Essential Readings in Light Metals: Alumina and Bauxite.
Edited by Don Donaldson and Benny E. Raahauge.

EFFICIENCY OF NEPHELINE ORE PROCESSING FOR ALUMINA PRODUCTION

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Abstract

The comparative economical analysis and energetic analysis of alumina production from various kinds of raw materials were carried out basing on industrial data. The main process parameters of nepheline raw materials processing through sintering adopted at large industrial scale are given. The said technology allows the wasteless utilization of nepheline to produce alumina, soda, potash, potassium sulphate and chloride, portland cement and gallium without polluting the environment. According to industrial data the production cost of alumina while using the sintering of nepheline raw material is considerably lower than in processing of high grade bauxites by the Bayer way due to complete utilization of wastes, and as for capital investments into the process facilities they are lower than those into alumina production from bauxites, production of soda, potash and cement by traditional methods taken together. Are cited the flowsheets of alumina, soda, potash and portland cement production from nepheline ore, the process interrelationships determining the efficiency of raw material processing, and ways of further improvement of the process.

Introduction

In the world-wide practice alumina is produced exclusively from the low-silica gibbsite bauxites processed by the Bayer way.

Given the absence of high quality bauxite reserves in the USSR the industrial processing of low grade bauxites by a combined Bayer - sintering way and by sintering, nepheline processing by sintering is practiced for a long time, the alumina production from alunites is carried out through calcination, reduction with sulphur, digestion and consecutive treatment of pregnant liquors as in the Bayer method.

The use of some above-mentioned methods for alumina production from inconventional raw materials in large industrial scale and the use at some plants of high grade imported bauxites by Bayer way allows to compare the actual process and economical performances which is of a certain interest for aluminium industry people.

The process characteristics acquired in industrial practice in the USSR while processing various raw materials in 1988-1989 are given in Table 1. For the convenince of comparison a total consumption of heat per ton of alumina considering its consumption for the generation of the steam and power to be consumed was calculated. But the given values of total consumption of power per ton of alumina are not sufficient for unbiased comparison because in bauxite processing the alkali are consumed while in nepheline processing for alumina production some valuable by-products are obtained which are then used for production of cement, soda, potash, potassium sulphate and chloride and gallium.

Particularly, in using Kola nepheline concentrate with evaporation of carbonate

Table 1. Technical performances of various raw materials processing to obtain alumina (Soviet data)

Raw material composition	Technology	Al_2O_3 content in raw material (%)	Al_2O_3 yield from the raw material (%)	Consumption ratios in alumina production per ton of alumina							Energy consumption with the same amount of end products kg of standard fuel
				Raw material consumption, t	Consumption of alkali expressed as Na_2O, kg	Fuel consumption in kg of standard fuel *	Steam consumption, Gcal	Power consumption, kWh	Total energy consumption, kg, of standard fuel **		
Imported gibbsite low-silica bauxites	Bayer	46.5	84.95	2.85	31.1	104.2	1.99	316.7	544		2558
Imported low-silica diaspore-boemite bauxites	Bayer-sintering (in parallel)	46.4-55	81.25	2.80	62.1	240.5	4.24	311.6	1062		3111
High-silica gibbsite bauxites	Bayer-sintering (in series)	43.7	87.99	2.92	74.5	635	3.13	466.5	1316		3379
High-silica diaspore-boemite bauxites	Sintering	45.2	79.2	3.24	123.7	1213	2.5	913	1930		4048
Nephelines	Sintering	28.6	84.2	4.06	-	1330	1.2	1000	1854		3574
Alunites	Reducing calcination hydrochemical method similar to Bayer	19.5	76.8	7.49	45.7	467.7	5.96	1029	1810		

* 1 kg of standard fuel has heat content 7000 kcal/kg

** 1 kWh = 0,32 kg of standard fuel; 1 Gcal = 170 kg of standard fuel; 1000 kg of standard fuel = 29.31 GJ

liquors originating from alumina production aside from alumina 0,749 t of soda is obtained per ton of alumina (additional power consumption is 298 kg of standard fuel per ton of soda or 223 kg of standard fuel per ton of alumina), 0,295 t of potash (additional power consumption is 747 kg of standrad fuel per ton of potash or 220,5 kg of standard fuel per ton of alumina), by sintering belite mud with limestone and additives in a wet way 10,216 tons of cement is obtained (additional power consumption is 149,7 kg of standard fuel per ton of cement or 1527 kg of standard fuel per ton of alumina). Thus, the total power consumption in wasteless processing of nepheline rocks is 3834 kg of standard fuel per ton of alumina in a wet way of cement production, while in a dry way of cement production using belite mud the power consumption will be 3574 kg of standard fuel per ton of alumina (power consumption is 125 kg of standard fuel per ton of cement).

For inbiased comparison one has to consider power consumptions in wasteless nepheline processing (production of alumina, soda, potash, cement) and in bauxite processing to produce alumina (considering the power consumption for soda and caustic production - 1118 kg of standard fuel per ton of Na_2O) as well as organization of production of "heavy soda" by traditional method (Salve method), of potash production by electrolysis and carbonization of potassium chloride, of portland cement production from the limestone and clay by the dry way (Table II).

Results of calculation of the total power consumption in production of the same quantity of the products (alumina, soda, potash, and cement - the same as in processing of nepheline ore to obtain one ton of alumina) by various methods given in the right column of the Table I, show that in nepheline processing total power consumption is higher by 15-40% than in processing of high-grade bauxites and production of soda, potash and cement by traditional ways.

It is necessary also to compare the economical performances of the raw material processing in comparable variants. As is evident from the analysis of data given in the Table III the nepheline ore case is characterized by the maximum profit-to-specific investment ratio with the same amount of products (right column of the Table III). Thus the wasteless processing of the nepheline ores in the USSR is characterized by the minimum pay-back period, 2-3 times less than in processing of bauxites and separate production of soda, potash and cement by traditional ways, though nepheline processing is connected with considerable capital investments and increased total power consumption. It is explained by the low cost of nepheline raw material, high cost of the resulting products and relatively low specific capital investments in wasteless processing which are 1,5 times lower than in processing of the high grade bauxites and separate production of soda, potash and cement by traditional ways.

