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PHOSPHATE ROCKS: IMPORTANT FACTORS IN THEIR ECONOMIC AND TECHNICAL EVALUATION

by

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and

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In comparison to phosphate resources, most of the other industrial minerals essential to man have much higher unit values. Furthermore, they generally have more narrowly defined geochemical origins, concentration requirements, and end-use markets. Consequently, an economic assessment of their mine production costs in relation to market value is more easily and relibbly defined.

Thus, the mere knowledge of existing or newly discovered ore bodies provides economic incentives for exploitation plans--even in the most remote geographical locations, as is well illustrated by iron, copper, tin, lead, and similar mineral commodities.

Unfortunately, this is not the case with phosphate resources. Much of the known vast reserves of 50 to 100 billion short tons (st), which are favorably dispersed among the worldwide markets, remain uneconomical to recover, despite the growing need for phosphate in world food production.

As most authorities recognize, a complex set of interrelated technical and economic factors decide the economic potential of any particular phosphate reserve. Although technical similarities may exist among deposits, economic analogies are rare, indeed!

CENTO Symposium on the Mining and Beneficiation of Fertilizer Minerals, pp. 194-242 (November 19-24, 1973).

The economic evaluation of ore minability and mill preparation of concentrates must also give preliminary thought to the geological setting and character of the ore deposit, as well as to the eventual end-use markets and their locations.

There are two separate aspects to the evaluation of phosphate rock raw materials--those of the mine producer and the ultimate consumer. Part I deals with phosphate raw-material production, and reviews the above mentioned factors in sequence, drawing mainly upon authoritative commentary from the technical literature.

Part II deals with the ultimate consumer's interests, and briefly summarizes the technical evaluation methods developed in TVA fundamental research for comparing the physical and chemical quality of commercial concentrates. While this assessment of raw materials relates mainly to manufacturer's specifications, their basis for selecting raw materials for some particular process has, nevertheless, a direct and obvious effect on the actual market value of mine products from specific deposits.

This poses a potential threat to the continued economical production of certain types of commercial phosphate, because of the growing trend towards a more discriminate selection of raw materials by manufacturers.

PART 1 -

I: GEOLOGICAL OCCURRENCE AND STRUCTURE OF APATITE DEPOSITS

The Nature of Phosphate Raw Material

Commercial mineral phosphates, known collectively as "phosphate rock," have but one property in common. Their phosphorus and fluorine are combined in apatite mineral structures in association with calcium.

Calcium apatites occur as the 10th most abundant mineral in the earth's crust and occasionally in massive concentrations of economic importance. Despite their crystal-structure similarity, the compositions of these apatites usually show significant departures from that of fluorapatite, $Ca_{10}(PO_4)_6F_2$, which is a commonly assumed composition of phosphate rock. These differences in chemical composition are mainly responsible for the variable properties of commercial concentrates.

Geological Occurrences

The principal reason for the various compositional types of apatites can be readily traced to their geochemical origins. Apatitic phosphate minerals occur under all geological settings --igneous, metamorphic, and sedimentary environments--with further subdivisions required to represent the almost infinite variety of phosphate-containing geological structures that have been recognized or mapped.

The apatite recovered from igneous and metamorphic rocks, including byproducts from iron ore deposits, have commercial importance, but supply only a small fraction of the world market, and that mainly for captive production (Kola, USSR; Phalaborwa, S. Africa; Araxa and Jacupiranga, Brazil).

Because of this, as well as their individual uniqueness in geological structure, no further consideration of the minability of igneous and metamorphic occurrences will be given in this report. These special mining situations have been described elsewhere.[21]

Sedimentary apatitic phosphates have been, and no doubt will continue to be, the major source of commercial raw material. These sedimentary apatites, because of their widely differing modes of occurrence in geological periods ranging from Miocene to Precambrian, display a wide range of chemical compositions. Significant amounts of magnesium and sodium usually have substituted for calcium in their structures, and up to 25% of their phosphorus may be replaced by carbonate plus fluorine, placing these apatites in a distinct mineral class.[13, 14, 15]

Structures of Sedimentary Deposits

Sedimentary (Marine) phosphorites display wide variations in the physical forms of the apatite mineral and types of associated mineral gangue. This is due to their different geological and geochemical histories and the natural physical forces that were active during their time of deposition. Quite often, the original phosphate placement has been physically reconstituted by action of post-depositional meteoric forces, as in the well-known example of Florida clastic deposits. These differing geological histories are mainly responsible for the range of structural features displayed by sedimentary ore bodies.

Consequently, phosphate ore bodies vary in their total thickness, areal distribution, degree of dilution by gangue min-

erals, and the depth to which they were subsequently buried under more recent sediments.

Sometimes, post-depositional tectonic disturbances resulted in structurally complex deposits due to faulting, folding, subsidence, uplift, and related deformations.

Some deposits which represent enormous tonnage reserves of P_2O_5 comprise not one, but a series of beds intercalated with other sedimentary strata of waste material or very low-grade ore. Thus, the tonnage reserve of P_2O_5 may be either highly dispersed or concentrated in relation to the waste fraction, depending upon the type of interbedding.

This brief summary of some of the more important structural aspects of sedimentary ore bodies suffices to introduce the rather complex problem that confronts both the mine geologist and mining engineer in defining the economic minability of some particular deposit.

The full magnitude of this task will become even more apparent by referring to the published detailed descriptions of the major worldwide deposits. A collection of highly detailed and authoritative articles covering this subject has been compiled under United Nations auspices.[21]

Favorable Sedimentary Structures

With further regard to minability, certain structural types of sedimentary phosphate deposits more or less defy economic recovery by present standards.

Phosphate beds that are buried well below stripping depths, or have a steep dip, present highly unfavorable conditions for profitable recovery. The same applies to deposits comprising a spaced sequence of thin phosphate strata interbedded with thicker seams of waste, beds with thick (vertical) separations of waste, or beds that are buried under a dense indurated cap rock.

The above types are dismissed from further consideration. There are few, if any, commercial mines operating in such geological settings, except perhaps on a subsidized basis as a necessity to obtain ore for internal national interests. On the other hand, there are numerous examples to be cited where such structural features have stymied efforts to exploit known phosphate reserves.

The potentially minable reserves, therefore, are those occurring in one, or a few thick, high-grade, reasonably level beds having uniform texture and mineral composition, with shallow overburdens and a minimum of structural deformation. The logical geographical setting for these highly uniform structures are the continental coastal plains, rather than mountainous interiors where phosphate beds are likely to have been structurally deformed, or otherwise altered by the crustal disturbances. Limited ore recovery may still be possible if ores have exceptional grade, as for example, in some areas of the Western U.S. mountain states. Most of the major commercial deposits are situated, however, on coastal plains, and potential reserves of submarine phosphate beds are known on the adjacent continental shelves (Baja California, Florida, North Carolina, Northwest Africa).

Thus, while Emigh's estimate of 50 to 100 billion st of phosphate reserves removes any threat of a worldwide shortage,[8] the minable fractions under present standards restrict the discussion to a relatively small number of the several hundred known deposit sites.

II: MINABILITY--TECHNICAL AND ECONOMIC FACTORS

To define the minability of a low-value mineral such as phosphate requires a simultaneous consideration of an interrelated set of both technical and economic factors. According to Cathcart,[7] the geological character of the ore body largely governs the types of mining methods that may be used, but economic restraints may apply to some, or all, of these technical options.

The difficulty in making a practical assessment of minability is that no one factor is absolute. For each deposit there exists a unique combination that defines whether mining is both technically and economically feasible. Phosphate deposits differ so in their geological character that precisely parallel conditions rarely, if ever, exist.

From the case histories of successful mine operations, however, it is possible to establish some limits on the flexibility of geological factors that permit economic mining under current conditions. At present, this requires high-volume removal of waste overburden plus ore, large annual tonnage output, high recovery efficiency, and upgrading to prevailing market-grade specifications.

The basis for making this technical and economic assessment of minability changes rapidly with time. If recent trends continue, evaluations based on the present state of the mining and beneficiation art, acceptable product specifications, and market values may readily prove to be overly conservative or pessimistic.[5, 6]

Within recent years, for example, minimum ore-grade limits have decreased from about 18% to 10% P_2O_5; industry now accepts a wider range of lower grade concentrates, and market values of mine products have been advancing steadily, with further increases imminent.

No permanent significance can be attached, therefore, to the currently assigned limits of flexibility that define minability under present market conditions.

