TWELFTH FORUM ON THE GEOLOGY OF INDUSTRIAL MINERALS

APRIL 22–24, 1976
ATLANTA, GEORGIA

STATE OF GEORGIA
DEPARTMENT OF NATURAL RESOURCES
Joe D. Tanner, Commissioner
ENVIRONMENTAL PROTECTION DIVISION
J. Leonard Ledbetter, Director
GEORGIA GEOLOGIC SURVEY

ATLANTA
1978

INFORMATION CIRCULAR 49

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CONTENTS

CONTENTS — Continued

CONTENTS – Continued

KEYNOTE ADDRESS FOR THE FORUM ON THE GEOLOGY OF INDUSTRIAL MINERALS, ATLANTA, GEORGIA, APRIL 22, 1976

PHILOSOPHICAL TEST-PITTING, OR THINKING WHILE WE DIG

Donald A. Brobst
Deputy Chief, Office of Mineral Resources
U.S. Geological Survey, Reston, Va.

It seemed appropriate to me as we start into the Twelfth Forum on the Geology of Industrial Minerals that we take a few minutes to examine the position of industrial minerals in the economy today and to consider how geology and geologists can assist in the continued availability of these minerals in the market. An old shaggy dog proverb says, "If you don't know where you are going, you may wind up someplace else." These are the days of highly specialized skills in business, technology, and science. Each of us looking in one of these fields has a tendency to get bogged down in the details that are strewn about our own particular part of the field. Today I choose to present the perspective of one professional geologist who has had an interest in industrial minerals for many years.

Mineral materials are a highly critical part of the economy of our nation. I am constantly amazed these days how few people really grasp this uncomplicated fact. Table 1 shows the value of the U.S. production and imports of fuels, nonmetals, and metals in wellhead and mine portal condition for the years 1972 and 1975. I chose 1972 for comparison with 1975 because 1972 was the last of the "normal" years prior to the oil embargo of late 1973. Those figures represent the peak of economic growth in the last upward cycle. The data show several especially important things about the changes of the last 3 years. The value of domestic production had nearly doubled by 1975 to 63 billion dollars, whereas the value of imports had more than doubled. In both 1972 and 1975, the lion's share of the values shown are for fuel materials, but note the greatly increased value of both the domestic and imported fuels in 1975. Another point needs to be made from this table, namely that the value of the nonmetals in both 1972 and 1975 was nearly twice that of the metals. These relative values of materials are virtually unknown to the public and, unfortunately, I suspect, unknown as well to a large segment of those involved in certain aspects of business, economics, and government. The table indicates that in 1972 the value of these materials was 35.5 billion dollars in a 1.1 trillion dollar economy, or 3.1 percent of the nation's GNP; in 1975, these materials had a value of 84 billion dollars in a 1.5 trillion dollar economy, or 5.6 percent of the GNP. In other words, in 3 years, the share of the value of these materials in the GNP also had nearly doubled, chiefly because

TABLE 1

Mineral materials in the United States economy
(In Billions of Dollars)

[Data from U. S. Bureau of Mines]

	1972	1975
U. S. Production		
Fuels	22.0	49.5
Nonmetals	6.4	8.6
Metals	3.6	4.9
	32.0	63.0
Imports		
Fuels	2.0	19.0
Nonmetals	0.5	
Metals	1.0	2.0
	35.5	84.0
GNP	1,150	1,500
Fuels & minerals (percent of GNP)	3.1	5.6

TABLE 2

Major mines (150,000 tons or more annually) in United States producing industrial minerals[1]

Number of Mines	Commodity	Area of U. S.
31	Phosphate	Southeast and West
7	Potash	New Mexico
2	Asbestos	California and Vermont
1	Boron	California
Total 41		

[1] Does not include cement, sand and gravel, crushed stone

of increased cost of fuels. Whether the value is 3 or 6 percent is again not the most important point, which is that these fuel and mineral materials are the very foundation of our economy. Who would care to do without them?

Another way to view this mineral business is the manner of the "London Mining Journal" in presenting its annual summary of the world mining industry. The magazine lists the major mines that produce 150,000 tons or more annually of 22 major commodities involved in world trade: asbestos, bauxite, born, diamonds, titanium, potash, fluorspar, phosphate, iron, copper, lead, zinc, tin, gold, silver, platinum, manganese, nickel, uranium, chromium, molybdenum, and mercury. The Journal claims that 95 percent of all noncoal mining activities can be accounted for by a close examination of only these 22 metals and minerals. The 1975 list contained the names of 1,072 mines. The United States has 172, or 16 percent of the world's largest mines. Of these, 41 mines, or 4 percent of the world's largest mines, produce the four industrial minerals shown in Table 2. Some high-volume commodities are conspicuously absent from the list of 22, in part because they do not make a sizable part of international trade, and also because, among other things, they are more widely distributed around the world, and their value is such that long-distance hauling to market is not feasible.

Thus, it becomes clear that one must do some digging to get into the subtleties of the industrial minerals. Although these materials traditionally have carried a little less glamour than metals, many of them are indispensable in the production of energy materials and metals themselves, all the way from exploration and mining to fabrication.

Table 3 is a summary of the first 20 industrial mineral commodities, ranked by dollar value and production in thousands of short tons. Although it does not show on the table, the dollar value of the first 10 items accounts for virtually all the 8.6 billion dollars for the industrial minerals in last year's GNP. The rank of the next 10 industrial minerals emphasizes the fact that the dollar value and production of these minerals drops off very quickly after the first 10. When it comes to dollar value and production figures, however, the table really does not tell the whole story of the critical nature of these materials.

Certainly the major share of the industrial minerals business lies in cement, crushed stone, sand, and gravel. Together these accounted for more than 5 billion dollars' worth of sales last year. The cememt came from 57 companies, which operate 175 plants. The crushed stone came from 4,800 quarries. According to the U.S. Bureau of Mines, however, 77 companies operating 740 quarries did 50 percent of the business. Limestone was 70 percent of the output, granite 10 percent, and other types of rocks constituted the remaining 10 percent. These rocks were used in construction and maintenance (70 percent), cement and lime (15 percent), agricultural uses (12 percent), and flux (3 percent). Sand and gravel last year came from 7,000 operations, but 68 percent of all the sand and gravel came from 1,200 commercial operations. Forty-eight percent was used in streets and highways, 49 percent in buildings and heavy construction, and 3 percent was consumed in all other uses.

The industrial mineral business has some features that make it quite different from the metals industry. Many of the companies that produce these industrial minerals are scattered over the United States, supplying materials to a great degree on a local or regional basis as well as nationally from centers of commercial ore deposits. Many of the companies have tended to be small. Because deposits of their materials have been relatively easy to find, and because relatively few restrictions have been placed on them until recently, these companies have survived and prospered, supplying the markets with needed materials at relatively low cost. Many companies have accomplished this without fulltime, or even part-time geological service.

Many signs now point to a new era ahead. Energy costs have risen for the production of everything from the basic raw materials to the finished product. Not only is it getting harder to find new, large mineral deposits close to the markets, but in certain types of industrial minerals, it is even getting harder to find a commercially usable deposit at all. The easy-to-find, well-understood types of deposits of many minerals close to the surface have already been found. This in turn is requiring longer lead-time to find and get new deposits into production. The reasons for this involve both

TABLE 3

Production, value, and rank of 20 major industrial minerals

[Data from U. S. Bureau of Mines]

	Rank	Value ($Millions)	Production (1,000 Short Tons)	Rank
Cement	1	2,000	69,600	3
Crushed stone	2	2,000	857,000	2
Sand and gravel	3	1,500	861,304	1
Phosphate	4	880	49,000	5
Sulfur	5	486	7,250	9
Clay	6	423	50,900	4
Lime	7	419	18,850	7
Salt	8	376	43,418	6
Potash	9	227	2,530	12
Bromine and compounds	10	200	208	21
Soda ash	11	167	4,122	10
Boron and compounds	12	150	1,100	13
Gypsum	13	45	9,760	8
Diatomite	14	42	567	19
Sodium sulfate	15	30	665	17
Barite	16	16	1,066	15
Asbestos	17	15	100	25
Vermiculite	18	14	340	20
Feldspar	19	12	775	16
Fluorspar	20	12	130	22

geologic availability and various aspects of economic availability.

Now land-use and environmental problems are arising. These problems involve environmental impact from the production of the materials and even the more basic questions of allowable land use for mining and beneficiation operations. The capital for investment in exploration and production facilities is getting harder to find and, even worse, it is becoming more expensive to use.

What we are seeing is a constant struggle between the constraints placed upon the supply system and the exponential growth of varying rates in each industry for supplies of those materials. The population grows, the urban community grows, and as America enters its third century there will be considerable demand for building of all sorts, including new construction as well as replacements of old facilities.