It has to be also noted that processing of nepheline ores is not accompanied by pollution of the environment by solid or liquid wastes, that this production employs the closed water circulation system. This is a considerable advantage compared with bauxite processing which is accompanied with the yield of considerable quantities of the red mud which is utilized nowhere and is being accumulated at mud fields or pumped away to sea.

In spite of the fact that the USSR uses only about 15% of the nepheline concentrate produced by the treatment of apatite-nepheline ore from the Kola peninsula and that its wasteless processing is highly efficient the further development of alumina production in the European part of the USSR using this raw material is handicapped by the necessity of considerable capital investments (regardless of large amounts of imported alumina and soda) and by the difficulty in solving organizational problems due to utilization of raw material and yield of products relevant to the competence of various state bodies.

The further expansion of nepheline processing was also hemmed in the 70-s by the difficulties in exploration of Kiya-Shaltyr nepheline deposit due to new technical concepts at large industrial scale.

Presently the financing of the expansion of Kola nepheline concentrate processing can be realised with the help of foreign capital for establishment of joint ventures for production of alumina, soda, potash and cement. This needs special economical assessment in collaboration with involved aluminium and cement companies.

To solve the problems of advisability of utilization of the nepheline ore in various countries one has to examine the influence of its composition on the economical performances of the wasteless processing.

Description of the nepheline ore processing technology. In the USSR the nepheline raw material from two deposits is used: Kola nepheline concentrate and Kiya-Shaltyr (Central Siberia) nepheline ore with the following composition:

Deposit	Components content, %							
	LOI	Al_2O_3	Fe_2O_3	SiO_2	CaO	Na_2O	K_2O	SO_3
Kola nepheline concentrate	1.2	28.6	3.7	43.6	1.3	12.95	7.55	0.06
Kiya-Shaltyr nepheline ore	4.3	26.6	4.6	40.0	7.9	11.05	2.74	0.25

Table II. Performances of the production of some materials in the USSR by traditional methods

Products	Technology	Production cost * rbl/t	Price rbl/t	Specific capital expenses * rbl/t / year	Consumption ratios per 1 t of products			
					Fuel kg of standard fuel	Steam Gcal	Electric power kWh	Total consumption of power kg of standard fuel
"Heavy soda"	Salve's ammonia method with steam calcination	66.7	55	363.5	93	1.96	88	454
Potash	Electrolysis of KCl solution and carbonization	162.7	165	305.7	185	2.6	62	647
Portland cement	Clay and limestone calcination by dry method **	13.4	18.5	59.3	105.1	-	115	141.9

* with utilization of all production wastes

** for the plant in the center of European part of the USSR

The processing of the Kola concentrate gives per ton of alumina 0,749 t of soda, 0,295 t of potash, 1,5 kg of potassium sulphate, 10,216 t of cement and gallium. The chemical extraction from the sinter is 87,8% of Al_2O_3, 89,1% of Na_2O, 86,7% of K_2O. The extraction from the raw material into the finished products is: Al_2O_3 - 84.2%, Na_2O - 83.2%, K_2O - 74.7%, SO_3 - 61%, Cl - 37%.

In the processing of nepheline ore we obtain per 1 ton of alumina 0.635 t of soda, 22 kg of potash, 53 kg of potassium sulphate, 1.5 kg of potassium chloride, 4.01 t of cement (using 33% of the belite mud due to limited capacity of the cement production facilities) and gallium. The chemical extraction from the sinter is: Al_2O_3 - 83.1%, Na_2O - 84.6%, K_2O - 73.94%. The extraction from the raw material into the finished products is: Al_2O_3 - 80%, Na_2O - 70%, K_2O - 59%, SO_3 - 67%, Cl - 16%. The decrease in alkali yield is explained by the fact that about 20% of carbonate liquor is used for the feed charge preparation due to low alkaline module of the raw material (0.8).

The total alumina produced by processing of nepheline rocks in the USSR is about 1.2 mln tpy.

The flowsheet of alumina production from nepheline by sintering [1,2] is given at Fig.1.

The process includes the wet grinding of nepheline ore (concentrate), limestone and recycled white mud (after liquor desilication) till particle size under 0.08 mm. The ground feed charge is subjected to

Table III. Technical and economical performances of alumina-containing raw material processing in the USSR

Raw material composition	Technology	Actual performances of existing enterprises					
		Price of alumina-containing raw material rbl/t	Production cost of alumina to production cost of all products rbl/t of alumina	Cost of all products related to 1 t of alumina rbl/t	Profit related to 1 t of alumina, rbl/t	Specific capital investments related 1 t of alumina rbl/t/year	Profit to specific capital inrestment ratio, 1/year
Imported gibbsite low-silica bauxites	Bayer	20	137/137 [*]	168.3	31.3	534	0.059
Imported low-silica diaspore-boemite bauxites	Bayer-sintering (in parallel)	29	146.6/146.6 [*]	168.6	22.0	342	0.064
High-silica gibbsite bauxites	Bayer-sintering (in series)	14	124.6/124.6 [*]	169.2	44.6	274	0.163
High-silica diaspore-boemite bauxite	sintering	19	198/198 [*]	160	-38	558	-0.068
Nephelines	sintering	7	72.8/288.2	436.5	148.3	807	0.184
Alunites [**]	reducing calcination, hydrochemical way	7.5	114.1/256	294	38	406	0.094

cont.table III

Raw material composition	Technology	Calculated performances with the same amount of end product per ton of alumina				
		Production cost of all products rbl/t of alumina	Cost of all products rbl/t of alumina	Profit rbl/t of alumina	Specific capital investments rbl/t/year	Profit to specific capital investments 1/year
Imported gibbsite low-silica bauxites	Bayer	371.7	447.3	75.6	1522	0.05
Imported low-silica diaspore-boemite bauxites	Bayer-sintering (in parallel)	380.7	447.5	66.8	1349	0.05
High-silica gibbsite bauxites	Bayer-sintering (in series)	359.7	448.1	88.4	1288	0.069
High-silica diaspore-boemite bauxite	sintering	432.7	438.9	6.2	1603	0.004
Nephelines	sintering	288.2	436.5	148.3	1064	0.139
Alunites [**]	reducing calcination, hydrochemical way	-	-	-	-	-

[*] only alumina is produced;

[**] in joint processing with imported bauxites;