Factors Affecting Mining Economics

The following list of considerations that McBride[3] proposes are widely recognized by other authorities,[2,10] and their collective viewpoints are summarized in the discussions of individual factors.

Geological Character of Deposit--The areal distribution, thickness, consolidation, and degree of structural deformation all help to define minable ore tonnage reserves.[7,8] For most prospective deposits, these reserves must be sufficient to permit a high-volume annual production rate to hold ore recovery costs low, and also to sustain this annual output rate over a reasonable period of time (10 years, or more) to amortize the total costs of the mine, plant, and equipment. A minimum reserve recommended by Everhart[5] is in the range of 50 to 100 million st of recoverable ore. The most favorable ore structures for economical, high-volume recovery are shallow-buried, thick, underformed phosphate beds having a uniform composition and friable texture.

Overburden-To-Ore Ratio--The relative thicknesses of overburden waste and ore beds is probably the most flexible technical parameter in determining economic minability. Under the current economic restraints, however, the technical options of mining strategy are principally those of open-cast methods for ore removal.

Inflationary trends in the cost of labor, mine equipment, and energy now virtually preclude economical underground mining methods, except for a few instances where extremely favorable conditions exist, or strip-mine recovery is impossible.[5,10]

In open-pit mining, the ratio of overburden to ore-bed thickness largely determines whether mine operations can achieve the high-volume scale of ore recovery at a low unit cost to be economical. For any given deposit, the upper limit of this overburden ratio is related to the geological setting and depends also on the ore grade.

Under the most favorable geological conditions of deposit uniformity and high grade ore, the practical upper limit is a ratio of overburden to phosphate bed thickness of about 3, or in terms of overall recovery, roughly 20 cubic yards of material mined per ton of final product concentration.[7] When additional waste material is intercalated with the phosphate ore, the added costs of more selective ore recovery becomes a factor.

Neither limit is absolute, nor rigidly observed in all mining operations. The comparison of several major mining operations in Table I illustrates the limits and flexibility in overburden ratios in relation to ore quality, recovery efficiency, and upgrading requirements. The minimum ore bed thickness of one meter is considered minable under the most favorable conditions.[7]

TABLE I

Mining Characteristics of Some Major Phosphate Deposits[a]

Phosphate deposit	Av. thickness, m.		P_2O_5 grade, %		Concn. ratio	Recovery effic., %
	Overburden	Ore bed	Ore	Product		
Florida (pebble)	4.6	9.1	10-15	33	2.2-3.3	60-70
North Carolina	27.4	12.2	15	30-32	2.0	85
Western U.S.[b]	(variable)	3-9	26-36	32-36	1.0-1.2	90
Angola	2-3	10-40	34-37	34-37	1	100[d]
Morocco[c]	9-10	2	33-34	35-37	1	100[d]
Spanish Sahara[e]	10-30	4	31-33	34-37	1.0-1.2	(?)

[a]Data source (3), with revisions and additions.
[b]High-grade ore for acidulation use (2).
[c]Khouribga open-pit mine; British Sulphur Corp. data (22)
[d]No beneficiation requirements, except drying.
[e]Reference (18).

Mining Methods--With few exceptions, open-cast mining in its various forms (strip, pit, bench, or trench methods) is now considered to be the only economically feasible approach by most authorities.[2, 3, 5, 6, 7, 10]

In practice, however, open-pit mining is economical only if geared to high volume-low unit cost recovery of phosphate ore, so as to hold direct mining costs to about $1.00/long ton (lt) of phosphate ore. This may rise to the upper limit of about $2.00 when other operating factors are highly favorable in relation to total production costs.[5]

The scale of mine output has had to increase steadily to economize on rising costs of labor, material, energy, and plant overhead. This requires large capacity bucket excavators, or jumbo draglines--commonly of 50-70 cubic yard (yd^3) capacity--and large-volume hauling equipment to provide an ore feed to beneficiation mills of about 800 lt/hour (hr).[2, 3, 5, 6]

Quite recent mining developments at the Spanish Sahara phosphate deposit demonstrate these economies of scale. Draglines of 60 yd^3 capacity are used to remove waste and recover ore, which is hauled in 100-metric ton (mt) capacity trucks to the mill in quantities to sustain a 680 mt/hr mill rate. This current mining capacity of 3.3 million mt/year (yr) will increase to 10 million mt/yr at full capacity. These economies of scale, along with low beneficiation requirements, compensate for the unusually high overburden ratio (Table I), and the further costs of drilling and blasting to facilitate overburden removal.[18]

Ore Quality and Grade--Specifications of ore grade and quality for economical mining differ, depending upon the type and composition of the phosphate ore. Cathcart's detailed classification of ore types and their minimum grade limits still broadly apply, with few modifications.[7] As his discussion emphasizes, these minability limits are fixed mainly by their beneficiation requirements, rather than by technical limitations in their mine recovery.

With recent advances in mining and upgrading practices, minimum grade limits have decreased slowly. A minimum grade of 10% P_2O_5 for sedimentary ores is now widely accepted, but applicable only if the added mining costs for such lean ores are offset by savings in other phases of production.

Concentration Ratios and Recovery Efficiency--A determination of minability in an economic sense must include the milling requirements of mined ore, and the overall recovery efficiency of P_2O_5. While these factors are discussed in a later section, it suffices here to say that few, if any, commercial

mines operate at ore concentration ratios above about 3.5, and with recovery efficiencies of P_2O_5 in final concentrates below about 60%.

Mine to Mill Transportation--Economical mining requires short haul transportation accesses to the mine site and from mine to beneficiation mill. The geographical settings of some deposits--for example, the Western U.S. reserves dispersed over four mountainous states--incur substantial costs for ore haulage as a direct cost of mining.[4]

Heavy-duty, large capacity trucks, belt conveyors, and hydraulic transport help to minimize mine to mill transportation charges. Hydraulic transport, as widely practiced in the Florida phosphate mines, allows for cost recovery as an extension of the beneficiation process, since it delivers process water, and initiates in transit the disaggregation of clay slimes and mineral gangue from the pelletal phosphate.

With mine and mill capacities necessarily geared to high production rates (700-800 lt/hr) for economic production, the connecting transportation link must match this capacity. For these ore tonnages, overland transportation costs escalate rapidly as mine to mill distance increases. This one factor can defeat economic exploitation of high-grade reserves.

Average Mining Cost -- Fact or Fiction?

Nowhere in the extensive technical literature sources available to TVA is there any factual cost analyses of direct mining charges for current ore recovery methods. Nor does one find references to "average unit-cost" of mining phosphate ore, other than admittedly crude estimates based either on fragmentary data, or an averaging over a broad cost range. Neither basis has much statistical validity or practical value as a firm economic guideline.

In a recent U.S. Bureau of Mines survey, Service[10] also comments that comparative data on direct mining and related costs are not available, except for inferences drawn by evaluating other phases of the total operation for various commercial mines.

In fact, he comments further that no real significance could be attached to "average mining cost," since direct costs of ore recovery vary over wide limits from mine to mine, citing the interrelated factors that have been discussed previously.

Some comparative mining costs are reported by Ruhlman.[2] Although out of date, they provide one of the few examples of published cost figures. A summarized version of his data is shown in Table II.

TABLE II

Phosphate deposit	Mining method	Av. mining cost, U.S. $/lt[a]		
		Overburden removal	Mining and washing	Flotation cost
Florida Pebble	Strip	0.30	1.20	1.55
Tennessee Brown	Strip	0.40	3.75	
Idaho high-grade	Strip	0.50	3.25	
Idaho high-grade	Underground		4.25	

[a]1958 Bureau of Mines data from Ruhlman (2).

In his 1967 survey, Service[10] states that direct, or even average, mining costs of surface mining are difficult to determine with any reliability because of the wide variations among deposits. He suggested an estimated average cost per long ton of phosphate rock delivered to the mill at about $1.26. This value did not include the *pro rata* surcharges for land royalties, site preparation, roads, maintenance, and long distance haulage. The normal cost of stripping overburden was said to range between $0.30 and $0.50/yd^3 (~2.1 mt based on a bulk density of about 2.8). Stripping costs appear to have increased little with time, mainly because the scale of mine production, and the capacity of stripping equipment, has risen steadily to hold overburden removal costs to this tolerance range. Everhart's more recent (1971) estimate of direct mining costs of $1.00 to $2.00/lt of ore recovered[5] brackets the estimate of $1.26 given by Service in 1967.