Through all this expansion, we must continue to keep in mind that exponential growth in the use of various geologic materials cannot continue forever. The rate at which we use or consume materials is certainly a very important factor, which must be considered along with the amount of materials geologically and economically available to us.

We have a tendency to look at mineral production on an annual basis. Was this year's production larger or smaller than last year's? Often we do not look at what the trends are really telling us. For example, the annual consumption of sand and gravel in the United States rose from 300 million tons in 1950 to nearly one billion tons by 1975. That is spectacular growth, even for a commodity that is geologically reasonably abundant.

Let me give you another example of this exponentially increasing production from some data on barite, 1850-1975 (Table 4). The year 1850 is a good place to start because we may consider that time to be about the beginning of modern industrial expansion in the western world. The world has produced 110 million tons of barite; nearly 80 percent of that has been produced in the last 30 years. The United States has produced nearly 75 percent of its barite and has received 94 percent of its imports in the last 30 years. In the 1970 edition of "Mineral Facts and Problems," the U.S. Bureau of Mines estimated that the growth rate of barite demand would be about 2 percent per year. On this basis, the U.S. demand by the year 2000 would require the cumulative production of about 37 million tons of barite; a similar projection for the rest of the world would require 105 million tons. Those numbers virtually equal the total U.S. and the total world production from 1850 to the present. The oil embargo of 1973 and the resulting scramble in the search for oil in other parts of the world suggest that these estimates of barite-demand made in 1970 are perhaps low. Work done recently by the U.S. Geological Survey suggests that the United States might consume 25 million tons of barite in oil and gas exploration and production drilling in the next 12 years. That will come very close to asking industry to produce, or at least to have available through production and imports, as much barite as has been produced domestically in the last 30 years.

The chilling finale is that a one billion year supply of anything at the present rates of consumption would be exhausted in less than 600 years if there is a 3 percent rate of expanded use annually—the general rate of U.S. economic growth in the post-World War II years. A 2 billion year supply would last only about 25 years longer, because it is equal to only one more doubling time of the use of the one billion year supply. Of course, the consumption of a finite supply would not follow such a pattern of increase to its final exhaustion, but the point is that the level of consumption is just as significant an element in the life of a supply as the magnitude of the resource. Exponential growth in consumption can quickly exhaust even a huge supply.

In short, we will have to learn to use better what we have geologically available to us. We shall have to go out and search for new types of deposits of many minerals. This will take time, money, and people working together and exercising brain power. We cannot afford to wait until the new deposits are needed tomorrow morning and then begin to look. Everyone involved in this business is going to have to begin taking a longer range view of the problems. It will be to everyone's advantage to do so. The cost of the burden of action can be spread around between industry, academia, and government, although it is the consumer who eventually pays. Let us not forget who that consumer is. We are, along with all the other people to whom we refer as "they" and "them" in our criticisms.

What can geologists do? A great deal, I think, from all the places where they might be working—within companies, as consultants to companies, in the universities, and within geologic agencies of the State and Federal governments.

Geologists can deal with specific problems. Wedding science and the dollar has been done in the past and can be done with greater intensity in the future. Geologists themselves have not always taken the pains to explain in their reports what some of their rather scientific-sounding prose really means in terms of practical applications. By the same token, management in many corporations perhaps has not utilized people with much or any geology in their background; as a result, management has been unwilling to be talked into expenditures whose value to them cannot be clearly foreseen. There have been countless cases in which some basic geologic information might have saved a company not only its shirt, but its life. A few hours of a geologist's time could have prevented disaster for a company who though it was buying a 50-foot-thick gravel deposit, only to discover when the bulldozers moved in that it was only 2 feet thick and simply looked 50 feet thick because the gravels on the edges of the perched deposit had rolled downhill.

To a great degree, our industry is specification-oriented. Specifications for raw materials and products are fine, but I think that there are times when we continue to live with specifications for materials that are more historical and more dinosaurian than they really need to be today. A rock becomes a resource only when someone is willing to pay for it to do the job that they have in mind. In some jobs, limestone has been the preferred material for crushed rock. It may be hauled a few extra miles to its market area, when a granitic rock or some other type of igneous rock may be closer and just as adequate for the work to be done. In the days of World War II, premium prices were paid for so-called ruby mica, when laboratory testing showed that the other varieties, including the much more abundant green variety, were just as good for the use intended. If a

TABLE 4

**World and United States barite production,
and United States imports, 1850 - 1975
(Millions of Tons)**

	World	U. S.	U. S. Imports
1850 - 1914	6.5	1.2	0.2
1915 - 1918	1.2	.7	.0
1919 - 1944	16.7	7.1	.9
1945 - 1975	86.2	27.6	15.2
TOTALS	110.6	36.6	16.3
Percent of total since 1945	78	75	94

mining company is looking for new deposits of a material within a given radius of its market areas, a geologist can assist the company in assessing the potential for finding a commercial deposit within the area in a short time. This is considerably more advantageous in time and money to the company than to have its marketing experts running around the area looking for something that may not be available geologically in that area.

Geologists undoubtedly will be looking for new types of deposits as well as hidden deposits of types that we know how to work. New methods must constantly be devised to do this, and we still have a long way to go. For the near term, geologists can assist in the search for what we might call undiscovered deposits of "conventional" types not known in some areas. I will give an example again from my own work with barite. Deposits of black bedded barite such as those of commercial value found in Arkansas and Nevada are not known in the Appalachian region now. On the basis of my work, I believe that such deposits probably do occur in the Appalachian region. Some successful effort in search of them could go a long way toward assuring a domestic supply of barite for the drilling of oil wells off the Atlantic coast. This pattern of thinking holds promise for more commodities than just barite.

State and federal agencies may produce regional, state, or national assessments of the various resources available in those areas. These generally well-documented reports will help the company and consulting geologists to provide quickly the information the company management needs to make decisions pertaining to its future operations. The results of such studies can be made available quickly through publication of maps and reports that may combine much data, presented in ways that are useful to more people than geologists. Maps showing potential mining districts are really more valuable than those showing where we used to mine.

Geologists, wherever they are, can assist in educating the public about the nature of resource problems. Such problems have two fundamental aspects – geologic availability and economic availability (I would include legal aspects in this category). Geologic availability simply defined says that you cannot mine what you do not have, and that you cannot mine anything until you find it.

In the area of economic availability, we have another sign of that new era that I was talking about – environmental problems associated with mining. This is the old story that nobody wants a big hole next-door, but everyone wants the mineral materials they need at the lowest price and as handy to their project as possible. A basic conflict of interest exists here, and unfortunately both the consumer and the mining industry get caught. Zoning laws are frequently set up by boards that really do not understand that the availability of raw materials plays a role in the delicate balances within the economic growth that they so greatly desire to control. Many a suburban development has covered up a sand and gravel or other mineral deposit needed to make its contribution to the life of the metropolitan area. The public has not yet grasped the face that trade-offs in environment and zoning will be required even to perpetuate a lifestyle that approaches today's, much less one that exceeds it.

I think that geologists, in industry, academia, and all levels of government, can offer constructive suggestions to assist in the formulation of realistic policies that will allow private industry to supply the needs of the nation at realistic and reasonable price levels. It will take the best and most creative thoughts of all of us; we must all keep on thinking and talking while we dig.

THE CRUSHED GRANITE INDUSTRY OF THE ATLANTA METROPOLITAN AREA

Robert L. Atkins
Georgia Geological Survey
W. Robert Power
Georgia State University
Atlanta, Georgia

Introduction

Atlanta is the only major metropolitan area in the United States that depends on crushed granite as its sole source of local aggregate. Most cities rely on crushed limestone, natural sands and gravels, or trap rock. However, none of these rocks are available in the Atlanta area. Hence, the greatest concentration of crushed granite quarries in the country are in this area.

In 1973, 1.06 billion tons of crushed aggregate and 935 million tons of natural sands and gravels were produced in the United States. Approximately 95 percent of the natural sands and about 67 percent of the crushed aggregate were utilized as aggregate. Of the crushed aggregate in 1973; limestone accounted for 75 percent; granite, 12 percent; trap rock, 8 percent; and other rock types, 5 percent (Minerals Yearbook, 1973).

Georgia produced approximately 28 percent of the crushed granite in the United States in 1973. Georgia, North Carolina, Virginia, and South Carolina produced approximately two-thirds of the crushed granite in this country (Minerals Yearbook, 1973). Though very important to the southeast, it is apparent that this industry is minor to the nation as a whole.

