (see table 10, test No. 2, and table 11, test No. 2). After recalculation, the final recoveries are found to be 71.5% and 77.2%, respectively.

[illegible]. CONCLUSIONS

On the basis of the investigations made on the Eastern-A phosphate ore submitted for study, the following conclusions have been reached:

1. Eastern-A phosphate ore is of low grade, as it contains only 24.41% (equivalent to 53.34% BPL). It is mainly composed of carbonaceous phosphorite, and the gangue minerals are mainly carbonates and, to a lesser extent, silicates.

2. The crushability and dry-screen analyses show that the ore is moderately hard. In the crushing steps to coarser sizes, forced feeding is possible without much harm and with production of a reasonable amount of fines. Closer control of the conditions becomes more essential in the stages of secondary crushing. In the different size fractions, the phosphorite frequencies and the carbonate contents are inversely related. By the simple treatment of crushing and dry screening, satisfactory upgrading at reasonable recovery could not be obtained; the best results were 29.82% P_2O_5 at 57.11% P_2O_5 recovery.

3. The chlorine content of the ore can be reduced to 0.02% Cl, which is substantially lower than the industrially acceptable value, simply by washing for 5 minutes with water from the neighbouring wells of El Bardan, El [illegible]wanah and El Bassirah.

4. Washing, wet screening and desliming fail to give any pronounced enrichment of phosphate content at reasonable recovery, the highest grade obtained having 29.16% P_2O_5 at 63.7% P_2O_5 recovery.

5. Under optimum conditions, direct calcination of the crushed ore to 4.75 mm followed by quenching-hydration and washing produces a reasonably high-grade grade concentrate having 32.85% P_2O_5 (equivalent to 71.77% BPL) at 94.50% P_2O_5 recovery.

6. Calcination of the crushed ore to 4.75 mm followed by dry grinding, scrubbing and separation yielded a concentrate having 30.18% P_2O_5 at 90.45% P_2O_5 recovery, a poorer result than that obtained by the wet process.

7. The combination of primary washing and wet screening of the -19-mm crushed ore, followed by calcination, quenching-hydration and secondary washing resulted in an end-product concentrate having 34.15% P_2O_5 (equivalent to

74.61% BPL) at 71.5% P_2O_5 recovery.

8. The combination of primary washing and wet screening of the -4.75-mm crushed ore, followed by calcination, quenching-hydration and secondary washing, was found to be the best route for upgrading this type of ore to high quality and reasonable recovery. This combined method yielded an end-product concentrate having 34.46% P_2O_5 (equivalent to 75.29% BPL) at 77.2% P_2O_5 recovery. Moreover, other components, such as silica, alumina, magnesia, iron and chlorine, whose contents affect the grade of the phosphate, are present in smaller amounts in this combined method than in the other methods.

9. A comparison between the chemical analyses of the technological head sample and the end-products obtained by direct calcination and by the combined method is presented in table 22.

Table 22. Comparison of the chemical analyses of the head sample and concentrates made by different methods (wt %)

Constituent	Head sample	End-product concentrate		
		Direct calcination of -4.75-mm crushed ore	-19-mm crushed ore	-4.75-mm crushed ore
P_2O_5	24.41	32.85	34.15	34.46
BPL	53.21	71.77	74.61	75.29
Cl	0.12	0.04	0.01	0.01
Al_2O_3	0.56	0.35	0.25	0.25
Fe_2O_3	0.23	0.21	0.13	0.14
MgO	1.68	0.91	0.63	0.54
CaO	48.41	49.85	49.92	50.58
SiO_2	7.76	9.92	9.06	9.04
F	2.65	2.64	2.69	2.55
CO_2	9.64	1.78	1.31	1.22
Insoluble residue	7.87	8.79	8.73	8.52
Loss on ignition	12.86	2.01	1.74	1.45

V. RECOMMENDATIONS

1. On the basis of the conclusions, three flow charts, figures XI, XII and XIII, are recommended for the concentration of the Eastern-A phosphate ore, depending upon the required grade of concentrate.