[***] for the projected plant

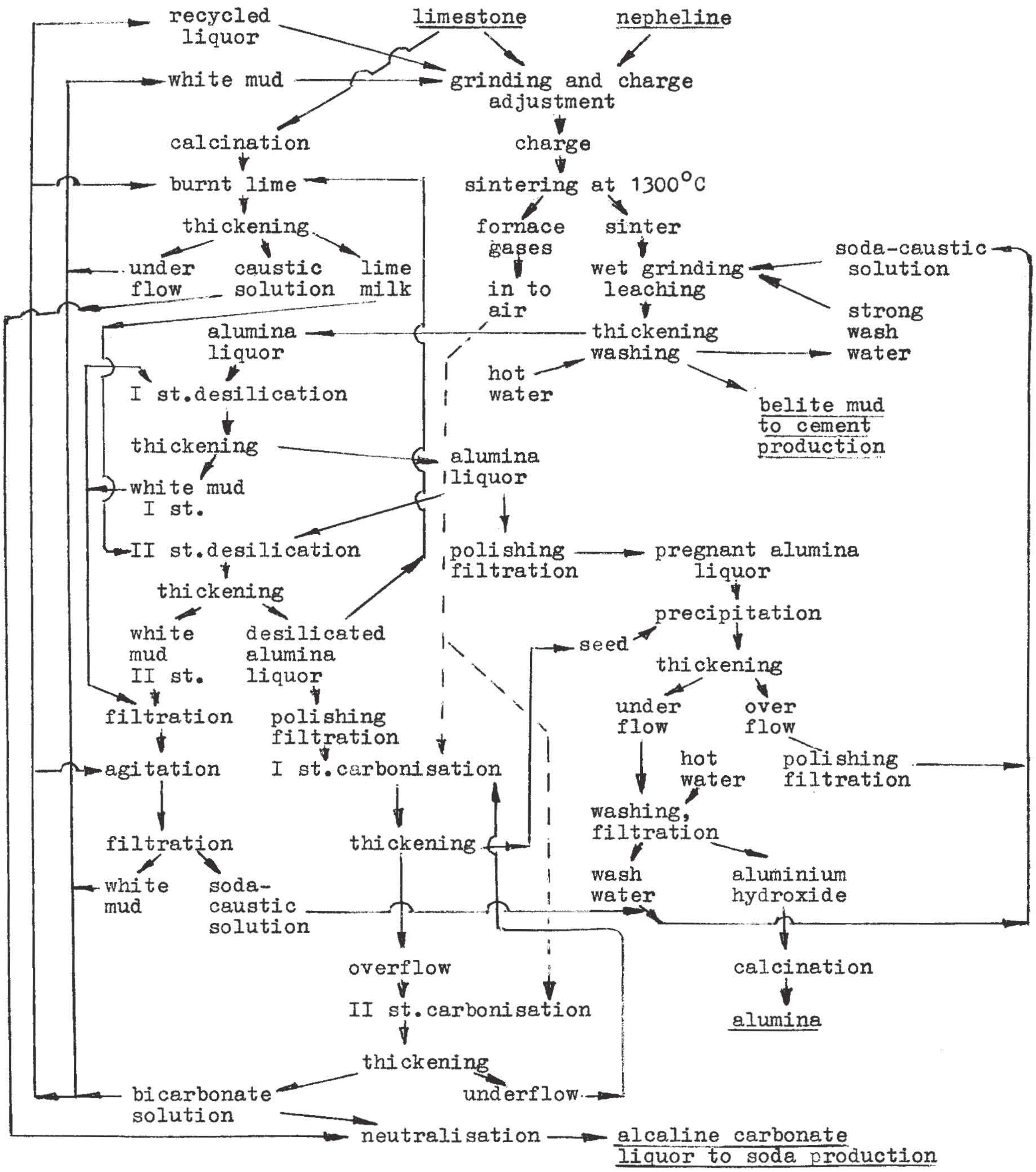

Figure 1: Flowsheet of alumina production from nepheline

sintering at 1300-1350°C. Here takes place the main reaction of interaction between nepheline and limestone with formation of alkaline aluminates and bicalcium silicate:

$$(Na,K)_2O.Al_2O_3.2SiO_2+4CaCO_3=(Na,K)_2O.Al_2O_3+$$
$$+ 2(\beta -2CaO.SiO_2) + 4CO_2$$

The feed charge with the moisture content about 30% is subjected to sintering in the rotary kilns. As a fuel is used the low sulphur fuel oil containing under 1% of sulphur. The sintering results in the sinter - porous particles mostly under 20 mm size. The sinter is then subjected to leaching including wet grinding till particle size 1 mm, thickening of the slurry with obtention of the pregnant liquor (silica ratio 30-35) and water washing of the belite mud. During the leaching of the sinter the alkaline aluminates pass into the liquor:

$$Na_2O.Al_2O_3 + 4H_2O = 2Na^+ + 2Al(OH)_4^-$$

and bicalcium cilicate is partially decomposed giving hydrogranate and passing into the solution of silicium oxide:

$$2CaO.SiO_2+Al(OH)_4^- +aq \rightarrow 3CaO.Al_2O_3.0,4\ SiO_2.5,2\ H_2O + SiO_3^{2-} + aq$$

With thickening of the slurry pregnant liquor is obtained containing about 85 g/l of Al_2O_3, 2,5 - 3 g/l of SiO_2, $\alpha_k \sim 1.48$, which is then subject to hydrochemical treatment to obtain aluminium hydroxide. The belite slurry containing mainly β-$2CaO.SiO_2$ is subject to repulpation washing 6 times by hot water in the system of thickeners and then sent to cement production. Hydrochemical treatment of the pregnant liquor comprises the autoclave desilication using recycled white mud seed, the separation of the liquor flow in two parts one of which is subject to seed precipitation to obtain aluminium hydrate product and soda-alkaline liquor returned for sinter leaching. The other part of the liquor is subject to desilication with addition of lime reactant and to carbonization. White mud generated in the liquor desilication is filtered and sent to feed charge preparation for sintering.

In autoclave desilication (at approximately 150°C) the sodium hydroalumosilicate is generated:

$$Al(OH)_4^-+Na^++SiO_3^{2-}+aq \rightarrow Na_2O.Al_2O_3.1,75SiO_2.2H_2O$$

and with intensive desilication (at about 90°C) using lime-containing reactants yielding hydrogranate:

$$Ca(OH)_2 + Al(OH)_4^- + SiO_3^{2-} + aq \longrightarrow 3CaO.Al_2O_3.0.15SiO_2.5.7H_2O$$

The both said compounds generate white mud.