Geology

Georgia includes parts of three major geologic provinces - the Coastal Plain, the Piedmont, and the Appalachian Valley and Ridge (Figure 1). As a result several different rock types are used for crushed aggregate within the state. The Valley and Ridge and Coastal Plain Provinces are comprised of sediments, indurated and non-indurated respectively, and limestone is the main rock type quarried for crushed aggregate purposes. Within the Piedmont Province, where granite and granitic gneiss are quarried, the major lithologies are granite, granitic gneiss, biotite gneiss, schist, and amphibolite. These high-grade metamorphic rocks and igneous rocks meet Atlanta's needs for crushed aggregate, and although the most economical aggregate sources are natural sands and gravels, the relative proximity and abundance of the city's aggregate are fortunate characteristics for this growing metropolitan area.

I. Cumberland-Plateau Section
II. Southern Valley and Ridge Section
III. Southern Blue Ridge Section
IV. Southern Piedmont Section
   IVa. Upland Georgia Subsection
   IVb. Midland Georgia Subsection
V. East Gulf Coastal Plain Section
VI. Sea Island Section

FIGURE 1. – Physiographic provinces of Georgia

Economics

The crushed granite industry began as a spin-off from the dimension stone industry. In 1883, crushers were installed at some of the dimension quaries to crush waste rock yielding stone for railroad ballast and macadam roads. (Watson, 1902).

Initially the location of the quarry depended on two factors: good rock exposures and railroad facilities. The quarries were located on flat rock exposures (pavements) or bosses with steep slopes. Many of the bare rock exposures were very large – up to several tens of acres. The rock was typically massive without internal joints except for exfoliation sheeting. With increasing costs of haulage and with the advent of modern highways, quarries were forced to locate closer to the market.

The distance from quarry to market is the single most important factor in the placement of a crushed stone quarry. Because the price of crushed stone at quarries is relatively standard, it is the additional cost of haulage that determines marketability. This haulage fee is based on the difficulty of loading and hauling the aggregate off of the work site; the total amount of time spent enroute to market; the amount of aggregate that is transported; and the size of the aggregate (Thurman, personal communication, 1976).

With haulage costs always on the increase, the closer a quarry is to its market, the more competitive its prices can be. An example is the small Bellwood Quarry which is located in west Atlanta. Blasting is required every two days to maintain production, and the size of its blasts are restricted. What is more, the quarry probably will not be able to expand laterally but must continue vertically (Pertsch, personal communication, 1976). Despite these restrictions, it is still able to compete for the downtown business because of its close proximity to the market.

At the present, there are 19 quarries serving the Atlanta area. Of the 19, four have been opened within the last three years (Figure 2). All are operating far below their capacity due to the current recession. These 19 quarries, all located within 20 miles of Atlanta, have individual capacities ranging from 300,000 to 3,200,000 tons per year and a combined capacity for producing 25 million tons of aggregate a year (Hitchcock, personal communication, 1976). The typical market area for each is usually within 6 miles of the quarry, and the haulage distances average between 6 to 15 miles.

Specifications and Testing

Specifications of the State Highway Department are written so that no rock type is automatically excluded. However, the highway business represents the major consumer of aggregate, and very few quarries are successful ventures when the rock being quarried is not acceptable for highway department use.

It is interesting to note some significant differences between limestone and granite, the two most widely used rock types for Georgia highway construction. Generally, limestone makes a more suitable highway base because it grades and compacts better than granite. Granite is less dense, more durable, and not as subject to polish, therefore making it a better surface treatment material. Typically, limestone contains deleterious materials such as chert and shale and is generally stratified (inhomogenous). Quarried granite, on the other hand, is usually massive with only one lithology (homogenous) (Malphurs and Crissler, personal communication, 1976).

Where large masses of uniform rock cannot be found in a marketable area, interlayered gneiss, amphibolite, mica schist, and quartzite have been substituted into the aggregate market. The lithologies must be selectively quarried to meet varying specifications. For

FIGURE 2. — Quarries in the Atlanta area.

example: up to 35 percent mica schist can be used in some asphalt mixes, and up to 5 percent schist can be used in concrete mixes. Mica schist weakens concrete mixes if it exceeds the 5 percent limit because of its platy characteristics. Hornblende gneiss or amphibolite is generally not used because of its high density since it costs more to transport and results in less competitive prices.

Before placing a quarry on the approved list of the Department of Transportation, the department performs the following: a geologic map of suitable lithologies and deleterious material is compiled; an appraisal of the crushing operation is made; the rock is sampled from the quarry walls and/or from test corings; and the following tests are run on the samples:

1. Soft fragments (mica schist)

Soft fragments, consisting primarily of mica shcist, fail the scratch test. This test consists of scratching a mineral with a rod that has the hardness of calcite. Up to 35 percent mica shist may be used in surface treatment and asphalt mixes, and up to 5 percent may be used in the concrete mixes.

2. Minus 200 Material

Minus 200 material is utilized in both asphalt and concrete. Problems with minus 200 material include the following: in asphalts, too many fines prevent the asphalt from sticking to the aggregate; and in concrete with too many fines, more water is required in order to get workability. (Too much water is not acceptable because it weakens the concrete.)

3. Flat or Elongated Pieces

Flat or elongated pieces are defined as having the longest dimension five or more times the least dimension. Flat or elongated pieces prevent adequate compaction and weaken the concrete. This can be a problem in the gneiss quarries due to breakage along the foliage planes. Force feeding the cone crusher usually eliminates this problem.

4. Sulfur Content

Georgia is probably the only state that controls the pyrite and other sulfides in the aggregate to be used for bridge construction. Sulfides do not weaken the structure but may cause staining which is not aesthetically pleasing.

5. Soundness

Soundness is not usually a problem with the granites of the state.

6. Abrasion

Abrasion is the percent weight lost due to wear, and it varies from quarry to quarry. Generally, quarries west of Atlanta are class A in terms of abrasion, and those east of Atlanta are class B. Class A indicates the rock has no restrictions placed upon it because of its abrasive characteristics, but class B granites are restricted due to a high abrasion loss. Class B granites cannot be used in surface coarse treatment and certain types of plant mixes. What is more, the caprock in quarries does not meet the state specifications; it can be crushed for use in private developments such as subdivisions where loads are not as heavy and traffic is not as great as along state roads.

The Georgia Department of Transportation groups the aggregate into two groups, siliceous and non-siliceous because the characteristics of limestones and granites are different. The state specifications on aggregate quality take these differences into consideration.

In order to meet the state's specifications, most quarries have initiated quality control programs and maintain rigorous standards. At least once a week, each quarry participating in the aggregate certification program is visited by representatives of the Georgia Department of Transportation Aggregate Control Branch. Samples are taken from the belt, truck, or stockpile, and tests are run to determine if the quarry is providing materials that meet the specifications of the state. In addition, a small number of samples is taken on the job site for statistical correlation with quarry samples. Each quarry is required to maintain an adequate quality control laboratory. The laboratories and personnel are tested and certified by the state to insure accuracy and proficiency. Each quarry has a commitment to maintain high standards of quality control and may be removed from the approved list for not fulfilling this commitment (Malphurs, personal communication, 1976).

Exploration and Development

When exploring for possible quarry sites in the Atlanta area, it should be remembered that the best quarry rocks are free of joints and other openings. This is just as important as the rock type itself because jointed rock is harder to break into smaller sizes and drilling and blasting costs increase. Also, chemical weathering which occurs along joints weakens the rock, sometimes below state specifications.

Once a suitable site is agreed upon, the rock is core drilled, and tests are run prior to the opening of the quarry. Usually a small or portable crushing plant is set up to determine if the rock is suitable to install a more permanent system (Figure 3). In addition, with a portable crusher, less expense is incurred if the market for the stone does not develop.

Most quarries are opened on flat rock exposures and require no initial stripping. If they expand horizontally beyond the exposed pavement, removal of overburden by pans may be necessary. Even where soil overburden is absent, the upper few feet of rock that is caprock may be unacceptable because of incipient chemical alteration. Some quarries must be expanded vertically using conventional open-cast techniques. Although this method may eliminate stripping and require less land, water becomes a problem and haulage costs increase. In some quarries haulage costs are reduced by placing the primary crusher in the quarry and transporting the rock from here to the plant on a conveyor belt.

The size and shape of a quarry varies. Some

FIGURE 3. — Example of a portable crushing plant

quarries have attained a depth of 300 feet; others occupy up to 100 acres of land. Typically, a quarry has two or more levels on which to work. Depending on the rock classification and/or the plant's capabilities, all sizes of crushed aggregate are produced for highway construction, railroad ballast, subdivision development, and building construction.

Production Methods

After a new quarry site has been selected and determined economically feasible, production begins. This entails blasting, crushing, and screening of the granite.