Underground mining costs, by comparison, are stated to be in the range of $4.00 to $5.00 for Western U.S. (Idaho) high-grade phosphate, which constitutes the only domestic example. Mine recovery costs vary with bed thickness and dip. About two-thirds of the stated price is direct mining cost; the remainder applies to surface transportation, mill costs, and fixed overhead charges (see tabulation, page 54 of reference 10).

In view of the widely differing mine situations reflected by the six major economic deposits used as illustrations in

Table I, there can be little doubt that "average mining costs," as well as "average milling costs," have little practical significance or value as an economic guide.

Among commercial mines, each of these direct costs may vary over a rather wide range, but for any given mine operation, any increase in either of these direct mining costs above the "average" must be counterbalanced by "below average" costs in other phases of the total mine-plant operation if the deposit is economically competitive, and is to remain so.

Cost guidelines, therefore, have little practical significance unless they are paired with the particular set of unique technical parameters for each phosphate deposit.

III: BENEFICIATION--TECHNICAL AND ECONOMIC FACTORS

A wide range of technical methods is employed to upgrade phosphate ores to marketable concentrates, but the beneficiation requirements for any given mine operation are largely determined by ore characteristics and end-use of the concentrates. The limits of commercial beneficiation are restricted by economics rather than technical know-how. Satisfactory upgrading methods exist for virtually every known ore type, but for some, the costs may be too prohibitive to consider further development plans.

Since no standard scheme of beneficiation applies to all phosphate ores, there can be no meaningful average cost for beneficiation and its component steps, or its *pro rata* share of total production costs.

Direct costs vary with the combination of methods required by specific ore characteristics (hardness, texture, mineral composition), the required enrichment ratio, and overall recovery ratio.

Commercial Methods of Beneficiation

The numerous upgrading procedures that are in commercial use have been extensively described elsewhere (2,3,4,5,6,7,9,10, and appended reference list), and are summarized briefly here for discussion purposes.

Physical Methods--The simple procedures of grinding, wet or dry screening, desliming, washing, and air classification are the least costly and most widely used methods. Wet classifica-

tion schemes based on gravity separation (belt, table, and spiral classifiers) or flotation, when required as supplemental steps, add to the complexity and cost of plant operations.

When ores require high concentration ratios of 2 to 3, economics dictate a choice of the simplest procedures yielding the highest recovery ratios--usually no more than grinding, screen sizing, and washing. The more costly methods of flotation, wet classification, and calcination require offsetting savings in mining, transportation, and ore quality.

Thermal Methods--Calcination is not broadly applicable as an upgrading scheme, but serves as a remedial treatment for three main ore categories: (1) Problem ores having excessively high organic matter; (2) ores that contain highly substituted apatites which can be altered thermally to reach marketable grade; and (3) carbonate ores that require thormal decomposition for wet-chemical separation of phosphate minerals.

Despite these potential technical applications, fuel costs of calcining largely restrict its use primarily to the first category where no other technical option presently exists that offers a more economical solution.

Chemical Beneficiation--With the shift in market end-uses to acidulation processes, the direct extraction of P_2O_5 from mined phosphate ore, using H_2SO_4 or HNO_3, is becoming attractive for certain ore types. Low-grade siliceous ores, which are frequently difficult to upgrade, may not require physical beneficiation if favorably priced to compensate for the higher handling costs of raw material in the acidulation processes. The possible cost advantages of chemical versus physical concentration of P_2O_5 will pertain exclusively to captive production, where transportation costs are mineral.

Character of Ore Affects Beneficiation

Sedimentary phosphate ores show a wide diversity in their gangue-mineral compositions, but generally fall into one of three principal categories, based on the major diluent phase.[7]

Siliceous Ores--These contain quartz, chalcedony, or opaline forms of silica, and differ markedly in beneficiation requirements. Ore types that contain mainly quartz are usually amenable to simple upgrading methods, and even lean ore feeds can be beneficiated economically by flotation and gravity methods.

Ores that contain chalcedony and opal tend to be hard, dense, indurated rock due to silica cementing. These require careful grinding to liberate phosphate, but enrichment ratios are often poor due to incomplete liberation, and recovery efficiency is reduced by slime losses of the softer phosphate.

Carbonate Ores--Included here are the phosphatic limestones, marine phosphorites containing detrital carbonates, and calcareous phosphates in which calcite, dolomite, or ankerite is intimately admixed with the phosphate, or occurs as intercalated seams.

The hard cemented types, such as the dolomitic ores of Rajasthan, the calcitic ores of Australia, and ankerite types in Iran, make it very difficult to liberate the softer phosphate in recoverable sizes, using physical methods.

In both soft and hard types of ore it is difficult to remove the carbonate minerals efficiently by commercial flotation processes.[7,9]

Calcining followed by wet-chemical separation (slaking) is technically feasible, but too expensive in most cases where ores contain less than about 25% P_2O_5. Commercial use of this upgrading procedure requires very high separation efficiency to reduce the $CaO:P_2O_5$ ratio in the product below the 1.6 limit. Furthermore, other quality factors of the ore, especially its magnesium content, must be highly favorable if the product concentrates are to be used for wet-process acid.

Clayed Phosphates--Many sedimentary deposits contain mainly clays and hydrous iron and aluminum oxides as diluents, where these gangue minerals are concentrated mainly in the silt and clay size ranges. Quartz is a common accessory mineral. These are amenable to release and removal by simple beneficiation steps such as scouring, blunging, followed by washing and screening with the aid of dispersing agents. Dry classification using fluidized bed techniques may also be effective and is gaining in popularity.

Some Examples of Beneficiation Behavior

McBride[3] summarized the beneficiation behavior of domestic (U.S.) ores to illustrate their widely differing requirements. Florida ores, which contain mainly quartz and clay impurities, require enrichment ratios up to 3, but with a recovery efficiency of only 60% to 70%.[4] Ruhlman[2] emphasizes the complexity of the upgrading steps involving simple methods, wet classification methods, and even calcination that is required to con-

centrate 10%-15% P_2O_5-grade Florida ore to 32%-37% commercial concentrates.

North Carolina ores, although of similar type and grade, yield lower concentration ratios of about 2, but with recovery ratios near 85%. Similar ores in Western U.S. have even lower concentration ratios (1.1 to 1.5), with recovery ratios near 90%. Low-temperature calcination is an added requirement to remove organic matter.

Thus, all three domestic ore types differ in beneficiation requirements and overall recovery of P_2O_5, despite their similarities in gangue-mineral components.

Assessment of Beneficiation Costs

The extensive literature sources available to TVA failed to disclose either comparative cost data for individual beneficiation methods, or direct costs of ore concentration in relation to total production costs. Such data as exist are of a proprietary nature, and then applicable to a particular mining situation. Other reports and surveys draw similar conclusions.[2,5,7,9, 10]

An earlier comparison of production costs by Ruhlman[2] for strip-mine production of Florida, Tennessee, and Idaho phosphates showed the following breakdown of costs per long ton of marketable concentrate. Overburden removal ranged from 30 to 50 cents; mining and milling (grinding, washing) costs varied between $1.20 and $3.75. The singular example of flotation costs (Florida) was $1.55. Fixed plant operation and overhead costs, plus drying, shipping, and depreciation costs were quite similar for all three deposits ($1.75 to $1.95), bringing total production costs to between $5.00 and $6.25/lt of product. While these costs no longer apply to current practice, they illustrate the flexible nature of combined mining and milling costs, depending upon ore character.

In a recent and more comprehensive survey, Service[10] flatly states that direct mining and related costs of production (beneficiation and plant operating costs) are not available, but he drew inferences of their veriability by comparing the average value of mine products for Idaho ($5.76), other Western U.S. deposits ($8.53), Florida ($7.01), and Tennessee ($7.64).

The allowable costs for upgrading are bracketed by the direct mining costs plus fixed plant operating costs, but this cost distribution differs for each mining operation. In general, however, the cost aloowance for upgrading steps restricts the

choice to the simplest steps (grinding, screen sizing, washing). The more expensive wet-classification procedures or calcination require offsetting savings from lower mining and transportation costs, and high quality ore feed.

Flotation processes, while similar in principle, bear only a superficial resemblance in actual practice. They differ in degree of complexity and recovery efficiency, depending upon ore characteristics and grade goals set by the producer. Hence, no fixed cost assessment is feasible.