With big amounts of white mud its treatment with carbonate solution to generate aluminium oxide is used:

$$3CaO.Al_2O_3.0,15SiO_2.5,7H_2O + CO_3^{2-}+aq \longrightarrow CaCO_3 + Al(OH)_4^- + SiO_3^{2-} + aq$$

Carbonization of the pregnant liquor is carried out at about 70°C in one or two stages till various degrees of neutralization of caustic (1,5 g/l of Na_2O_k - 11 g/l of $NaHCO_3$) using the recycled seed.

The technology of the liquor carbonization determines the coarseness and the impurities content in the resulting aluminium hydroxide which is then sent as a seed to the liquor precipitation stage.

The resulting pure white alumina contains 0,22-0,25 % of R_2O_{rec}, 0,022-0,032% of SiO_2, 0,008-0,01% of Fe_2O_3. The reduced (as much as 3 times) content of iron oxide in alumina is characteristic of this method compared to traditional Bayer method which can be explained by feedle solubility of iron compounds in sinter and by reduced concentration of liquor in the hydrochemical cycle.

By its physical structure the obtained alumina is somewhere between the floury and the sandy types. Till present times the sandy alumina was not produced because the gross price of alumina in the USSR did not depend on its coarseness and considers only its chemical composition.

In processing of the nepheline ore can be obtained the alumina with low content of such impurities as Na_2O, SiO_2, Fe_2O_3 which is very important for its utilization for the non-metallurgical purposes. Particularly the silicium oxide content may be easily reduced through the increase of lime consumption in deep desilication and the alkali content can be reduced by alteration of conditions of carbonization and precipitation.

After carbonization the carbonate alkaline liquor is sent to neutralization of bicarbonates, evaporation and separation of soda products (soda, potash, potassium sulphate and chloride). The flowsheets employed (Fig.2) vary with the composition of liquor and quality requirements for resulting salts.

After sinter leaching the belite mud

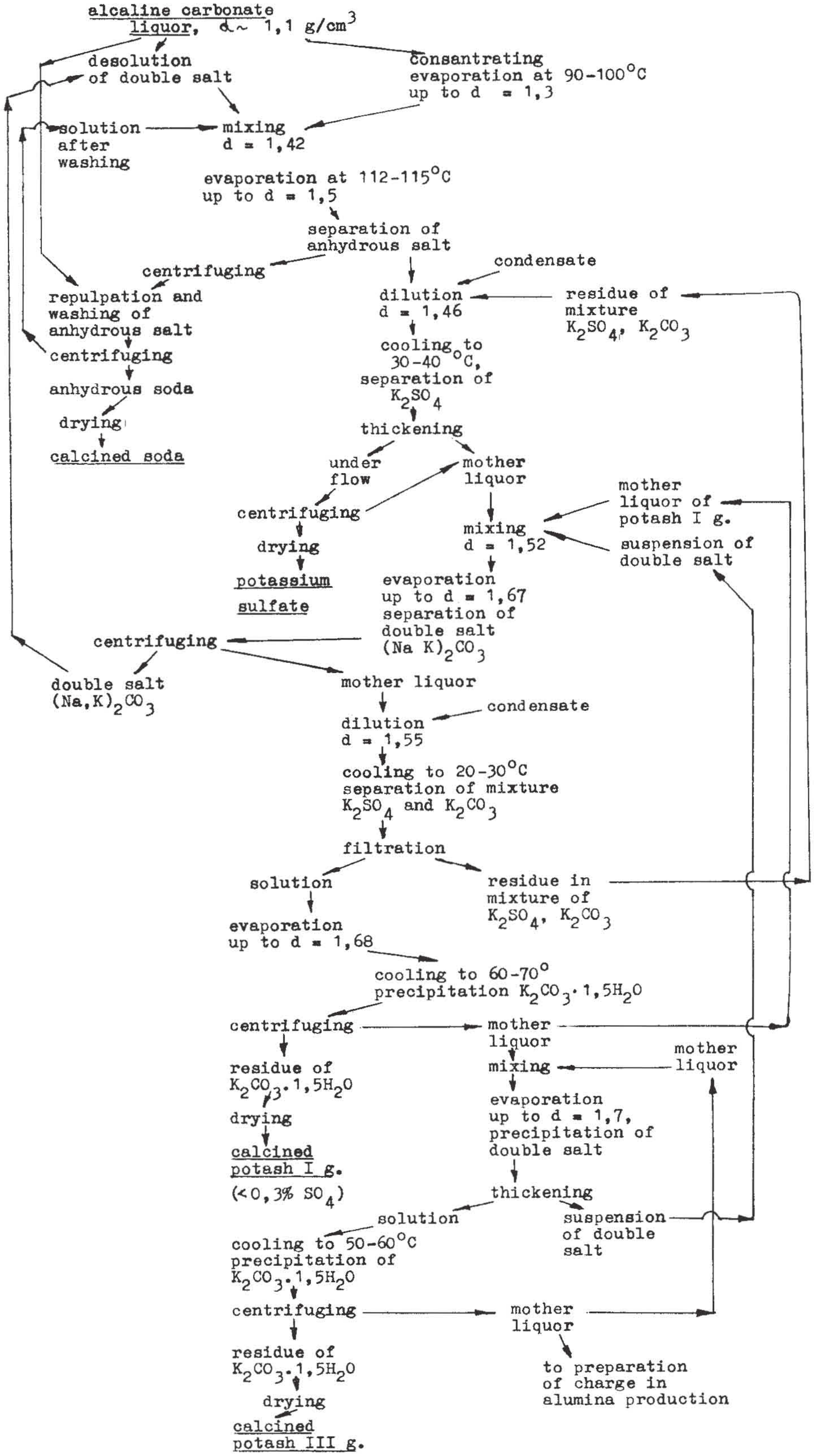

Figure 2: Flowsheet of soda and potash production from nepheline

is sent to cement production facility which permits to increase its efficiency by 27-33% as a result of reduction of limestone consumption, for the main phase is the bicalcium silicate. The cement production flowsheet is given in Fig.3.