Blasting patterns, the number of blasts necessary, and the depth to which the blasts are set all depend upon the type of rock and equipment, the strength of the explosives used, and the environmental factors involved. Next, the rock is typically loaded with an electric shovel of 4 to 7 yards capacity. Oversized blocks are drop-balled, and a rubber tire loader cleans up. Then trucks of 35 to 85 ton capacity haul the rock 300 to 3500 feet to the primary crusher. Usually, this crusher is located on the lip of the quarry, but several operators have located them within the quarry. Though the primary plant operation varies slightly from plant to plant, it generally consists of crushing and screening until the desired size of aggregate is obtained. This crushing of the granite is expensive and very hard on the equipment. In fact, the crusher has to be adjusted almost on a weekly basis (Lambert, personal communication, 1976). When the screening process is completed, the stone is stockpiled according to size. Underneath each stockpile is a vibrating feeder which can transfer the stone to the loading bins or blend different sized stones together. The aggregate is ready for transporting to its respective market at the end of this process.

Environmental Protection and Restoration

In 1969, Georgia passed its first state law governing the removal of rock, the control of waste products, and reclamation of the land. As a result, every operator must be licensed and bonded, and each year he must submit a mined land-use plan to the Environmental Protection Division. The purpose of this is to make the operator aware of his future needs for expansion, development, location of reserves, water control, dust control, and land reclamation (including the anticipated cost of the reclamation). This requirement is for all types of mining in Georgia. The following is a summary of points which the land-use plan must include:

1. Total acreage involved in the property.
2. Total acreage to be affected when the mining is complete.
3. An anticipated date for the completion of the mining.
4. Number of acres to be affected and licensed during this licensing period.
5. Map illustrating use of the land-haul roads, quarry, and ponds.
6. Reclamation date and objective. None of the quarries in the area is yet reclaimed. Most operators suggest that the quarries would make a lake or a solid waste disposal site.
7. Plans for natural drainage and water disposal. Quarries typically have closed water systems. The water is recycled through the plant.
8. Dust control plans. The operator must give the source and details of control methods. Equipment may include waterwagons to control the dust on haul roads and spray bars in the plant to control dust in the crushers, belt, and stockpiles.
9. Noise. Noise is regulated by the city, county, and Mining Engineering Safety Administration. Quarries located near housing or apartments are restricted as to their working hours; others are not. Most complaints about noise refer to trucks hauling the stone from the quarry. The in-house noise is controlled by the MESA. Ninety decibles per hour is the maximum noise allowed for an eight hour day.

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THE ORIGIN OF GEORGIA'S KAOLIN DEPOSITS

Roger S. Austin
Freeport Kaolin Company
Gordon, Georgia

Introduction

Commercial deposits of white kaolin clay occur along the northern margin of the Georgia Coastal Plain. The center of kaolin production is in Twiggs, Wilkinson, and Washington Counties, with related operations extending into Alabama and South Carolina. Low grade bauxite is associated with most of these deposits, and an important alumina industry has developed in the same geological area, near Andersonville, Georgia. The kaolin and associated bauxite occur within Cretaceous and Early Tertiary strata which dip gently toward the present coast. These strata unconformably overlie metamorphic and igneous rocks which crop out immediately to the north in the Piedmont Province.

Raw kaolin is processed to form a fine, white, high-brightness, pure kaolinite clay product. This material is shipped in dry or slurry form to the paper, paint, plastics, and rubber industries and is used as both pigment and filler.

Of all the kaolin occurring in the "laolin belt", only a small portion is of commercial value. These unique deposits generally must:

1. Have an overburden to kaolin ratio of less than 10:1.
2. Contain less than 10% sand.
2. Have a clay particle size distribution of 55% to 90% finer than 2 microns.
4. Be extremely white and have high reflectance.
5. Be able to produce a highly fluid aqueous slurry containing up to 70% kaolin.

The commercial kaolins are thus only those portions of the strata which consist of nearly pure kaolinite. The deposits must be at least 1.2 to 3.0 meters (4 to 10 feet) thick, many acres in areal extent, and contain hundreds of thousands of tons of raw clay.

The origin of the kaolin includes three principle factors.

First – The Georgia kaolin deposits occur within a regional system of aluminous laterite and lateritic sediment extending from Arkansas through Mississippi, Alabama, Georgia, and South Carolina in the Gulf and Atlantic Coastal Plains and from Alabama to Virginia in the Appalachian Mountains (Fig. 1).

Second – The kaolin and associated bauxite occur among 2 significantly different stratigraphic units. The difference is not simply one of age, but also mode of origin. The kaolin and bauxite in the Cretaceous units were formed by laterization, and are residual products derived from former aluminous sediments. The kaolin and bauxite in the Tertiary units include both lateritic and sedimentary deposits. They were derived largely from the Cretaceous deposits by erosion and were deposited in normal depositional environments. Some were then subjected to even further laterization.

Third – Though many of the Georgia kaolin deposits occur among Cretaceous sediments, the episode of laterization, erosion and redeposition which formed them is an Early Tertiary phenomenon. The kaolin and bauxite deposits are not the result of some unusual mechanical sorting or sedimentation process.

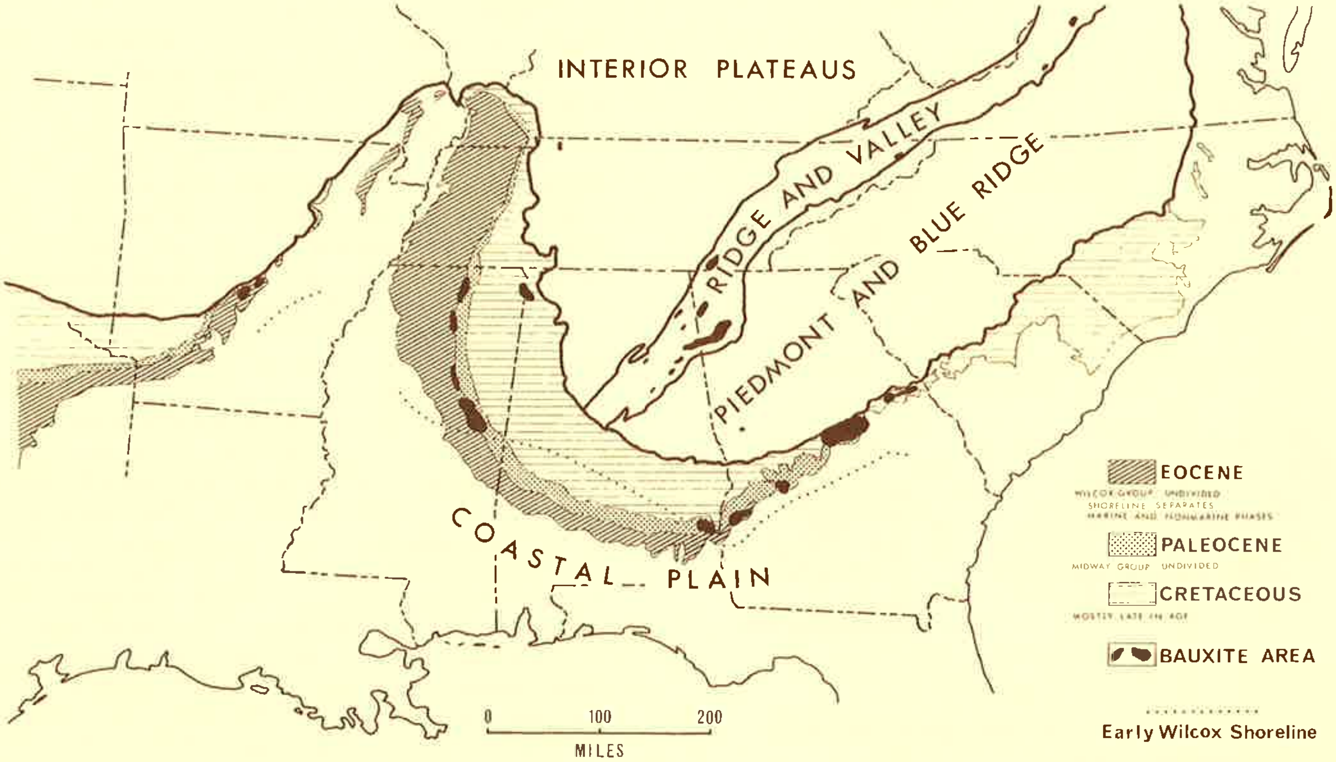
Description of the Deposits

Below the kaolin-bearing strata are a wide variety of metamorphic and igneous rocks. Where exposed and when encountered by exploratory drilling, the upper portion of these rocks are weathered, in some cases to a kaolinite and quartz residuum almost indistinguishable from the overlying sediment. These rocks are separated from the overlying Cretaceous sediments by a distinct unconformity.