Even individual producers must periodically adjust the sequence of beneficiation steps, including flotation cycles, to accommodate intermittent shifts in ore feed arriving from several mine pits, so that *pro rata* costs of beneficiation in relation to total production costs are time-averaged overall values. This makes cost allocations to specific beneficiation steps difficult, even if the proprietary data were to be made available.

Beneficiation steps based on thermal treatment are more easily defined. Most sedimentary ores require drying, where fuel costs are the major factor, and fuel requirements depend on moisture content and particle size. Florida concentrates, for example, have typical moisture contents in the range of 8%-15%, and require from 3 to 10 gallons/lt of fuel oil to reduce the moisture content to about 1% for shipping. Drying costs are highly sensitive to escalating fuel costs, therefore.

The costs of calcination depend on the objectives of this beneficiation process. For full calcination treatment at about 800°-850° to increase BPL grade and improve other quality factors, about 2.5 to 3 million Btu/lt of concentrate is required. With current fuel costs at roughly 50 cents/million Btu, the direct cost of calcination may range from, say, 75 cents to $1.50, depending upon the Btu fuel credit for organic matter in the calcine feed.

Calcination costs are proportionately lower if carried out at about 450°-500°C merely to destroy objectionable organic matter.

On the other hand, thermal treatment for the purpose of decomposing carbonate type ores usually requires heating for longer periods at higher temperatures (900°-1000°C), with a corresponding increase in fuel requirements. This one cost factor has been the major economic obstacle to exploitation of large high-grade reserves of carbonate-type ores located throughout the world. Their prospects for exploitation grow less attractive as fuel values continue to increase relative to those of market-grade phosphate.

In summary, the shortcoming of the formal literature to supply reliable cost prorations for specific beneficiation processes is not to deny that specific examples are available, perhaps, from other sources. The inherent risk of using such cost data, however, is to make unwarranted broad comparisons or to strictly apply it as an economic basis to a quite unrelated phosphate mining situation.

IV: QUALITY FACTORS OF CONCENTRATES

Concentrate quality is having an increasingly important impact on market value. As just discussed, the singular objective of nearly all beneficiation processes relates to improving the P_2O_5 quality factor. Any improvement in other chemical or physical quality factors is more or less incidental, depending upon the specific minerals that end up in the rejected gangue-mineral fraction.

The main parameters that are nearly always considered are enrichment ratios, recovery efficiency, and BPL grade. Concern by mine producers over undesired impurities such as Al, Fe, and Mg is mitigated by the lack of market specifications, or penalties other than that imposed by the more discriminate buyers.

Today's more sophisticated fertilizer processes in worldwide use require specific types of raw materials. All sources of commercial concentrates are not mutually suitable for a particular process, nor is one particular concentrate broadly suited for all manufacturing markets. Thus, sellers and buyers differ on the market value of concentrates priced on a P_2O_5 unit basis.

Each of the major end-use markets--wet process acid (WPA), superphosphoric acid (SPA), ammonium phosphates (AP), superphosphates (OSP, CSP), and nitric phosphates (NP)--have preferred characteristics for their phosphate raw materials.

Tha actual economic value of a mine concentrate is becoming increasingly dependent upon its technically feasible end-uses, and buyer preference for some particular set of quality factors,[3, 4, 5, 6, 7] and not merely its P_2O_5 content.

For superphosphates, the long-established criterion is high P_2O_5 content, rather than other chemical factors, to produce marketable products.[4] Similarly, electric-furnace producers are much less concerned with chemical quality than with energy costs.[6]

In acidulation processes, however, a host of quality factors--both physical and chemical--become highly significant, and these are summarized in the following section.

Physical Quality Factors

Phosphate concentrates that consist of hard, coarsely textured particles, or highly crystalline apatite (Kola, Phalaborwa, Jhamar Kotra types) are often too unreactive for WPA, and especially so for superphosphates. These types of concentrates may require additional grinding by the manufacturer to some acceptable particle size for the particular acidulation steps.

Calcined grades of concentrates may also have low reactivity, due to agglomeration, loss of porosity, and thermally induced crystal growth of the apatite mineral. All tend to drastically reduce surface area.

In general, the preferred textures of concentrates for acidulation processes are soft, porous, pelletal or sand-sized phosphate particles that are easily handled and metered. Grinding of such phosphates to the pulverulent forms required for superphosphates presents no major problem. These desired textural properties are common to many sedimentary deposits, but not to igneous or metamorphic phosphate ores.

Chemical Quality Factors

(1) Iron and Aluminum (R_2O_3)--These two impurities are particularly troublesome in WPA intermediates and their ammoniation products (AP). They are mainly responsible for sludges and postprecipitation in WPA, scale formation in SPA production, insoluble phosphate compounds in liquid AP products, and unwanted agglomeration in nongranular solid AP products.[19] Phosphate concentrates having an R_2O_3 content above 3%-4% are unattractive for WPA use, but not for superphosphate processes.[5]

(2) Magnesium--The detrimental effects of Mg in WPA processes and in ammoniation products are now widely recognized, but costly to overcome. Magnesium occurs in concentrates mainly as residual accessory carbonates (Mg-calcite, dolomite, ankerite), or as a constituent element in the apatitic phosphate mineral.[13, 14, 15] Usually the latter source contributes most, if not all, of the Mg and cannot be corrected by physical beneficiation. The amount of Mg in apatites from different deposit sources varies widely, ranging from about 0.05% to 0.7%

MgO;[14, 15] sodium and fluorine show a corresponding increase (see discussion in Part II).

In WPA, magnesium precipitates F in the reactor stage as colloidal phases that blind the gypsum filters. It raises the viscosity of SPA products, and it is the primary cause of insoluble phosphate precipitates in AP liquid fertilizers.[19]

(3) Fluorine and Si/F Ratio--Concentrates high in F tend to yield WPA high in F. The presence of reactive silica aids in removing F, but is not the exclusive control, even when Si/F ratios appear satisfactory. The F content of WPA has a major influence on the kinds and amounts of insoluble precipitates that form in its ammoniation products. The concentration of residual F in liquid fertilizer grades is especially critical.[20]

(4) Chlorine--The effect of soluble Cl is its corrosive action on plant equipment, rather than a chemical interference in manufacturing process. The usual sources in phosphate concentrates are the evaporite salts common to sedimentary ore beds, or from sea water used in beneficiation steps. In either case, the corrective treatment is fresh-water washing. Generally, Cl content higher than 0.1%-0.2% cannot be tolerated because corrosion rates become too severe.[5]

(5) Minor and Trace Elements--Minor constituents such as Mn, Fe, Zn, and Cu are beneficial as micronutrients, but all contribute to post-precipitation of insoluble phosphate.[19]

Other elements such as Cd, Pb, Cr, As, Hg, Se, and V are either toxic, or potentially harmful in agricultural products. There are no rigid specifications, but concentration limits have been tabulated.[5]

Recent interest has been shown in the radionucleides of U and Ra, both from a recovery viewpoint, as well as potential dust hazards around manufacturing areas.

(6) Organic Matter--With increasing amounts of organic matter in sedimentary ore concentrates, the foaming problem in WPA processes adds to reactor equipment costs, and discoloration of products reduces their market value and sales appeal. Some high-grade rocks require additional costly calcination treatment merely to remedy this quality factor.

(7) CaO/P_2O_5 Wt. Ratio--This quality factor mainly affects process economics. Above a value of 1.6, sulfuric acid requirements become uneconomical.[5,7]

Since the CaO/P_2O_5 ratio of the apatite mineral ranges up to 1.61,[14] a total removal of all other calcium-bearing min-

erals by appropriate beneficiation may be required. This may not be technically or economically feasible, particularly with carbonate type phosphate ores.

For captive use, the additional costs for H_2SO_4 reagent may be weighed against added beneficiation costs to find the most economical route. The additional beneficiation costs may be unavoidable, however, if concentrate products are to compete in open markets.

This one quality factor may rank above all others in the evaluation of carbonate-type phosphate deposits.

Combined Effects of Quality Factors

Bradley[4] summarizes the combined effects of quality factors on end uses of mine concentrates, using the major U.S. phosphate deposits as examples.

Florida concentrates, because of their high grade (32%-37% P_2O_5) and quality, find major use in WPA manufacture. The main detrimental factors are R_2O_3 and organic matter contents.