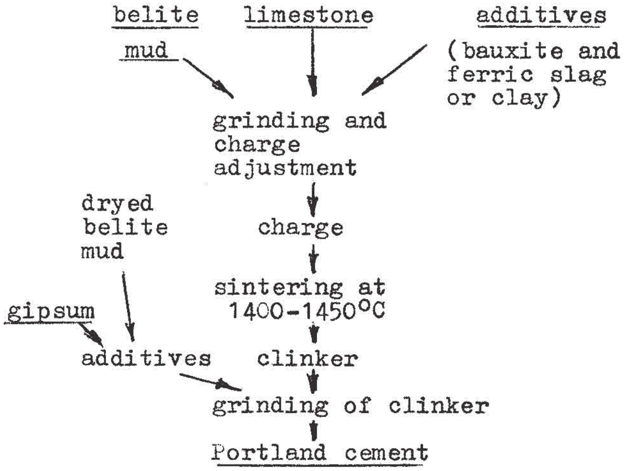

Figure 3: Flowsheet of cement production from nepheline

In cement production using the belite mud resulting after the processing of the Kola peninsula nepheline concentrate used the feed charge consisting of the belite mud, limestone, bauxite and ferrous metallurgy slags (the consumption being correspondingly 692, 621, 58 and 14 kg per ton of clinker) is used to produce the clinker characterized by the following correlations:

$$KH = \frac{CaO - 1,65\ Al_2O_3 - 0,35\ Fe_2O_3}{2,8\ SiO_2} = 0,92$$

$$n = \frac{SiO_2}{Al_2O_3 + Fe_2O_3} = 3.2$$

$$p = \frac{Al_2O_3}{Fe_2O_3} = 1,05$$

In processing the Kiya-Shaltyr ore, given the absense of bauxite and the increased content of alkali in the mud the feed charge is used which consist of belite mud, limestone and clay (consumptions being 530, 700 and 68 kg per ton of clinker) to obtain the clinker with following correlations: KH = 0,91, n = 2,43, p = 1,06.

In clinker grinding in order to control the setting time of cement the gypsum is added in the reason of 35-58 kg per to of cement and up to 15% of dry belite mud.

In wasteless processing of the nepheline ore about 30% of gallium is extracted too, which is connected with some additional expenses.

Process performances of alumina production from the nepheline ore

In the process of preparation of feed charge for the sintering the specific consumptions ratio per 1 ton of alumina for the white mud (q_{wm}) and for the nepheline ore (q_n) is q_{wm}/q_n = 0,12 and varies with the technology of liquor desilication. Below is given the tentative composition of the solid phase of the recycled white mud:

LOI	Al_2O_3	SiO_2	CaO	R_2O*
30	16.2	10.4	36.6	5.2

The K_2O/R_2O in white mud and in nepheline ore according to industrial data is expressed by the following equation:

$$\frac{K_2O_{wm}}{R_2O_{wm}} = \frac{K_2O_n}{R_2O_n} - 0,082$$

which demonstrates that the most of the sodium alkali has passed into the white mud.

The proportioning of the limestone (q_l) and of the recycled liquor on the basis of total alkalis ($q_{R2O\ tot}$) for feed charge preparation is imposed by the necessity to maintain molar ratio in the feed charge best suited for the process:

$$\frac{R_2O_{ch}}{Al_2O_{3\,ch}} \sim 0.98\text{-}1.1; \quad \frac{CaO_{ch}}{SiO_{2\,ch}} \sim 1,95\text{-}2.0$$

The optimum ratio of the feed charge components corresponds to the maximum efficiency of the process taking into consideration the variations in the chemical extraction from the sinter, in fuel consumption during sintering, in the productivity of the rotary kilns, in the composition of the sinter and other factors,

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* The summary of Na_2O and K_2O expressed as Na_2O

and is determined through the statistical processing of industrial data.

With $\frac{R_2O_{ch}}{Al_2O_{3_{ch}}} = 1.05$; $\frac{CaO_{ch}}{SiO_{2_{ch}}} = 2.0$ in the feed charge the limestone and total recycled alkali to nepheline ratio used in the feed charge preparation is the following:

$$\frac{q_L}{q_n} = \frac{1.867\ SiO_{2\ n} - CaO_n - 0.0206}{CaO_L - 1.867\ SiO_{2\ L}} \qquad (1)$$

$$\frac{q_{R_2O_{rec}}}{q_n} = 0.6383\ (Al_2O_{3\ n} + \frac{q_L}{q_n} \cdot Al_2O_{3_L}) - R_2O_n - \frac{q_L}{q_n} \cdot R_2O_L + 0.00617 \qquad (2)$$

where q_L, q_n, $qR2O_{rec}$ are the consumption values of limestone, nepheline and total alkalis (taken as recycled liquor) in the feed charge preparation per ton of alumina; SiO_2, CaO, Al_2O_3, R_2O, Na_2O, K_2O, LOI - the content of oxides in materials, in fractions of unit; indices n, L, wm, rec describe nepheline, limestone, white mud and recycled liquor.

As follows from equation (1) the content of SiO_2 in nepheline and limestone is proportional to the limestone consumption for the feed charge preparation, therefore in alumina production is used mainly the limestone containing under 2,5% of SiO_2 and with decreasing of the molar ratio in nepheline ($R_2O_n/Al_2O_{3\ n} < 1,05$) the amount of recycled carbonate liquor for feed charge preparation increases (2).