Above are the Cretaceous units which consist of poorly stratified, nearly horizontal beds of quarz sand, kaolinitic sand, sandy kaolin, and kaolin. Locally, portions of the kaolin-rich sediments contain pisolitic kaolin and bauxite. Only the gross features of stratification remain to be seen, ranging from the characteristic parallelism of sediments to extensive crossbedding and associated quartz and kaolin pebble conglomerates within large cut and fill structures. The Cretaceous sediments in Central Georgia contain no marine invertebrate fossils. Only recently have palynological studies supported the oft-proposed Cretaceous age for these units.

The Cretaceous and Tertiary strata are separated everywhere by a major unconformity. The Tertiary strata consist of a variety of sediments including quartz sand, kaolin, conglomerates, lignitic kaolin, montmorillonitic clays, and limestone. It is within Tertiary sediments that marine invertebrate fossils first appear in the Georgia kaolin district. The distinctly marine units of Eocene age form the upper boundary of kaolin occurrence.

Residual bauxite and kaolin deposits are mostly of Cretaceous age in the central and eastern portions of the Georgia kaolin belt. The Tertiary kaolins in this area are largely non-pisolitic, though some exceptions do occur. In the western portion of the belt, the Tertiary kaolins and bauxites are the dominant commercial units, for most of the Cretaceous deposits appear to have been removed by erosion since their.



Kaolin and bauxite areas (black) of the southeastern United States (after Gordon, Tracey, and Ellis, 1958)

development. Only partially kaolinitized feldspathic sands and fossiliferous marine shales remain. Throughout the belt, bauxite occurs mostly in the middle portions of larger kaolin and sandy kaolin beds, or at the top of kaolin beds if erosion has removed the upper portion of formerly larger units.

A detailed comparison of Tertiary and Cretaceous kaolins reveals several marked differences. White Cretaceous kaolin commonly has very faint brown spots or mottled patterns compared to the uniformly white or greenish white Tertiary kaolin. The Tertiary kaolins commonly contain tubular forms, ranging in diameter from 1 to 3 mm (.04 to .1 in.) Whether burrows, fecal remains, or bryozoa casts, these tubes apparently are absent in Cretaceous kaolin. Isolated grains of very coarse sand and pebbles are present in some Cretaceous kaolin, suggesting in many cases an unnatural degree of poor sorting, a fabric produced only by extensive solution and recrystallization. In contrast, large isolated grains of quartz sand are less common in the Tertiary kaolin. Abundant submicroscopic quartz occurs only in the Tertiary kaolins, suggesting a truer sorting. Tertiary kaolins generally are more dense, have a lower porosity, and appear more compact than the Cretaceous kaolins. These have often been referred to as "hard" kaolins in contrast to the "soft" Cretaceous kaolins which tend to be mealy or even powdery when dry.

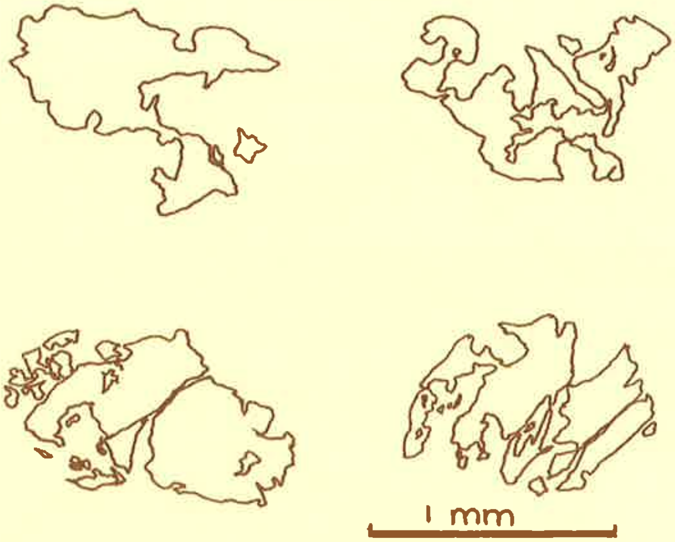
Examination with the petrographic microscope of kaolin thin sections showed even more distinction. Quartz grains in Cretaceous kaolins (Fig. 2) exhibit severe chemical corrosion, sometimes to the extent of deep embayment or even division into several fragments, all in optical continuity. Severe corrosion of the Tertiary quartz grains is less common. Large vermicular crystals of authigenic kaolinite, up to several mm in length are abundant in Cretaceous kaolin, yet uncommon or rare in Tertiary kaolins. The Tertiary kaolins are generally finer-grained than the Cretaceous, the large majority of their clay fraction being finer than 2 microns.

Heavy minerals of microscopic size are present in both Cretaceous and Tertiary kaolins, and there seems to be no significant difference between the mineral suites of the two. The suite, however, is remarkable in its simplicity. Zircon, rutile, some tourmaline and amorphous iron compounds were commonly the only phases present. These are all extremely stable phases under conditions of even the most severe chemical weathering.

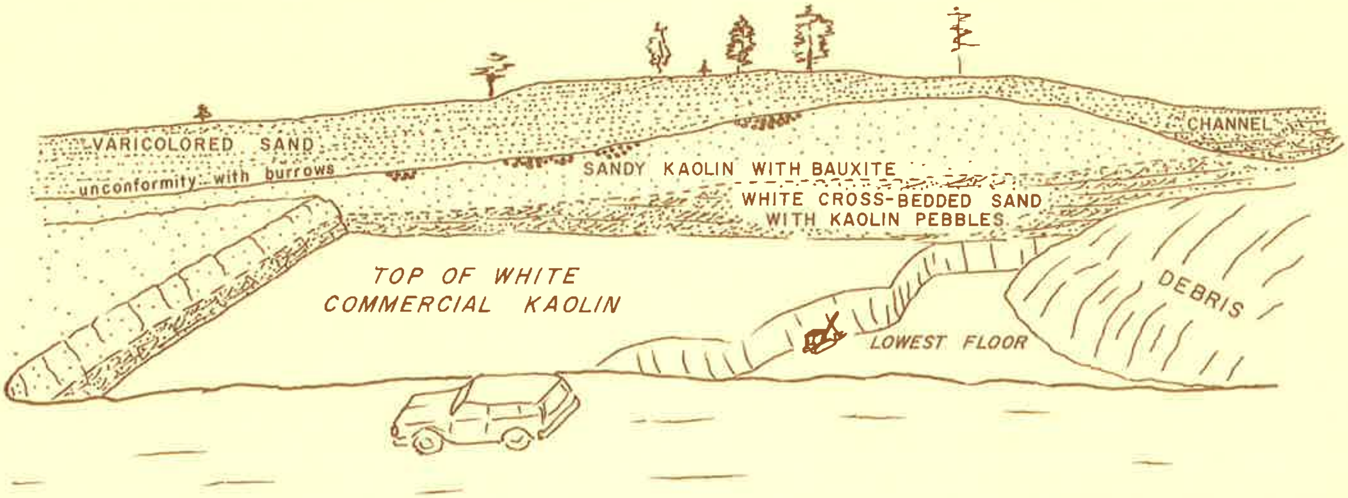
Kaolin mines in central Georgia provide excellent exposures of the Cretaceous units. One in Washington County (Fig. 3) contains a 6 meter (20 foot) thick layer of nearly pure kaolin. Above is 6 to 9 meters (20 to 30 feet) of the more characteristic sandy kaolin and sand with visible large-scale sedimentary structures. These are separated from overlying Tertiary sediments by an unconformity. Immediately below the unconformity, as is most common, is bauxite in discontinuous pods and lenses. Another mine nearby (Fig. 4) is similar in many respects. Again, sands and sandy kaolin are dominant. A bed of commercial kaolin has been exposed at the base. Bauxite occurs at the unconformity at the top. Above the unconformity is a typical Tertiary sequence of sands and montmorillonitic clays. The development of the bauxite appears ot have gone to completion prior to the final development of the Cretaceous-Tertiary unconformity. The bauxite had been partially removed by erosion prior to Tertiary sedimentation.

At a mine in Twiggs County, the relative age of the bauxite is established by the burrow of a small animal that lived on the erosion surface. Figure 5 shows a sketch of a portion of the network of chambers and tunnels these organisms formed. At the bottom of the sketch is an enlarged view of the burrow showing where the animal cut indiscriminately through individual pisolites. The bauxite certainly predates this activity.