North Carolina concentrates have lower P_2O_5 content (30%-32% P_2O_5), but have excellent physical quality and high chemical reactivity. Their major end uses are WPA, SPA, and AP. The primary causes of processing difficulties are high contents of Mg, Na, F, and organic matter. Calcination to remove organics is often necessary to maintain product quality.

Western U.S. phosphate production is divided about equally between WPA and electric-furnace end uses, largely on the basis of chemical quality. Even in the higher grade concentrates (~32%-P_2O_5) reserved for WPA use, problems arise from R_2O_3, Mg, minor elements (V, Cr, Cd, As), and especially their high organic-matter content; the latter often requires the additional beneficiation cost of calcination.

Tennessee phosphates are now largely restricted to electric-furnace use because of their declining P_2O_5 grade and their broad range of adverse chemical quality factors for WPA use. Their physical qualities require an agglomeration step to make acceptable furnace burden, and minimum grade requirements (24% P_2O_5) are met by blending ore grades or concentrates.

Future Impact of Quality Factors

A growing awareness by fertilizer manufacturers over these other chemical qualities of concentrates, aside from P_2O_5, is a recent development arising from the major shift to WPA and SPA technology. The detrimental effects of certain chemical factors in WPA processes--in contrast to superphosphate processes--are becoming widely recognized.

This is having an obvious effect on the relative market value of phosphate concentrates that otherwise appear competitive on a BPL grade basis. As St. Guilhem[6] points out, there are no immediate prospects for industry-wide specifications on these other quality factors, but nevertheless, it is becoming imperative to consider this marketing parameter in any evaluation process of phosphate reserves.

V: TRANSPORTATION FACTORS

General Considerations

Transportation factors that contribute to higher selling costs of mine concentrate include (1) ore movement from mine to beneficiation plant, (2) transport of concentrate products to domestic markets or seaport terminals, (3) ocean transport to export markets or distant domestic markets, and also (4) transport of inflowing mine supplies, plant equipment, fuel, and related material needs to the deposit site.

These aggregate transportation costs impose such rigid limitations on profitable marketing spheres that most authorities consider the collective transportation problems as an initial step in the evaluation of phosphate reserves.[4, 5, 7, 9, 10] Cathcart[7] stresses that it may be the single overriding factor, despite other favorable aspects of the deposit.

The reason that freight allowances become such a major cost factor rests on the law unit-value of concentrated phosphate raw materials in comparison to most other metallurgical ores. Everhart[5] illustrated this by comparing the 1971 f.o.b. price of about $6.00/lt for 72 BPL Florida concentrate to the local market prices of $5.00 for sand and $8.00 for construction gravel. Few, if any, metallurgical raw materials for industry approach the bargain price of phosphate valued at about 2 cents/kilogram (kg) P_2O_5.

Deposit Location Relative to Markets

Remoteness to domestic production centers or seaport terminals may readily add surface transportation surcharges equal to, or exceeding, the total mine production cost. Similarly, deposits close to seaport outlets, but remote to world export markets, may incur shipping charges that grossly exceed the f.o.b. market value of mine products. Some coastal deposits may remain uncompetitive, therefore, in current world markets.

To point up the key importance of transportation factors, Everhart[5] makes the interesting observation that virtually all commercial deposits competing successfully in world markets today are found located within 100 miles of deepwater seaport facilities.

Compensating Factors

If interior phosphate deposits present extremely favorable mining conditions--high grade ore, low mining costs, and requiring little or no beneficiation--then higher shipping cost allowances are permitted, which extends the marketing radius.[2] The magnitude of reduced production costs relative to railroad shipping rates, however, would allow a gain in shipping distance of perhaps no more than 100 miles. The selection of overland belt conveyance at Bu Craa, Spanish Sahara, is expected to become more economical than rail transport as ore output increases from the initial 3.3 million mt/yr.[17]

Another approach to economizing direct production costs arises from the shifting emphasis to wet-process acid manufacturing, where grade requirements are lower. The additional costs of beneficiating to some long prevailing BPL level may not be offset by transportation saving or economies in wet-process acid production. With the shifting emphasis to phosphoric acid intermediates, the high BPL market grades established formerly for superphosphates no longer rigidly apply. For some phosphate rocks, no technical benefit arises from their high state of beneficiation.

To cite an extreme example, some Colombia, S.A., rocks with P_2O_5 contents of only 22%-24% have yielded a good quality wet-process acid without any prior upgrading (TVA unpublished results). No standard grade requirement applies to phosphate rocks supplied to wet-process acid markets.

The net reduction in rock beneficiation costs that can result by not exceeding technical grade specifications may extend the profitable market range of product concentrates.

Still another recent trend is the shift of fertilizer production facilities closer to the phosphate resources to save transportation costs. The degree to which interior phosphate deposits will benefit from this movement has practical limits, since a compromise also must be reached in minimizing transportation costs of other fertilizer raw materials (sulfur, ammonia, natural gas). A unique combination of these transportation factors will exist for each geographical deposit, however.

Examples of Transportation Costs

Our domestic (U.S.A.) rail-freight rates, for example, are not computed simply on a ton-mile basis, but vary according to hauling distance, volume shipped, value of commodity, and method of rail conveyance.

Because of this, the commercial freight rates for single carload units (40-50 st) of 70 BPL concentrate shipped from central Florida mines to TVA's plant site (750 miles) would be $9.11/st, which adds about 150% to the f.o.b. sale cost of the mine concentrate.

Yet the same product can be shipped three times as far to southeastern Canadian markets at comparable cost if moved in unit-train volume.

Since freight rates increase in proportion to the unit value of material shipped, the benefits from captive use of mine product in nearby production facilities is offset to a considerable degree by the much higher freight rates levied on manufactured fertilizer products.

Whether or not a phosphate reserve is profitably located in relation to domestic production and end-use markets, is determined to a large degree by the prevailing rail freight rate structures. Some examples of typical rail freight rates and ocean shipping costs are compared in Table III.

TABLE III
Comparable Transportation Costs
(Fla. 70/72 BPL Concentrate)

Destination	Shipping mode	Miles	Cost cents/st mile
Japan	via ocean/backhaul	10,000	0.085
California, U.S.A.	via coastal shipping	4,000	0.15
S.E. Canada	rail (unit-train)	2,000	0.45
Alabama, U.S.A. (TVA)	rail (carload unit)	750	1.21

The lower rates for ocean shipping offer distinct advantages to phosphate reserves located near seaport terminals, not only for export markets, but quite possibly domestic markets as well.

As Table III shows, the same Central Florida mine product can be shipped 4000 miles by sea to U.S. west coast production markets for about $4.50/st,[4] or roughly half the cost of rail shipment 750 miles inland to TVA's plant in North Alabama. In fact, rail transportation charges effectively preclude competition by Western U.S. phosphate mines for Pacific coast markets, although they are separated by only 500-600 miles.

The effect of transportation factors on market outlets is best illustrated in Table III by comparing the shipping costs of Florida concentrate to Japan. Shipping costs for the 10,000-mile haul to Japanese markets were quoted in the range of $8.50/st ($24.00/st P_2O_5) in 1972. This is slightly less than the commercial rail freight rate for transporting the same rock about 750 miles inland, say, to TVA's plant site.

Transportation factors thus place seaboard phosphate deposits in a favorable position to reach quite distant export markets, while restricting their domestic market sphere to nearby production markets, as is illustrated by our Florida and North Carolina phosphate reserves.

The same transportation factors deny export markets to remote interior deposits, thereby restricting their use to local captive production or regional markets. The economics of transportation with respect to the extensive Western U.S. phosphate reserves that are scattered over four interior mountain states have been described in considerable detail.[3,9, 10]

Similarly, transportation has been a crucial problem in consummating exploitation plans for several major new phosphate reserves in Africa, South America, and Australia, so that their marketing potential still remains to be resolved. Detailed reports of their individual case histories have now been published (see appended bibliography).

VI: MARKET FACTORS

Market Incentives

Earlier forecasts in the mid-1960's that world phosphate output would continue to expand at an annual rate of 10% to meet expanding food needs[11] have been revised downward sharply by the oversupply in the late 1960's.

According to a recent comprehensive analysis of consumption trends, food demands, and marketing factors in both developed and developing nations, Harre, et al.,[12] projects a 4% to 5% annual increase in phosphate fertilizer consumption through 1980.