Taking into consideration that the content of Al_2O_3 in alumina is 99%, the mechanical losses in sintering and grinding are about 1% of the sinter, losses in liquor treatment and in calcination are 0.44% of the aluminium oxide that has passed into the liquor during leaching, the specific consumption of nepheline is expressed by a value of chemical extraction of Al_2O_3 during sinter leaching - X_A (fractions of unit):

$$q_n = \frac{0{,}99}{(Al_2O_{3\ n} + \frac{q_L}{q_n} \cdot Al_2O_{3_L}) - (Al_2O_{3_n} + \frac{q_L}{q_n} \cdot Al_2O_{3_L} + 0{,}01944) \cdot (0{,}99 - 0{,}98564 \cdot X_A)} \qquad (3)$$

The specific consumption of the charge feed to the sintering is equal (in tons per ton of alumina):

$$q_{ch} = 1{,}12 q_n + q_L + 1{,}71 \cdot q_{Na_2O_{rec}} + 1.47 \cdot q_{K_2O_{rec}} \qquad (4)$$

LOI in the feed charge are:

$$LOI_{ch} = \frac{LOI_n + 0{,}036 + LOI_L \cdot \frac{q_L}{q_n} + 0{,}71 \cdot \frac{q_{R_2O_{rec}}}{q_n}}{q_{ch}/q_n} \qquad (5)$$

Taking into consideration that the LOI in the sinter are 0.7% and the sinter losses are about 1%, the specific consumption of sinter, q_{sin}, arriving to leaching (tons per ton of alumina) is:

$$q_{sin} = \frac{q_{ch}\ (1 - LOI_{ch}) \cdot 0.99}{0{,}993} \qquad (6)$$

The components of the sinter (fractions of unit) considering the industrial data on relative losses of sodium alkalis - 0.8% and of potassium alkalis - 3,77% of the feed charge with flue gases as a result of dissociation of the compounds during sintering and incomplete condensation in the kiln and in dry gas cleaning are:

$$Al_2O_{3_{sin}} = 0{,}99 \cdot \frac{q_n}{q_{sin}} \cdot (Al_2O_{3_n} + \frac{q_L}{q_n} Al_2O_{3_L} + 0.01944)$$

$$CaO_{sin} = 0.99 \cdot \frac{q_n}{q_{sin}} \cdot (CaO_n + \frac{q_L}{q_n} \cdot CaO_L + 0.0439)$$

$$(7) \quad SiO_{2_{sin}} = 0.99 \cdot \frac{q_n}{q_{sin}} \cdot (SiO_{2_n} + \frac{q_L}{q_n} \cdot SiO_{2_L} + 0.0125)$$

$$Na_2O_{sin} = 0.982 \cdot \frac{q_n}{q_{sin}} \cdot (Na_2O_n + 0.12 Na_2O_{wm} + \frac{q_L}{q_n} \cdot Na_2O_L + \frac{q_{Na_2O_{rec}}}{q_n})$$

$$K_2O_{sin} = 0.953 \cdot \frac{q_n}{q_{sin}} \cdot \left(K_2O_n + 0.12 K_2O_{wm} + \frac{q_L}{q_n} \cdot K_2O_L + \frac{q_{K_2O_{rec}}}{q_h}\right)$$

$$LIO_{sin} = 0.007$$

Taking into consideration the extraction into the pregnant liquor during sinter leaching is in the order of 1,2% of SiO_2 as well as aluminium oxide (X_A), sodium oxide (X_N), potassium oxide (X_K) or total alkalis (X_R) and some hydratation of the mud (about 3% LOI) we obtain the specific consumption of belite mud, q_{mud}, (in tons per ton of alumina):

$$q_{mud} = 1.03 \cdot q_{sin} \cdot (0.993 - Al_2O_{3_{sin}} \cdot X_A - Na_2O_{sin} \cdot X_N - K_2O_{sin} \cdot X_K - 0.012 \cdot SiO_{2_{sin}}) \tag{8}$$

In the same way as in (7) can be calculated the components content in the belite mud.

The concentration of the pregnant liquor in g/l (content of $Al_2O_3 = C_s^A$, of the total alkalis - R_2O_{tot} - $C_s^{R_2O_{tot}}$, with preset $\alpha_k = 1{,}48$ and R_2O_{carb} content about 12 g/l) and pregnant liquor consumption (q_s) in m^3 per ton of alumina are determined through solution of the system of equations for sinter leaching process, Here the moisture content of the washed thickened mud is 40% and the required water consumption for mud washing (q_{wmw}) is proportional to the specific consumption of mud and to the concentration of pregnant liquor:

$$q_{wmw} = 0{,}0203 \cdot q_{mud} \cdot C_S^A \tag{9}$$

Concentration of the recycled soda-alkaline liquor sent to the sinter leaching and obtained after precipitation and dilution of the hydrate with washwater is:

$$C_{SA}^A = 0.301 \cdot C_S^A; \quad C_{SA}^{R_2O_{tot}} = 0.84 \cdot C_S^{R_2O_{tot}}$$

Making equations for extraction of Al_2O_3 and R_2O from the sinter we obtain the system of two equations which allows to determine the Al_2O_3 concentration in the pregnant liquor - C_S^A (g/l) and the alkaline liquor consumption (q_{SA}):

$$\begin{aligned}
&(q_{SA} + 0.0203 \cdot q_{mud} \cdot C_S^A - 0.67\, q_{mud}) \cdot C_S^A = 0.301 \cdot C_S^A \cdot q_{SA} + 1000 \cdot q_{sin} \cdot Al_2O_{3_{sin}} \cdot X_A;\\
&(q_{SA} + 0.0203 \cdot q_{mud} \cdot C_S^A - 0.67 q_{mud}) \cdot (0.9 \cdot C_S^A + 12) = 0.84 \cdot (0.9 \cdot C_S^A + 12) \cdot q_{SA} + 1000 \cdot q_{sin} \cdot R_2O_{sin} \cdot X_R
\end{aligned} \tag{10}$$

Here the pregnant liquor consumption is:

$$q_S = q_{SA} + q_{wmw} - 0.67 \cdot q_{mud} \tag{11}$$

The amount of the sinter to be processed (q_{sin}) and of the pregnant liquor (q_s) per ton of alumina are the main parameters determining the specific power consumption, equipment productivity and specific capital expences in alumina production.

The extraction of Na_2O and K_2O from the raw material into the product per ton of alumina (q_{Na_2Opr}, q_{K_2Opr}) considering the losses according to industrial data are, correspondingly, 4.11% and 3.97% of those that have passed into the liquor during sinter leaching is:

$$\begin{aligned}
&q_{Na_2O_{pr}} = q_{sin} \cdot Na_2O_{sin} \cdot X_N \cdot 0.9589 - q_{Na_2O_{rec}} - 0.12\, q_n \cdot Na_2O_{wm}\\
&q_{K_2O_{pr}} = q_{sin} \cdot K_2O_{sin} \cdot X_K \cdot 0.9603 - q_{K_2O_{rec}} - 0.12 q_n \cdot K_2O_{wm}
\end{aligned} \tag{12}$$

Here the carbonate liquor has to have the following Na_2O/K_2O ratios:

$$\frac{q_{Na_2O_{pr}}}{q_{K_2O_{pr}}} = \frac{q_{Na_2O_{rec}}}{q_{K_2O_{rec}}} \tag{13}$$

which calls for iterative calculations starting with (4).