2. Any one of the three flow charts could be applied also to the sterile phosphate of Khneifiss plant which has more or less the same chemical analysis as the head sample of the Eastern-A phosphate ore. The annual production of the sterile phosphate from the Khneifiss plant is about 100,000 tons of ore with an average P_2O_5 content of 23.5%. At present, there are about 150,000 tons of this sterile ore stored near the Khneifiss plant.

3. The amount of water required for washing and wet beneficiation processes could be provided by the water from the already known wells situated in the area surrounding the phosphate deposits. However, underground water surveys should be continued, as it is important to have large water resources if much wet beneficiation of phosphate ore is to be done.

4. It would be advisable to carry out pilot-scale tests of the recommended flow charts to see if any problems might arise in applying them on an industrial scale and also to estimate the cost of the ore-dressing plant and other economic factors of the project more precisely.

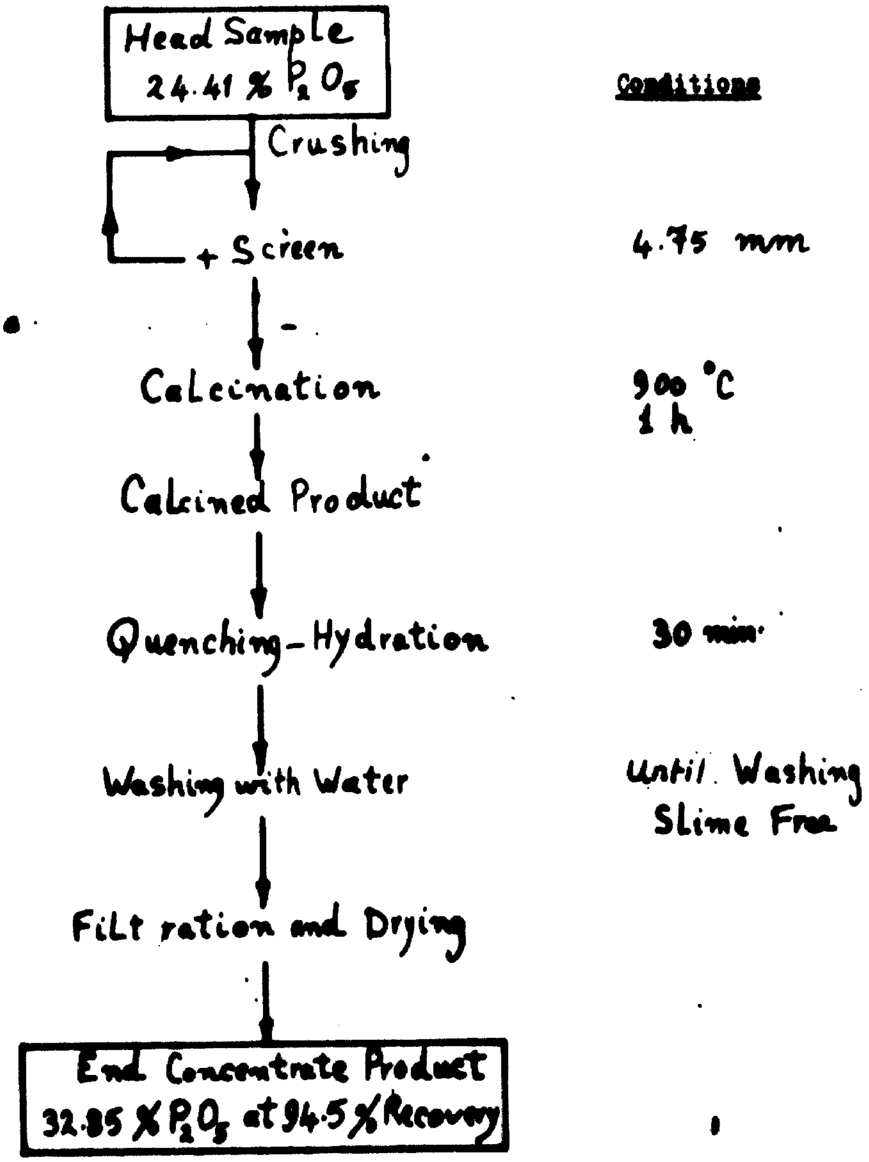


Figure XI. First recommended flow chart. Direct calcination