TABLE IV

World Phosphate Supply and Demand
(million mt P_2O_5)

	Phosphate production		Phosphate consumption		
Year	Tons	Annual growth rate, %	Tons	Annual growth rate, %	S/D ratio
1965	14.7	~10	13.7	~10	1.07
1968	18.4	5.7	17.1	7.5	1.08
1970	19.2	1.1	18.6	3.9	1.03
1973[a]	22.6	5.1	22.3	4.6	1.01
1975[a]	24.9	4.9	24.3	3.9	1.03
1980[a]	26.5	(?)	29.3	4.4	0.91

[a]Preliminary revision on basis of higher than anticipated utilization of production capacity (Harre, private communication).

Kahnert[17] forecasted much higher phosphate fertilizer production by 1980 in the range of 32.8 to 39.5 million mt P_2O_5, based primarily on a crude extrapolation of production trends from the 1960's. The current and planned worldwide expansion of rock output[16] may upset the present supply-demand balance through 1980, if we adopt Harre's more conservative estimates based on actual fertilizer consumption needs.

A larger influx of phosphate raw material may overstimulate new production, as well as plant operating levels beyond the projected 90% of capacity for developed countries, and 80% for developing countries by the mid-1970's. Harre, et al.,[12] cautions that such a worldwide acceleration in plant operating levels alone can create a large oversupply--especially so if the agricultural forecasts of demand and consumption prove overly optimistic.

With the prospect that several large, newly developed phosphate reserves (S. Sahara, Angola, Queensland, Sechura) may soon enter the supply picture, coupled with stimulated output

from existing major sources, the potential for recreating market conditions of oversupply and buyer discrimination already exists.

Thus, market incentive would appear to offer little stimulus to exploitation efforts on undeveloped phosphate reserves or new discoveries. Imported rock will likely continue to be the more economical source of raw materials for developing nations. Despite this, the pace of phosphate resource development no doubt will continue to be spurred by other incentives and national goals. If pursued vigorously, the long term impact will be to contribute to a market oversupply.

Market Value of Products

Everhart[5] states a widely shared view that market prospects along with transportation factors be considered ahead of technical evaluation steps, since the entire evaluation process hinges on base-price structures of specific market outlets for a particular concentrate grade.

The estimated market value of product concentrates fixes cost limits and places economic restrictions on the technically feasible options of mining, and especially those of ore dressing, despite their relative technical merit.

While P_2O_5 content is the conventional sales basis in open markets, other quality factors of a concentrate (size, texture, impurities, organic matter) may restrict its market outlets to certain sectors of fertilizer technology, thereby affecting its competitive sales value.

For example, not all commercial concentrates are suitable for superphosphate manufacture. Some are suited only for electric furnace use and must compete for only 8% of the total world market. The effect of quality factors on market value of concentrates is also reflected in the growing trend of more discriminate selections of raw material by wet-process acid producers.

Thus the criteria for establishing product value are changing. Phosphorus content no longer is an absolute basis for economic evaluations, but merely fixes an upper limit in market value.

If a prospective phosphate product meets quality requirements to compete an open domestic and world markets, then

its unit value at point of shipment is pegged to prevailing price scales in free enterprise markets, based on P_2O_5 content.

In the United States, for example, export (f.o.b.) prices for commercial-grade concentrates are estimated at $7.00/st in 1974, as compared to $5.40 in 1973 and $5.00 in 1972.[1] Product value in domestic markets is not projected to increase as rapidly. The average selling cost in 1972 for the 43.8 million st sold or used in U.S. markets was $5.10/st (TVA survey). These market differentials must be taken into account when using the P_2O_5 basis for estimating the potential market value of product concentrates.

The P_2O_5 basis for economic evaluation also depends on the time frame. Historically, market values fluctuate sharply during periods of chronic shortage or oversupply. St. Guilhem[6] stresses the importance of this time factor when making economic assessments of mining and beneficiation costs in relation to some future state of the market.

A different base value applies to phosphate produced for a captive market. This outlet offers economic advantages through transportation savings and the higher sales value of P_2O_5 in more concentrated finished products. Higher cost allowances for mining and beneficiation are possible. The pricing structure for captive products thus differs from open-market values and also differs for each captive phosphate deposit.

The basis for economic evaluation of phosphate reserves is not restricted solely to free enterprise value scales. National interests may best be served by subsidizing mine development and production costs, either to acquire a stable captive supply of essential raw material for a fertilizer development program, or to gain on maintain a world trade position with an exportable mineral resource. National policy and objectives determine subsidy levels and the value of phosphate products.

The economic basis applies in varying degrees to recent developments of major new phosphate reserves in South America (Peru, Brazil, Colombia), Angola, and Australia. Their competitive position in open world markets is still unresolved, however.

A decision must be reached, therefore, early in the economic evaluation process as to marketing outlets--either captive, free-enterprise, or subsidized production--to fix the base value of beneficiated mine product. For this, P_2O_5 content is still the best available criterion, but with the recognition that it is no longer an absolute basis--only an approximation.

PART II:
TECHNICAL EVALUATION OF PHOSPHATE CONCENTRATES

As discussed in Part I, the physical and chemical quality factors of commercial concentrates have considerable importance in determining their actual market value. The methods and approach developed in TVA fundamental research for dealing with this evaluation problem are summarized very briefly here.

This technical evaluation of phosphate concentrates from the consumer's viewpoint provides comparisons of:

1. Mineral composition and the potential for further physical beneficiation.
2. The potential value, or need for calcining treatment.
3. Acidulation and reagent requirements.
4. Chemical quality factors that may affect processes or final products.

The primary distinction among concentrates from different deposit sources is that the compositions of their apatite minerals differ markedly. The important initial step, therefore, is to establish the precise chemical form of the apatite component as the basis for evaluating the four objectives stated above.

In TVA's approach, apatite compositions can be rapidly and precisely determined by an x-ray diffraction procedure, using simple statistical models based on crystal-chemical studies that now extend to over 500 reference samples from worldwide phosphate deposits. Details of the x-ray diffraction method and its applications have been published elsewhere.[13, 14, 15]

Compositions of Sedimentary Apatites

The apatite mineral component is usually the singular source of phosphorus and fluorine and frequently contributes most, if not all, of the magnesium and sodium impurities.

Sedimentary apatites have been shown to have compositions that can be represented quite adequately by the series with end-member empirical formulas of:[14, 15]

$$Ca_{10}(PO_4)_6F_2 \text{ —— } Ca_{10-a-b}Na_aMg_b(PO_4)_{6-x}(CO_3)_xF_{0.4x}F_2$$

(fluorapatite) (francolite types)

These correspond to the chemical composition:

TABLE V

Constituent (%)	Fluorapatite: (x = o)	Francolites: $x/6 - x \cong 0.30^a$
CaO	55.6	55.1
P_2O_5	42.2	34.0
CO_2	0	6.3
F	3.77	5.04
Na_2O	0	1.4
MgO	0	0.7

[a]Maximum degree of substitution found and predicted for francolite-type apatites.

These apatite compositions can be derived by the linear-approximation statistical models shown below, which have been found to be quite satisfactory for routine analytical appraisals of concentrates.

Statistical models:

$$\underline{a}_{obs.}\ (\text{x-ray}) = 9.374 - 0.204\ (x/(6-x))$$

$$(Na)_a = 1.327\ (x/(6-x))$$

$$(Mg)_b = 0.515\ (x/(6-x))$$

In the above equation, the $\underline{a}_{obs}$ term is the crystallographic $\underline{a}$-axis dimension determined by the x-ray diffraction procedure.[14]

The following illustration of this analytical method shows typical apatite compositions in concentrates from six well-known geographical deposits. These examples are arranged in the order of increasing carbonate substitutions, but do not fully span the possible range of compositions given in Table V.

TABLE VI

Some Typical Apatite Compositions

Source	Composition, % of					
	CaO	MgO	Na_2O	P_2O_5	CO_2	F
Western, U.S.	55.6	0.13	0.26	40.1	1.59	4.09
Tennessee (U.S.)	55.5	0.24	0.47	38.7	2.71	4.31
Florida (U.S.)	55.5	0.36	0.72	37.1	3.95	4.56
Morocco	55.4	0.43	0.85	36.3	4.53	4.68
N. Carolina (U.S.)	55.3	0.52	1.04	35.3	5.36	4.85
Tunisia	55.2	0.60	1.20	34.7	5.70	4.93

The most significant features revealed by this comparison are the relative increases in amounts of F, Mg, Na, and CO_2, the decrease in P_2O_5 content, and the nearly constant CaO content, as the degree of substitution in the apatite increases. Note also that all these compositions contrast sharply with the fluorapatite composition that is commonly assumed for phosphate raw materials, or the composition inferred by the commonly used "BPL" basis.