The passage of SO_3 into the product is determined by the total balance of the sulphur in the fuel fed to the feed charge sintering ($q_f sin$):

$$q_{SO_{3_{pr}}} = (q_n \cdot SO_{3_n} + q_L \cdot SO_{3_L} + q_f\, sin \cdot SO_{3\,f}) \cdot T_{SO_3} \tag{14}$$

The same for Cl:

$$q_{Cl_{pr}} = (q_n \cdot Cl_n + q_L \cdot Cl_L) . T_{Cl} \qquad (15)$$

where T_{SO_3}, T_{Cl} - SO_3 and Cl extraction into the product from the raw material (fractions of unit).

Due to the fact that SO_3 and Cl bind first of all potassium into sulphate and chloride the following salts are obtained as end products:

K_2SO_4 $\quad q_{K_2SO_4\,pr} = 2.175 \cdot q_{SO_3\,pr}$

KCl $\quad q_{KCl_{pr}} = 2.099 \cdot q_{Cl_{pr}}$ $\quad$ (16)

K_2CO_3 $\quad q_{K_2CO_3\,pr} = (q_{K_2O_{pr}} - 1.175 . q_{SO_{3_{pr}}} - 1.324 q_{Cl_{pr}}) . 1.468$

Na_2CO_3 $\quad q_{Na_2CO_3\,pr} = 1.71 . q_{Na_2O_{pr}}$

As every end product - soda (q_{sd}), potash (q_p), potassium sulphate ($q_{s\,ph}$), potassium chloride (q_{ch}) contains all four sals, in order to determine the yield of products one has to solve the system of four equations (16) considering the salts yield and the composition of products which is the function of the carbonateliquor composition and the technology of its treatment.

The amount of the carbonate liquor, q_{Cl}, (in m^3 per ton of alumina) passing from the alumina production to evaporation and separation of salts is determined by the total balance of the hydrochemical processes and with water consumption for the hydrate washing (m^3/t), water content in the washed hydrate being 10%, with the balance of dilution with live steam during desilication and of evaporation of liquors into atmosphere, not considering the water content of the white mud returned to the sintering, is:

$$q^{L}_{Cl} = (1.37 + q_{wmw} - 0.67 . q_{mud}) . \frac{q_{R_2O_{pr}}}{q_{R_2O_{pr}} + q_{R_2O_{rec}}} \qquad (17)$$

The amount of carbonate liquor after evaporation determines the steam consumption in the soda products separation.

In preparation of the raw material mixture for cement production consisting of belite mud, limestone and clay to obtain in clinker the values of KH = 0.92, n=3.2, p = 1.05; with addition of 15% mud for cement and gypsum grinding; the cement yield per ton of alumina is:

$$q_c = 1.65 \cdot q_{mud} \qquad (18)$$

The exposed system with certain additions in the form of computer program determines the main parameters of material balance, raw material consumption and quality of products obtained in processing the nepheline raw material.

The power consumption and capital expenses can be determined considering the performances of all process areas found in the industrial experience of the nepheline ore processing in the USSR.

Approximate assessment of nepheline ore quality

For approximate assessment of nepheline ore quality its composition and values of components extraction from the sinter - $Al_2O_3(X_A)$, $Na_2O(X_N)$, $K_2O(X_K)$, yield of SO_3 (T_{SO3}) and of Cl (T_{Cl}) are needed; afterwards, in order to determine the product yield (soda, potash, potassium sulphate and chloride, cement, sinter) per ton of alumina the calculations are made using equations (1-6), (12-16), (18). According to industrial practice the extraction of components from the sinter:

Al_2O_3 - $X_A = 0.83 - 0.88$

Na_2O - $X_N = 0.86 - 0.892$

K_2O - $X_K = 0.76 - 0.87$

The yield from the raw material is 61-67% of SO_3, 16-37% of Cl. As an extraction values of soda, potash, sulphate and chloride per ton of alumina the corresponding salt amounts can be used.

In simplified and approximated form the product yield can be calculated using the yield of Al_2O_3 from the raw material, $T_A \sim 0.82$ and the yield of alkalis without their recyclation for sintering, $T_R = 0.77 - 0.81$. In this case neglecting the aluminium oxide and alkalis content in the limestone one can make calculations in the following sequence:

$$\frac{q_L}{q_n}\ (1);\quad q_n = \frac{0.99}{Al_2O_{3_n} \cdot T_A} \qquad (3^*);$$

$$\frac{q_{R_2O_{rec}}}{q_n} = 0.6383 . Al_2O_{3_n} - R_2O_n + 0.00617 \qquad (2^*);$$

$$\frac{q_{ch}}{q_n} \sim 1.12 + \frac{q_L}{q_n} + 1.71 . \frac{q_{R_2O_{rec}}}{q_n} \qquad (4^*);$$

$$LOI_{ch}\ (5);\quad q_{sin}\ (6);\quad q_{R_2O_{pr}} = q_n . R_2O_n -$$

$$-(q_n . R_2O_n + q_{R_2O_{rec}} + 0.00624 . q_n) .$$

$$.(1-T_R) \quad (12^*);\quad q_{SO_{3\,pr}}\ (14);\quad q_{Cl_{pr}}\ (15);$$

$$q_{salts}\ (16);\quad q_c \sim q_{sin} \cdot 1.32 \qquad (18^*).$$

To calculate the extraction of Na_2O and K_2O into the products the same oxides ratio in initial raw material can be adopted and for the yield of soda, potash, potassium sulphate and chloride the corresponding amounts of salts can be used.

In the dimensionless form the cost of all products - Q per 1 ton of alumina is:

$$Q = 1 + q_{sd} \cdot K_s + q_p \cdot K_p + q_{sul} \cdot K_{sul} + $$
$$+ q_{ch} \cdot K_{ch} + q_c \cdot K_c \qquad (19)$$

where K_S, K_p, K_{sul}, K_{ch}, K_C - costs of soda, potash, potassium sulphate and chloride, cement related to cost of alumina (fractions of unit).