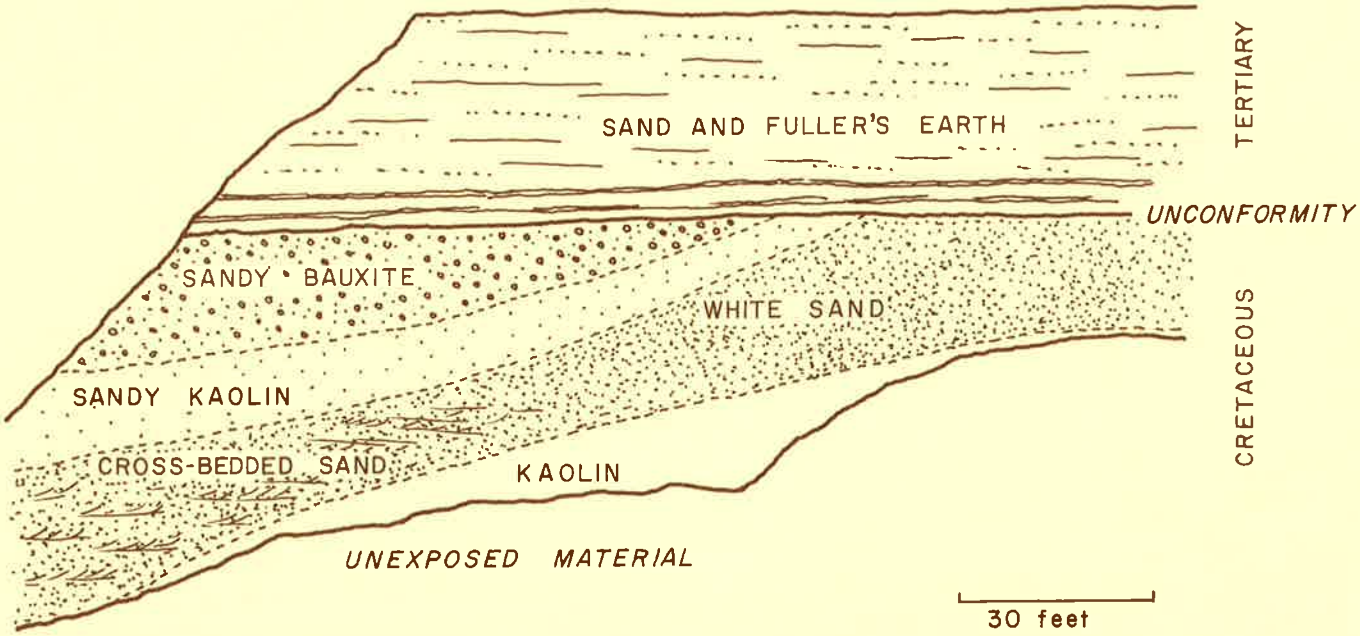
Another mine in Twiggs County provided an excellent exposure of Tertiary strata, and a sketch of the mine wall is shown in Figure 6. The Cretaceous-Tertiary unconformity is exposed throughout the mine. The Cretaceous materials include massive commercial kaolins



Pen sketches of selected quartz grains observed in thin sections of kaolin.



Sketch of typical kaolin mine in Washington County, Georgia.

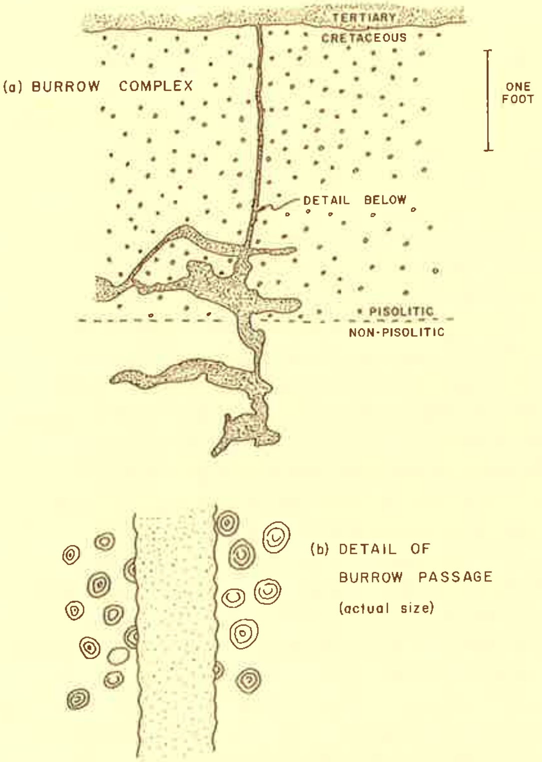


Sketch of kaolin mine high wall showing relation between kaolin and bauxite.

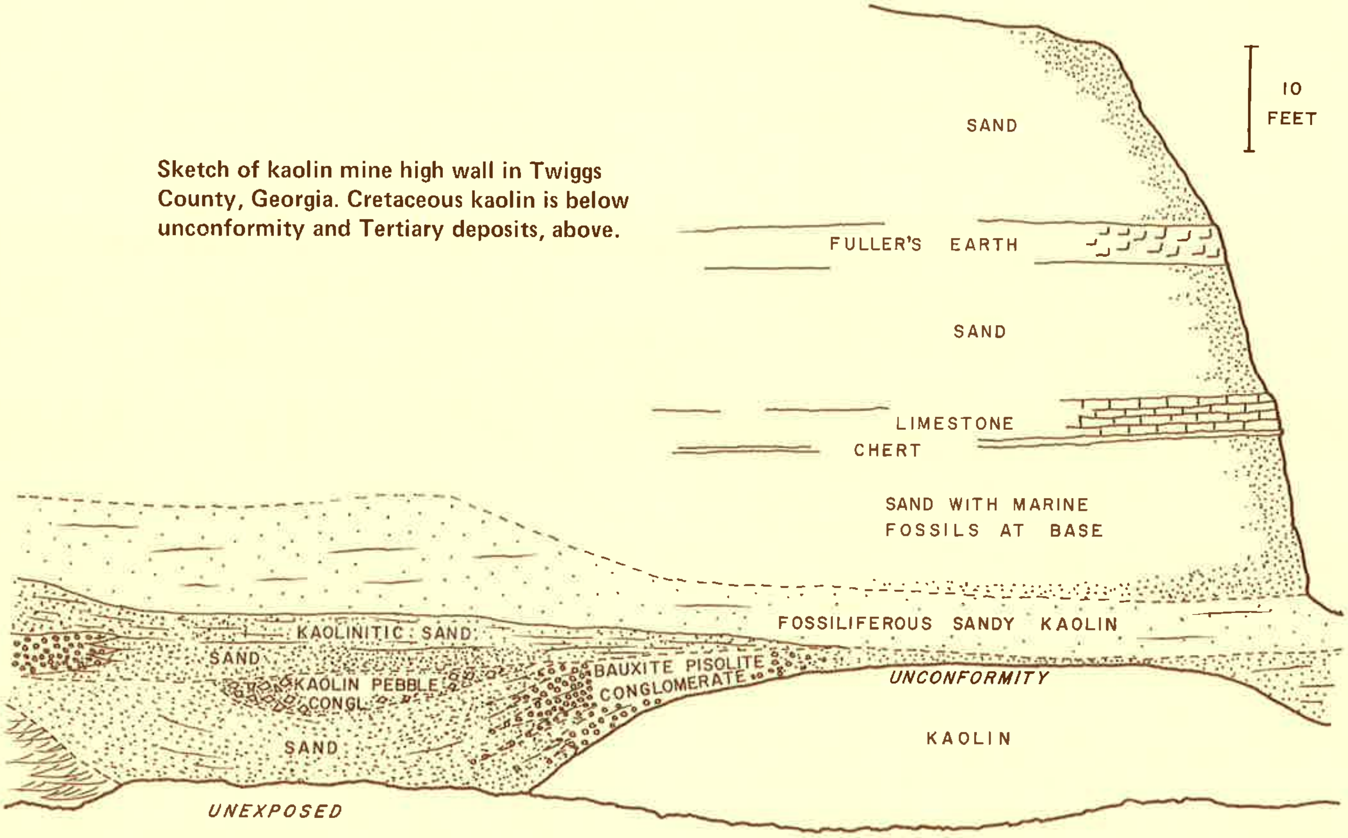
and poorly bedded sand and sandy kaolin. The Tertiary strata are the most striking. A large channel is present and is filled with quartz sand, kaolin pebble and boulder conglomerates, bauxite boulders, and cross-bedded bauxite pisolites. Though the bauxite detritus is present, no residual bauxite was exposed. These units grade upward into sandy kaolin beds containing terrigenous plant fossils and finally, small invertebrate marine fossils. A succession of marine sands, limestones, and montmorillonitic clays follow.