Effect of Apatite Composition on Concentrates

Once established, these individual apatite compositions provide an immediate test of mineral quality of concentrates. This is illustrated below by applying the theoretical apatite compositions in Table VI to a corresponding series of hypothetical concentrates on an equivalent P_2O_5 grade basis.

TABLE VII

Mineral Composition of Concentrates[a]
70 BPL Grade (32% P_2O_5)

Deposite source	Weight % of: Apatite[b]	Gangue Minerals
Western U.S.	79.8	20.2
Tennessee (U.S.)	82.7	17.3
Florida (U.S.)	86.3	13.7
Morocco	88.2	11.8
N. Carolina (U.S.)	90.7	9.3
Tunisia	92.2	7.8

[a]Data source (13).
[b]Wt. % apatite = actual P_2O_5/theor. P_2O_5 (100% apatite).

This comparison, based solely on knowledge of apatite composition and P_2O_5 content, immediately reveals differing degrees of beneficiation with respect to the mineral impurities, and its implications regarding the chemical quality of concentrates. It can also be seen why acid requirements vary so widely for concentrates from various deposits. The illustrated examples all contain 32% P_2O_5, but acidulation requirements increase as the weight fraction of apatite increases, since their calcium contents have been shown to be nearly identical (Table V).

Further use can be made of these weight distribution data if the chemical analysis of the concentrate is extended to include other important constituents aside from P_2O_5. The primary objective here is to determine how certain key impurities (see discussion of quality factors, Part I) are distributed between the apatite and gangue-mineral fractions. From this, one can determine the potential benefits of further upgrading by physical methods to improve the chemical quality of the phosphate raw material.

This rapid approximation is illustrated below, using as an example the 73 BPL Florida concentrate shown in Table IX. A mean theoretical apatite composition for this source was derived by the previously described statistical approximations, using a typical value of $\underline{a}_{obs}$ (9.340 - 9.341), which also corresponds closely to the composition given in Table VI.

TABLE VIII

Modal Analysis of Chemical Constituents

Constituent	Florida rock example: Apatite (theor.)	Florida rock example: Concentrate (73 BPL)	Apatite[a] fraction (90 %)	Gangue[b] fraction (10.0 %)
CaO	55.51	48.9	49.9	0
P_2O_5	37.15	33.4	(33.4)	0
CO_2	3.95	3.0	3.6	0
F	4.56	3.9	4.1	0
Na_2O	0.72	0.53	0.65	0
MgO	0.36	0.29	0.32	0
R_2O_3	-	2.12	-	2.12
SiO_2	-	4.5	-	4.5

[a]Estimated amounts; based on 90% apatite derived from the P_2O_5 ratio (33.4/37.15).

[b]Remainder includes organic matter, heavy mineral fraction, and combined water not shown in listed analysis.

In the above example, all the carbonate, fluorine, and the undesired impurities, Na and Mg, can be reasonably assigned to the apatite. The iron, aluminum, and silica impurities (about 7.5 wt. % as clays, hydrous oxides, and quartz) demonstrate the potential limits of further beneficiation by physical methods. An analogous distribution of the F, Mg, and Na impurities mainly to the apatite component applies to many other sources of commercial concentrates, so that their removal by physical methods is not usually possible.

Evaluation of Some Commercial Rocks

This evaluation scheme has been applied to a series of 14 phosphate rock concentrates to demonstrate how their physical and chemical quality factors, beneficiation potential, and acidulation behavior can be rapidly estimated. The 14 samples were selected to represent both well-known sources and some more recently developed deposits. All are authentic products of beneficiated ore, but not necessarily representative of current market grades. Strict comparisons based on the examples reported here, therefore, may not be valid and this use of the data is not recommended.

Comparison of Chemical Quality Factors

The compositions of the 14 phosphate rocks, which were obtained by chemical analysis (TVA data), are compared in Table IX. For simplicity, only the important constituents discussed in Part I are included here.

These analyses (Table IX) immediately reveal the chemical quality of individual raw materials with respect to P_2O_5 content and their contents of undesired impurities.

However, such comparisons based on chemical composition give no direct indication of their potential for further upgrading to improve their chemical quality and acidulation requirements.

State of Physical Beneficiation

Concentrates differ in their degree of beneficiation, and their potential for further upgrading by physical or thermal methods to improve their quality, as is illustrated by the 14 examples in Table X.

The basis for making this evaluation follows the illustrations described in Tables VI and VII and proceeds through the following steps. By substituting the appropriate x-ray $\underline{a}_{obs.}$ value in the statistical equations, the empirical formula of the particular apatite can be obtained, and its chemical composition then computed on the usual weight percent basis.

Since the theoretical P_2O_5 content thus obtained represents the maximum limit of physical beneficiation (100% apatite), then the ratio of actual P_2O_5 content to this theoretical P_2O_5 value establishes the respective weight fractions of apatite and gangue minerals in the concentrate. The results obtained in this manner for the 14 examples of concentrates are those shown in Table X.

Note that the gangue-mineral fraction ranges from 2% to 19%, indicating their respective potential for further physical upgrading.

The above procedure requires only a P_2O_5 analysis. If more complete analyses of concentrates are available (as given in Table IX), then the approximate distribution of these constituents between the apatite and gangue minerals can be made. This is illustrated in Table VIII, using the Central Florida sample described in Tables IX and X; similar analyses of the other 13 samples are omitted here.

TABLE IX

Chemical Composition of Concentrates From Commercial Deposits

Deposit Source	BPL grade	Constituent, %[a]							
		CaO	P_2O_5	F	CO_2	R_2O_3	Na_2O	MgO	SiO_2
Central Florida	73	48.9	33.4	3.9	3.0	2.12	0.53	0.29	4.5
North Carolina	66	48.5	30.2	3.7	5.5	1.14	0.83	0.54	2.1
Morocco	70	51.6	32.1	4.1	5.3	0.55	0.79	0.43	1.4
Gafsa, Tunisia	63	48.3	28.8	3.4	6.3	1.22	1.30	0.59	1.8
Taiba, Senegal	82	51.2	37.4	4.0	1.7	2.06	0.20	0.06	2.9
Togo	80	52.3	36.6	4.0	1.8	1.78	0.27	0.11	1.8
Kola, Russia	83	52.0	38.2	3.1	0.2	3.14	0.50	0.06	2.0
Spanish Sahara	78	51.9	35.8	3.8	2.3	1.17	0.40	0.14	3.4
Angola	81	51.3	37.2	4.0	2.1	1.47	0.62	0.10	1.5
Jhamar Kotra, India	88,	54.2	40.1	3.6	0.7	0.70	0.11	0.04	1.2
Jordan	74	53.0	33.8	4.0	4.9	3.36	0.51	0.18	5.6
Israel (Oron)	68	52.7	31.3	3.6	7.5	0.45	0.75	0.24	0.2
Sechura, Peru	66	46.5	30.2	2.9	4.4	1.65	1.85	0.50	3.2
Algeria	63	49.3	29.0	3.6	7.4	0.70	2.00	0.81	1.0

[a]TVA analyses of authentic samples, as received; moisture-free basis (105°C)

Potential Benefit of Calcining Concentrates

All concentrates do not have the same potential for upgrading their P_2O_5 values by thermal treatment. This mainly depends on the particular composition of their apatite component, where the degree of upgrading increases with increasing carbonate substitution in the apatite's composition.

Thermal alteration of carbonate apatites proceeds by the main reaction:

$$Ca_{10-a-b}Na_aMg_b(PO_4)_{6-x}(CO_3)_xF_{0.4x}F_2 \xrightarrow{600^\circ-850^\circ C}$$

$$Ca_{10}(PO_4)_6F_2 + CaO + CO_2 \uparrow + (0.4x)F \uparrow$$

where the weight loss due to evolution of CO_2 and part of the fluorine determines the P_2O_5 content of the residual solids (calcine). Results based on this calculation are also included in Table X for the 14 phosphate concentrates.

These results indicate increases in BPL value ranging up to 4 units. This does not include a correction for the combined weight losses due to organic matter and bound water, which would nearly double the weight losses given in Table X.[13,15] A further correction could be applied for the decomposition of carbonate minerals, based on their estimated abundance shown in Table XI.