Due to the fact that the power and capital expenses in the process are proportional to the amounts of the processed sinter the efficiency of the raw material processing with constant price of alumina within one country with convertible currency circulation is determined by the following relationship:

$$J_1 = \frac{Q}{q_{sin}} \qquad (20)$$

As the main component of power consumption is the fuel consumption the price of which is different in various countries in relation to alumina using the fuel/alumina price ratio, K_T, one can obtain the relationship between the product cost and power consumption which gives the relative efficiency of raw material processing in various countries:

$$J_2 = \frac{Q}{q_{sin} \cdot K_T} \qquad (21)$$

where K_T is fuel (1 ton of standard fuel) and alumina price ratio.

Both parameters, J_1 and J_2, may be used as an index of nepheline ore quality for evaluation of the raw material reserves, for determining the explotation area in order to find out approximate efficiency of its processing which gives the possibility to take into consideration price level of alumina, soda products, cement and fuel in various countries.

In particular, we have made calculations of quality index for nepheline ore used in the USSR as well as nepheline ore of Razgah deposit (North-West of Iran) and of Agarai deposit (Pakistan) the tentative composition of which is given below.

Deposit	Components content, %				
	LOI	Al_2O_3	Na_2O	K_2O	SiO_2
Razgah (Iran)	3.1	20	3.6	10	54.3
Agarai (Pakistan)	1.0	23.5	10.63	3.96	55.54

Cont.

Deposit	Components content, %				
	CaO	Fe_2O_3	MgO	P_2O_5	SO_3
Razgah (Iran)	2.2	3.7	0.9	0.4	0.05
Agarai (Pakistan)	1.85	2.06	1.0	-	-

In table IV are given values of product yield per 1 ton of alumina in processing of various raw materials, interrelationship of prices as well as calculated values of raw material quality index, J_1 and J_2, in relation to its processing in the USSR, Iran and Pakistan.

The maximum value of parameter J_2 for Iran was calculated indicating the minimum level of energy consumption compared to the

Table IV. Parameters specifying the quality of various nepheline raw material

Parameter	Name of nepheline deposit			
	Existing plants		Calculated values	
	USSR Kola concentrate	USSR Kiya-Shaltyr	Iran Razgah	Pakistan Agarai
Specific yield of products per 1 t of alumina, in t				
soda (q_{sd})	0.75	0.635	0.264	0.765
potash (q_p)	0.295	0.022	0.601	0.007
potassium sulphate (q_{sul})	0.0015	0.052	0.017	0.291****
potassium chloride (q_{cl})	-	0.0015	-	-
cement (q_c)	10.216	12.15*	19.33	15.24
Products price to alumina price ratio				
soda (K_s)	0.317	0.325	0.952	0.545
potash (K_p)	0.927	0.947	3.0	1.847
potassium sulphate (K_{sul})	1.0	0.971	1.167	0.69
potassium chloride (K_{cl})	-	0.257	-	-
cement (K_c)	0.113	0.137	0.252	0.169
Parameter Q (19)	2.667	2.954	7.945	4.206
Sinter consumption per 1 ton of alumina, t q_{sin}	7.95	8.59	13.785	11.5
Fuel price (t of standard fuel) to price of alumina ratio, K_T	0.167	0.162	0.0977***	0.219
Price of alumina, C_{Al}**	164 rbl	169 rbl	21000 Rls	6629 Rs
Nepheline ore quality				
J_1(20)	0.335	0.343	0.576	0.365
J_2(21)	2.01	2.11	5.9	1.67

* with complete utilization of mud

** 1 $ = 18.76 Rs = 70 Rls

*** natural gas, in other case - fuel oil

**** considering transition of the bulk of potassium carbonate into potassium sulphate

cost of the products. With utilization of fuel oil as a combustible having a price per 1 ton of standard fuel by 18% lower than natural gas the parameter J_2 value would have been even higher.

For Pakistan the value of J_2 is somewhat lower than that for the Soviet nepheline which is explained by relatively high fuel prices.

Evaluation of the raw material by quality index J_2 is of course rather approximate because the capital investments in the raw material processing were not considered.

The cited evaluation and economical assessment results prove that nepheline raw material even with low content of aluminium oxide alkalis and with elevated content of silicium oxide can be efficiently processed to yield alumina, soda products and cement.

Outlook for technology improvement

Further improvements of technology of nepheline ore processing through sintering were achieved recently. In order to reduce the power consumption the dry ways of feed charge sintering in alumina and cement production are employed; the degree of utilization of the secondary steam in autoclave desilication and liquor evaporation, the high polymeric flocculants in the slurry thickening also contribute to improvement of the process.

Efficiency of raw material processing can be increased in the production of finely dispersed aluminium hydroxide, in the production of alumina with reduced impurities content for non-metallurgical purposes and with elevated silicium oxide content for fabrication of Al-Si alloys by electrolysis, in the production of caustic alkaline liquors, sodium hydroaluminate, in the extraction of rubidium from alkaline carbonate liquors.

The synthetic ceolites as well as foundry, stressed, non shrinking, expanding and white alumina-containing cements possessing several advantageous properties can be produced from the white muds of alumina production.

Belite mud can be efficiently used [3] for the fabrication of binders, silicate bricks, for road construction, for ceramics fabrication, for production of some valuable chemicals etc.

In order to increase the quality of nepheline and other alkaline alumosilicate raw materials the magnetic separation can be employed to remove iron minerals, as well as chemical beneficiation consisting in alkaline autoclave treatment to remove partially silicium oxide from the raw material in the form of silicium alkaline liquor used for fabrication of various chemicals.

Conclusions

The wasteless utilization of the nepheline raw material for production of alumina, soda, potash, Portland cement and gallium by sintering is characterized by complete utilization of production wastes, by absence of any environmental impact, by reduced capital investment and production costs compared with the production of the said products by traditional methods.

The perspectives of the process improvement, of expansion of the product mix, of raising it quality promise high efficiency of nepheline raw material utilization for production of alumina.

Certain advantage of the developed technology is the possibility to arrange the production of alumina and other products from the local alkaline alumosilicate raw material using cheap fuel - natural gas and fuel oil in oil producing countries - and lignite. The reduction of transportation cost and the cost of produced alumina compared with the imported one allows to achieve the high efficiency of aluminium production in countries not possessing high grade bauxites.

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