Geologic History

The geologic history of the Georgia deposits is essentially outlined in the generalized stratigraphic section shown in Figure 7. By Late Cretaceous time, a thick sequence of feldspathic sands, silt, and clay (perhaps already kaolinitic) had been deposited in the Georgia region, part of a more extensive series of apron-like deposits generally near and paralleling the present inner margin of the Atlantic and Gulf Coastal Plains. At the end of Cretaceous or in very early Tertiary time, the Cretaceous sea retreated and exposed the sedimentary deposits to subaerial weathering. The sediments were subjected to a climatic and subsequent weathering episide of intensive or prolonged rainfall which brought about extensive leaching of the newly-exposed sediments by a continual



Sketch of burrow (crustacean) at the Cretaceous-Tertiary unconformity in Twiggs County, Georgia.



Sketch of kaolin mine high wall in Twiggs County, Georgia. Cretaceous kaolin is below unconformity and Tertiary deposits, above.

downward flux of fresh ground water. The products or residues of this episode were an aluminous laterite which consisted of massive beds of bauxite and kaolin interspersed among quartz sands. The bauxite formed in the upper portions of the Cretaceous strata where leaching was relatively more effective (Type IA); and kaolin below (Type IB), where leaching was less effective, the ground waters having become too saturated with silica for bauxite to be stable.

During the development of the lateritic deposits of kaolin and bauxite, streams were cutting down into the altering sediments, eventually causing the removal of large portions. Some of the eroded material accumulated in the stream channels. These new kaolin beds, if deposited early enough, were then subjected to further leaching, further kaolinitizing and bauxitizing portions (Types II A and B).

By Late Paleocene or Early Eocene time the climate became more temperate, the amount of dissolved silica in the ground water increased, and bauxitrization ceased. Portions of the bauxite bodies then began to be resilicated especially around their margins, forming a secondary residual kaolin (Types IC and IIC). This is the only adequate explanation for the common occurrence of bauxite bodies being completely enveloped by kaolin. The kaolin is simply an alteration rim around an unaltered bauxite core. Erosion and redeposition of kaolin and bauxite continued during this period, producing deposits which were subjected to little or no further alteration (Type III).

During Middle to Late Eocene time, the sea began to advance across the altered, eroded, and redeposited Cretaceous, Paleocene and early Eocene deposits. Stream valleys were flooded by marine waters. Where the saline waters invaded the bauxites, montmorillonite, rather than kaolinite, became the alteration product.

Finally, the sea covered the entire region and the formation episode was brought to a close. A blanket of sand, montmorillonitic clay, and limestone was deposited, preserving the deposits for our use today.

Economic Aspects

The kaolin industry now benefits from several specific episodes.

1. Solution, recombination and recrystallization have produced great quantities of kaolinite. At the same time, it reduced the abundance of or eliminated most other minerals. Quartz and degraded white mica constitute most of the nonclay residue.
2. Solution was effective enough to remove most carbon and iron, producing a bright, white product.
3. Resilication altered large masses of bauxite to kaolin, much to the benefit of the pigment industry.

On the other hand

1. Not all deposits had enough quartz or mica removed by solution and are too sandy for current production standards.
2. Resilication under saline conditions permitted the development of montmorillonite, the major cause of non-fluid clay slurries.
3. More recent infiltration by ground water has carried in iron which has been subsequently oxidized or reduced, producing red, yellow and purple colors on the one hand and grays on the other.
4. Extensive beds of kaolin still remain buried too deeply to permit economic mining operations because of excessive overburden.

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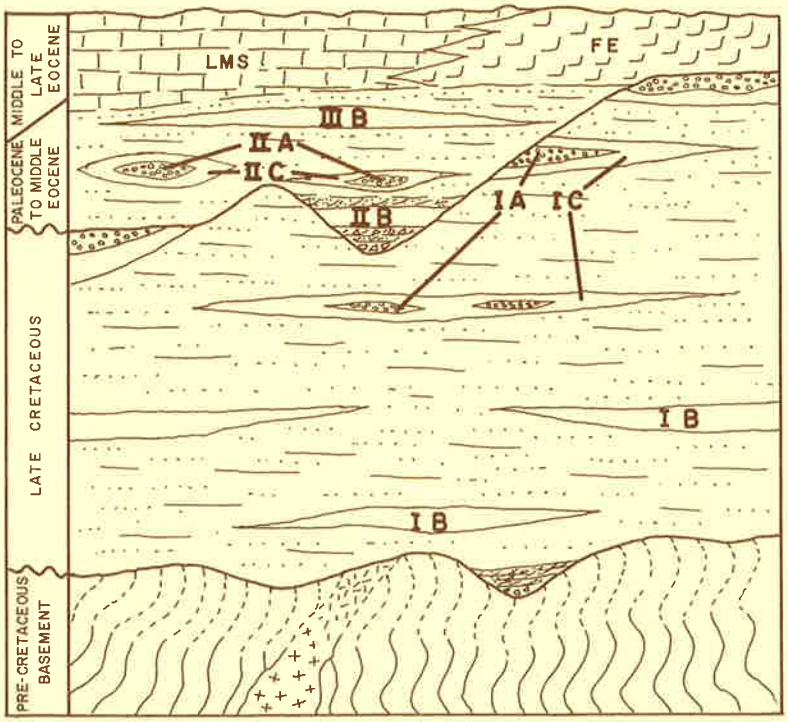

FIGURE 7 Generalized stratigraphic section in the middle Georgia kaolin region.

CRETACEOUS

TYPE I			
	A	Bauxite	Lateritic product derived
	B	Kaolin	Cretaceous sediments
	C	Kaolin	Alteration rim derived from bauxite.

TERTIARY

TYPE II			
	A	Bauxite	Lateritic product derived from transported altered Cretaceous materials.
	B	Kaolin	
	C	Kaolin	Alteration rim derived from bauxite.

TYPE III			
	A	Bauxite (rare)	True sedimentary deposits from I and II.
	B	Kaolin	

ALUMINA FROM DOMESTIC RESOURCES

Don H. Baker, Jr., Research Supervisor
U. S. Department of the Interior
Bureau of Mines
Boulder City Metallurgy Engineering Laboratory
Boulder City, Nevada

Introduction

At present, nearly all of the primary aluminum produced in the United States is obtained by the electroreduction of alumina extracted from bauxite by the Bayer process. Although world reserves of bauxite are adequate for many decades at the current rate of aluminum production, domestic reserves are limited, and as a result, the United States now relies on foreign suppliers for about 90 percent of its raw material. Global demand for bauxite is increasing rapidly, and this combined with production levies and the nationalization of foreign bauxite mining operations, has placed the United States in a position in which it faces additional increases in the cost of imported bauxite, as well as possible constraints on its availability.

There are, however, vast quantities of nonbauxitic aluminous minerals in the United States, such as clay, anorthosite, alunite, and dawsonite, which represent potential alternatives to bauxite. In the interest of establishing a domestic productive capability for alumina from substitute materials, the Bureau of Mines is pursuing a miniplant program at the Boulder City Metallurgy Engineering Laboratory. This program is designed to provide a comparative evaluation of the most promising technologies applicable to these materials.

J. E. Husted (1974)[1] estimated that there are about 9 billion tons of clay. His tabulation outlined only the significant clay deposits, most of which contain more than 50 million tons. Clay reserves that would be amenable to acid leaching (those containing 30 percent or more Al_2O_3 and less than 6 percent iron) amount to 7 billion tons, representing the equivalent of 1.24 billion tons of aluminum. This is sufficient aluminum to meet the domestic annual primary demand (industrial demand less secondary supply) for almost 200 years, based on the annual consumption in 1973 of 5.242 million tons (Bureau of Mines, 1975). Husted also indicated that there are massive (440 billion) tonnages of anorthosite with an average alumina content of about 26 percent, which is sufficient to supply the primary demand at the 1973 rate for more than 19,000 years.

In December 1970, the National Materials Advisory Board made a study entitled "Processes for Extracting Alumina From Nonbauxite Ores". The immediate recommendation of the report centered around nitric and hydrochloric acid processes to treat clay. Other resources were considered to have potential, but each had technical problems associated with the extraction of alumina which needed additional study. The plan was not implemented at that time. However, in July 1973, the Bureau of Mines instituted a six-point miniplant program to provide comparative engineering data for clay/HNO_3, clay/HCl, anorthosite, clay/H_2SO_3, alunite, and dawsonite processes outlined in the literature. The first-year funding was $400,000, and the program was scheduled to continue through September 1979.

The program was accelerated in July 1974, when eight aluminum companies joined in a cost-sharing effort, with each company contributing $50,000. The original cooperators were Aluminum Company of America, Aluminum Company of Canada Ltd., Alumax Inc., Anaconda Aluminum, CONALCO Inc., Kaiser Aluminum and Chemical Corp., Martin Marietta Aluminum Inc., and Reynolds Metals Co. Combustion Engineering Inc., and Vereinigte Aluminum-Werke AG of West Germany joined the group in the spring of 1975. Representatives from each of the 10 companies and the Bureau meet quarterly as a steering committee to coordinate and assist in directing and evaluating the miniplant program.