It becomes clearly evident from the results shown in Table X that thermal treatment of certain types of phosphate concentrates offer distinct advantages in reduced transportation costs due to weight reduction and higher BPL content.

Estimation of $CaCo_3$ in Gangue Fraction

Here again, the apatite composition is used as the basis for the evaluation. By comparing the theoretical CaO/P_2O_5 wt. ratio of the apatite (Table X) with the actual CaO/P_2O_5 wt. ratio of the apatite concentrate (Table IX), as is done in Table XI, an approximate distribution of total CaO can be made. CaO in excess of that required for the apatite is assigned to $CaCO_3$; the validity of this assumption can be easily verified by microscopic inspection of concentrates.

These estimated quantities of free $CaCO_3$ in the 14 concentrate samples (Table XI) account for up to 70% of the gangue-mineral fractions listed in Table X. Knowledge of this shows where economies in acid reagents might be realized by further upgrading efforts.

Acidulation Properties of Concentrates

Concentrates differ in their reagent requirements for WPA manufacture, depending upon their CaO/P_2O_5 ratio; ratios above about 1.6 are generally viewed as uneconomical to process. The reason for these differing requirements was discussed in relation to the examples given in Table VII.

The final comparison of these 14 concentrates is a straightforward analysis of reagent requirements, assuming 100% conversion of CaO to calcium sulfate, and an excess of 2.5% SO_3 in the 54% merchant-grade WPA product. These results are reported in Table XII, expressed in several commonly used bases.

TABLE X

Properties of Beneficiated Phosphate Concentrates

Source	Properties of apatite component			Degree of beneficiation			Effect of Calcining[a]	
	X-ray $a_{obs.}$	Theor. P_2O_5 (%)	Theor. CaO/P_2O_5 ratio	P_2O_5 grade (%)	Wt. % apatite	Wt. % gangue	P_2O_5 grade (%	Weight loss (%)
Central Florida	9.340	37.2	1.483	33.4	89.8	9.2	35.0	4.6
North Carolina	9.326	35.4	1.553	30.2	85.2	14.8	32.1	5.9
Morocco	9.333	36.5	1.520	32.1	88.5	11.5	33.9	5.3
Gafsa, Tunisia	9.325	35.3	1.558	28.8	81.5	18.5	30.7	6.3
Taiba, Senegal	9.352	38.8	1.427	37.4	96.4	3.6	38.6	3.1
Togo	9.351	38.7	1.432	36.6	94.6	5.4	37.8	3.2
Kola, Russia	9.384	42.1	1.314	38.2	90.4	9.6	38.2	0
Spanish Sahara	9.352	38.8	1.427	35.8	92.3	7.7	36.9	3.1
Angola	9.350	38.5	1.437	37.2	96.5	3.5	38.5	3.4
Jhamar-Kotra, India	9.366	40.9	1.356	40.1	98.0	2.0	40.6	1.2
Jordan	9.333	36.6	1.520	33.8	93.1	6.9	35.7	5.3
Israel (Oron)	9.336	36.7	1.506	31.3	85.4	14.6	32.7	4.3
Sechura, Peru	9.340	37.2	1.487	30.2	81.2	18.8	31.6	4.4
Algeria	9.325	35.3	1.558	29.0	82.1	17.9	30.9	6.1
(Fluorapatite)	9.374	42.2	1.318					

[a] CaO/P_2O_5 ratio on wt. % basis for ease of comparisons (see discussion in text).

[b] Effect of thermal alteration of apatite only; does not include losses from organic matter, water, or decomposition of $CaCO_3$.

TABLE XI

Estimations of $CaCO_3$ Content of Phosphate Concentrates

Product Source	Basis of calculation: CaO/P_2O_5 wt. ratio: Theor. (A)	Actual (B)	Difference (B - A)	CaO content (wt.%): Total	As apatite	As $CaCO_3$	$CaCO_3$ content: Wt.% in product	% of gangue
Central Florida	1.483	1.464	-	48.9	48.9	0	0	-
North Carolina	1.553	1.606	0.053	48.5	46.9	1.6	2.9	20
Morocco	1.520	1.607	0.087	51.6	48.8	2.8	5.0	43
Gafsa, Tunisia	1.558	1.677	0.119	48.3	44.9	3.4	6.1	33
Taiba, Senegal	1.427	1.370	-	51.2	51.2	0	0	-
Togo	1.432	1.429	-	52.3	52.3	0	0	-
Kola, Russia	1.314	1.361	0.047	52.0	50.7	1.3	2.3	24
Spanish Sahara	1.427	1.458	0.031	51.9	51.0	0.9	1.6	21
Angola	1.437	1.380	-	51.3	51.3	0	0	21
Jhamar-Kotra, India	1.356	1.352	-	54.2	54.2	0	0	-
Jordan	1.520	1.568	0.048	53.0	51.6	1.4	2.5	36
Israel (Oron)	1.506	1.684	0.178	52.7	47.1	5.6	10.1	70
Sechura, Peru	1.487	1.540	0.053	46.5	44.9	1.6	2.9	15
Algeria	1.558	1.700	0.142	49.3	45.2	4.1	7.4	41

TABLE XII

Acidulation Properties of Concentrates for WPA Production

Product Source	Properties of concentrate			Acidulation requirements[a]						
	% CaO	% P_2O_5	Wt. ratios CaO/P_2O_5	Tons rock per ton 54% WPA	Tons H_2SO_4 per ton of: Rock	54% WPA	P_2O_5	Tons sulfur per ton of: Rock	54% WPA	P_2O_5
Central Florida	48.9	33.4	1.464	1.617	0.856	1.384	2.563	0.279	0.451	0.835
North Carolina	48.5	30.2	1.606	1.788	0.849	1.518	2.811	0.277	0.495	0.917
Morocco	51.6	32.1	1.607	1.682	0.903	1.519	2.811	0.294	0.495	0.917
Gafsa, Tunisia	48.3	28.8	1.677	1.875	0.845	1.584	2.933	0.276	0.516	0.956
Taiba, Senegal	51.2	37.4	1.369	[illegible].444	0.896	1.294	2.400	0.292	0.421	0.780
Togo	52.3	36.6	1.429	1.475	0.915	1.350	2.500	0.298	0.440	0.815
Kola, Russia	52.0	38.2	1.361	1.414	0.910	1.287	2.383	0.297	0.419	0.776
Spanish Sahara	51.9	35.8	1.458	1.508	0.908	1.369	2.535	0.296	0.446	0.826
Angola	51.3	37.2	1.379	1.452	0.898	1.304	2.415	0.293	0.425	0.787
Jhamar-Kotra, India	54.2	40.1	1.352	1.347	0.949	1.278	2.367w	0.309	0.417	0.772
Jordan	53.0	33.8	1.568	1.598	0.928	1.483	2.746	0.302	0.483	0.894
Israel (Oron)	52.7	31.3	1.684	1.725	0.922	1.590	2.944	0.300	0.518	0.959
Sechura, Peru	46.5	30.2	1.540	1.788	0.814	1.455	2.694	0.265	0.474	0.878
Algeria	49.3	29.0	1.700	1.862	0.863	1.607	2.976	0.281	0.524	0.970

[a]Assuming 100% reaction of CaO to Calcium Sulfate, plus a 2.5% SO_3 loss in 54% WPA product.

No strict linear relationship exists between tons of rock and sulfur required to prepare one ton of 54% WPA, because of differences in their apatite mineral compositions. The last column in Table XII shows that the tons sulfur/ton P_2O_5 required varies from 0.772 to 0.970. The difference (0.198 tons) amounts to about $6.14 in added costs, assuming a current delivery price for sulfur of about $31/ton, or roughly the value of a ton of phosphate raw material.

Although the calculated reagent requirements in Table XII will obviously change for other grades of concentrates, the underlying cause for the difference in reagent requirements among rock sources relates to the different compositional forms of their apatite component.

Some Implications of Technical Evaluation

The discussions in this technical evaluation section amply demonstrate that competitively priced commercial concentrates are not comparable in their technical end uses.

Commercial raw materials can be classified on a relative value scale, using the selection principles described, rather than a P_2O_5 basis alone.

There is obvious reason, therefore, for mine producers to adopt similar evaluation procedures for guidance in beneficiation practices so as to upgrade other chemical quality factors besides P_2O_5 content.

If additional beneficiation is indicated to meet these newly developing market specifications to insure marketability, then the added cost to be incurred becomes another factor to include in the economic evaluation of mine production costs.

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