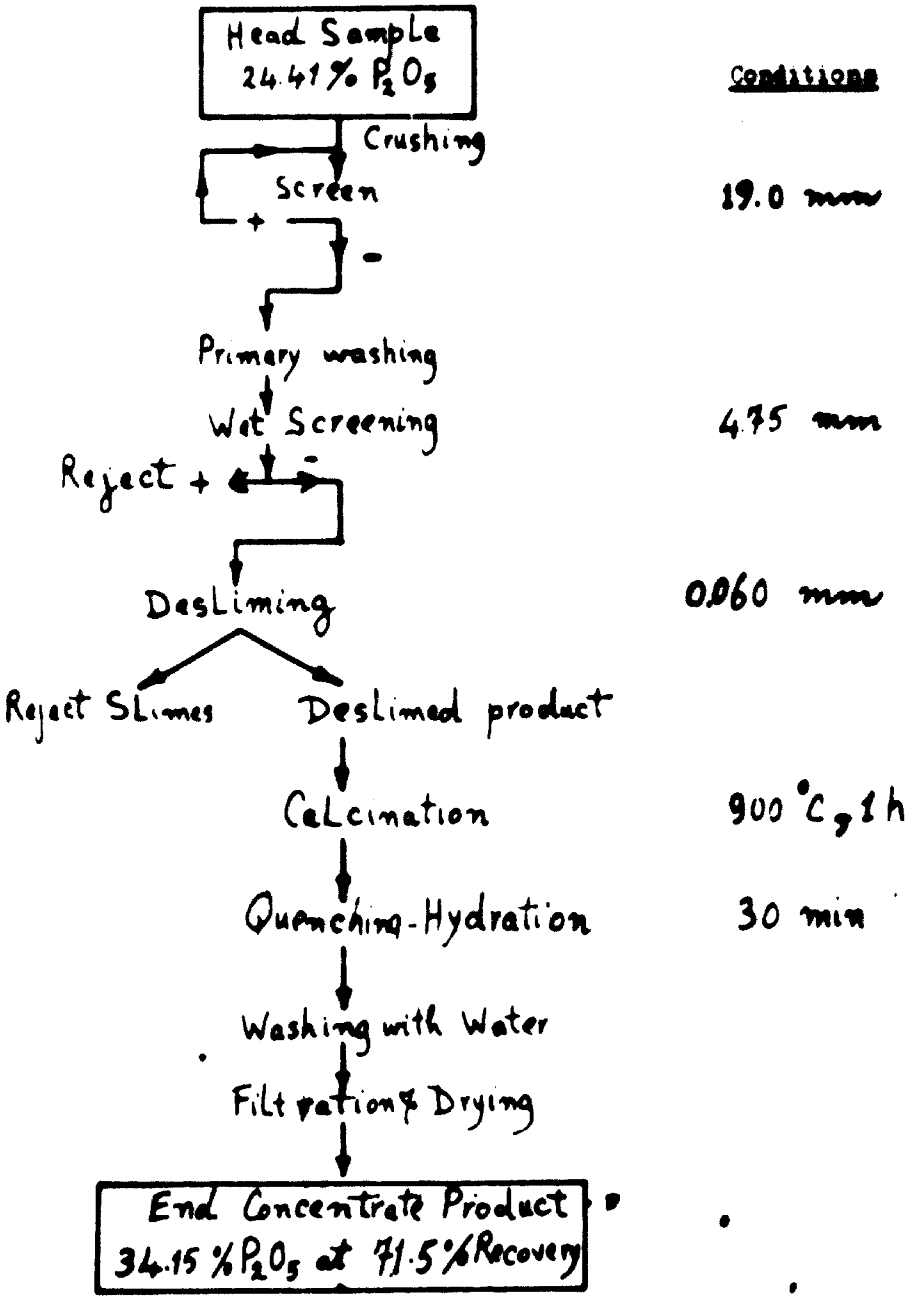


Figure XII. Second recommended flow chart. Combined method

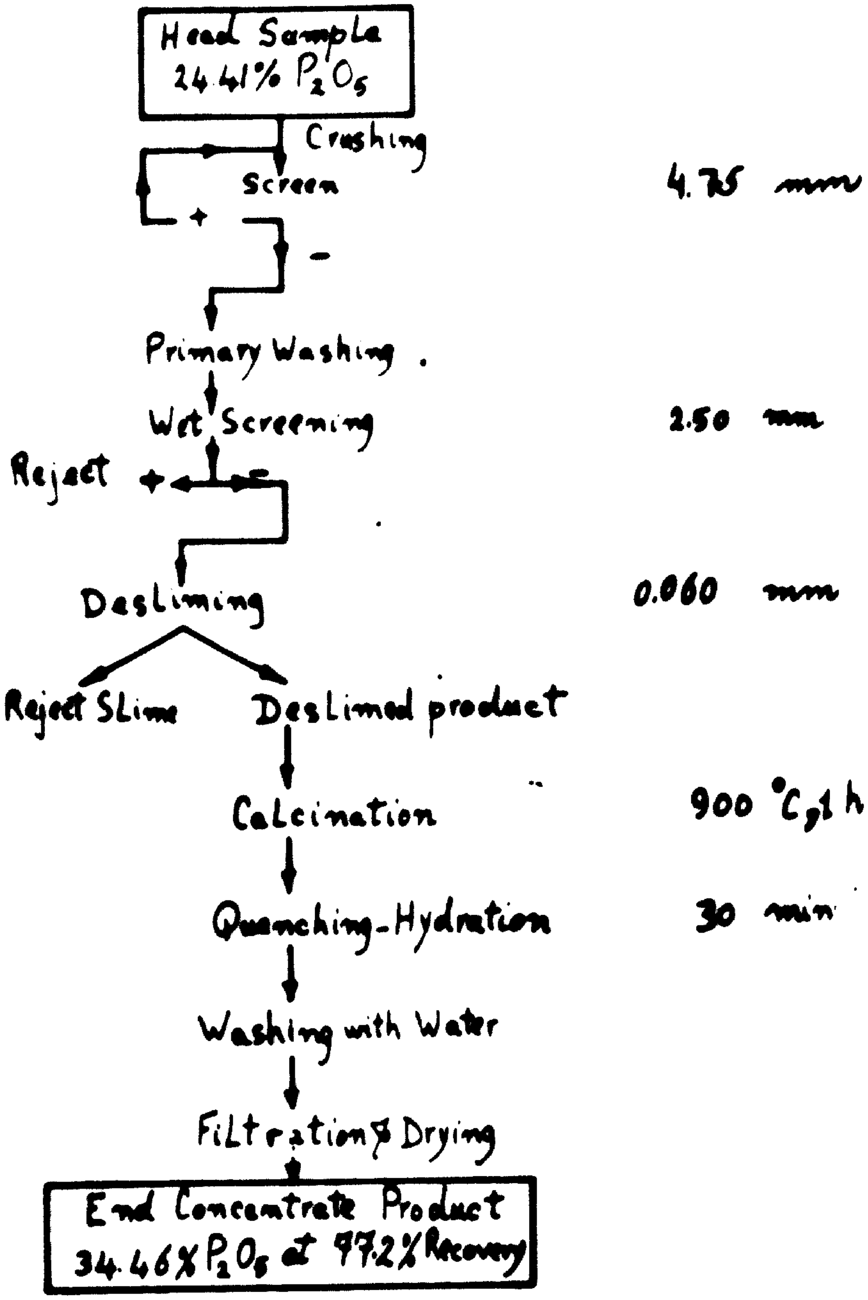


Figure XIII. Third recommended flow chart. Combined method

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Annex

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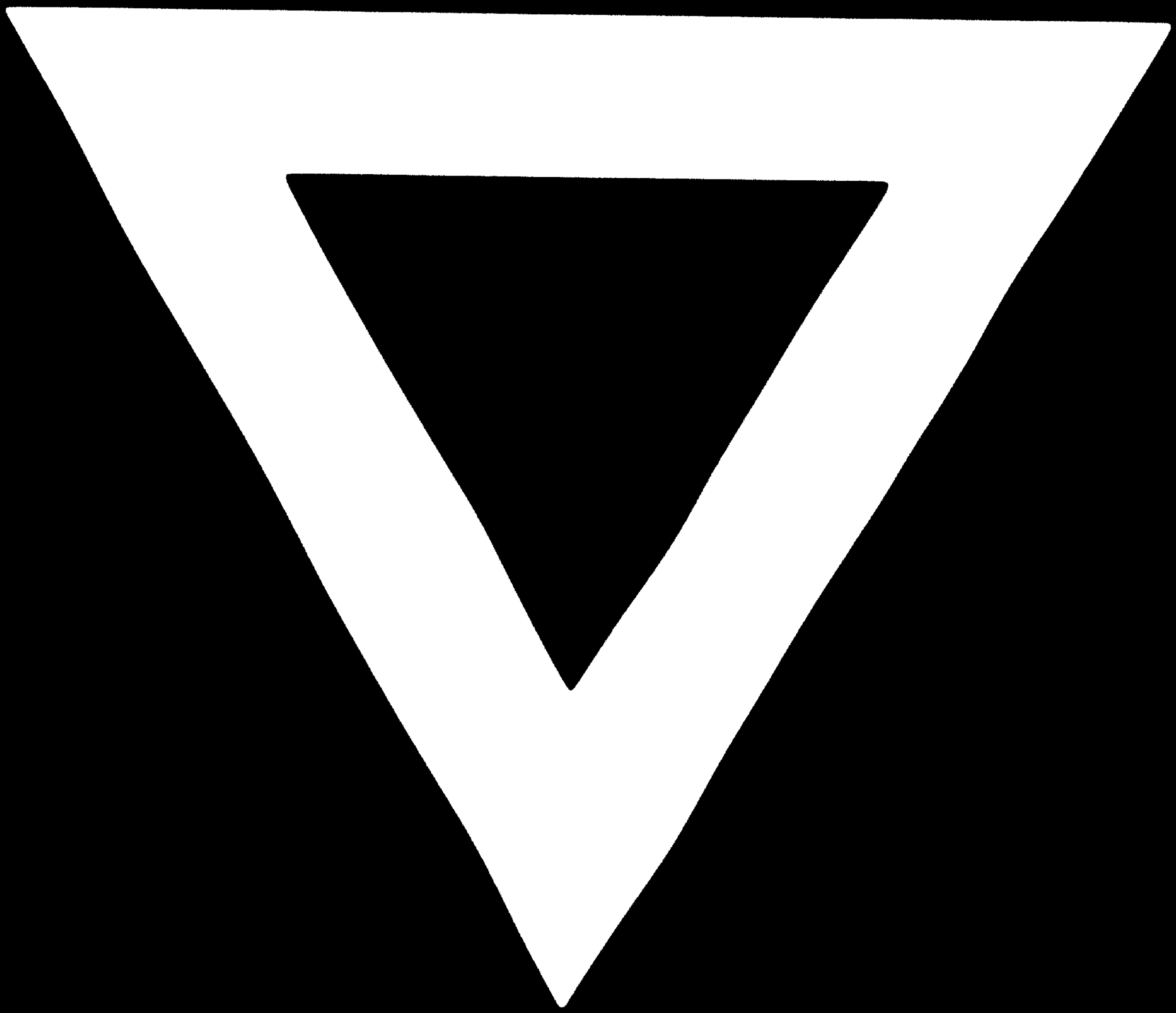

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