Accomplishments to date include nitric acid leaching of kaolinitic clay which has been tested in a miniplant, and about half of the unit processes involved in leaching clay with hydrochloric acid have been tested. Some of the observations and data developed in these efforts are reviewed in this report.

General Acid-Leaching Process

The clay raw material must be heated to a temperature sufficiently high to cause decomposition of the hydrated aluminum silicate and thereby make the alumina soluble in acid. This occurs between 700° and 800° C, at which temperature amorphous oxides of aluminum and silicon are formed. The calcined clay is then treated with acid, which dissolves the alumina as the aluminum salt of the acid but does not dissolve the silica. This salt is crystallized and separated from the mother liquor and is subsequently thermally decomposed to produce a pure alumina and the acid for reuse in the process. A generalized flowsheet for this process is presented in Figure 1.

Installation of the nitric acid miniplant was initiated in August 1973, and the plant was first operated as an integrated series of unit processes through the crystallization step in May 1974. Construction of the hydrochloric acid miniplant is still in

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[1] Numbers refer to items in the list of references.

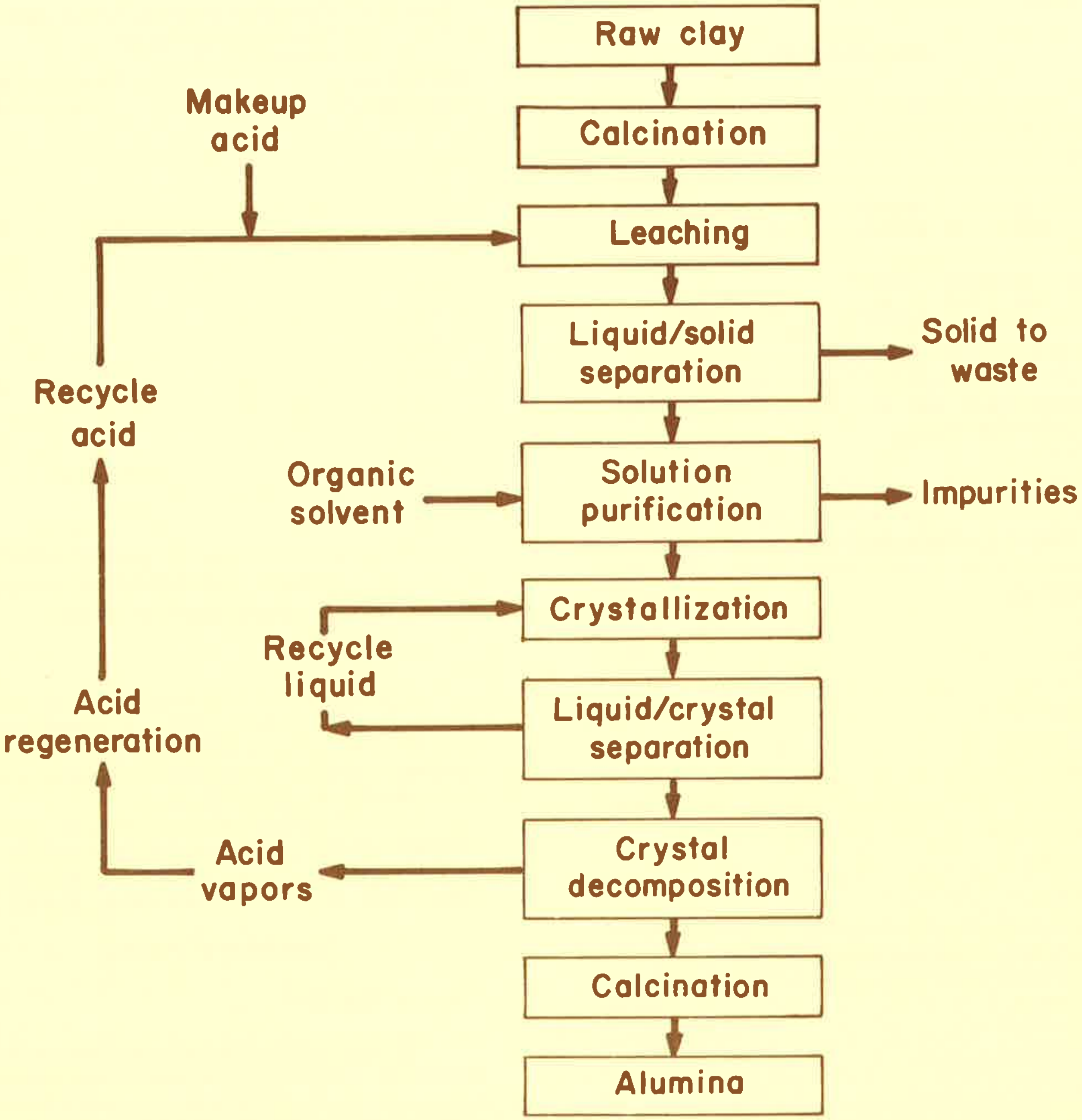

FIGURE 1 - Generalized acid-leaching-process flowsheet

progress, but major portions of the plant have been operated.

Raw Materials

Clay was obtained in 50-ton quantities of run-of-mine material through the courtesy of the Thiele Kaolin Company[2] of Sandersville, Ga. Petrographic analysis showed minor quartz fragments and feldspar (mostly potassium) and scattered accessory minerals in the clay matrix. Most opaques were extremely fine grained and probably kaolinite. A petrographic grain count is presented in Table 1.

Table 2 gives a typical chemical analysis of this clay after drying at 115° C. The analysis indicates the clay to be of high quality and reasonably low iron content. Data in Table 3 indicate the fineness of the dispersed clay as determined by the ASTM D-422-63 soil hydrometer method.

TABLE 1. — Petrographic grain count

Constituent	Estimated percent
Clay	96-97
Quartz	1
Feldspar	1-2
Epidote	Not detected
Fluorite	1
Zircon	Not detected
Organic matter	1
Muscovite	1
Opaque	1

TABLE 2. — Chemical analysis of dry clay

Component	Wt-pct
Al_2O_3	37.41
FE_2P_3	.85
Na_2O	.09
K_2O	.09
CaO	.09
MgO	.06
P_2O_5	.05
SiO_2	43.31
Inerts 1/	4.85
LOI	13.20

1/Obtained by difference, mostly TiO_2

TABLE 3. — Cumulative weight-percent of material finer than indicated size

Size fraction, micrometers	Cumulative weight, percent finer
105	99.6
74	98.9
74	97.8
37	97.6
16.1	91.0
16.4 1/	90.9
9.6 1/	88.9
6.8 1/	87.9
4.8 1/	86.9
3.5 1/	85.9
2.0 1/	84.8

1/Equivalent spherical diameter of particles determined using soil hydrometer method. Calculations were made using 2.62 as the specific gravity of the clay.

In the nitric acid miniplant, commercial-grade acid (42° Be) diluted to 50 percent concentration was used. Di (2-ethyl-hexyl) phosphoric acid (DEHPA) combined with kerosine comprised the organic phase in the solvent extraction (SX) circuit.

For the clay/hydrochloric acid miniplant, commercial-grade acid (20° BE) was used. Reagents used in the solvent extraction circuit consisted of kerozine, a tertiary amine (Alamine 336), and n-decyl alcohol.

Discussion of Process

Clay Preparation

The clay crushing and calcination sections were common for both the nitric and hydrochloric acid miniplants. The bulk clay was crushed to pass a 2-mesh screen and the minus 2- plus 20-mesh fraction was fed to a countercurrent gas-fired kiln. The minus 20-mesh fraction was pelletized on a disc pelletizer, and the 3/8-inch wet pellets also were fed to the kiln.

The kiln was 15 inches inside diameter by 21 feet long. Calcination temperature was maintained by an automatic gas burner controlled by the temperature of the clay as it entered the enlarged section of the combustion zone. Calcined clay, the analysis of which is shown in Table 4, was crushed to a minus 10 mesh for feeding to the leach circuit of the miniplant.

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[2] Reference to specific companies or brands is for identification only and does not imply endorsement by the Bureau of Mines.

TABLE 4. – Chemical Analysis of Calcined Clay

Al_2O_3, total	43.10
Acid-extractable Al_2O_3	[1]40.81
FE_2O_3	.98
Na_2O	.10
CaO	.10
MgO	.07
P_2O_5	.06
SiO_2	49.90
Inerts	5.59
Total	99.90

[1] Extractable under the condition employed in the miniplant.

Nitric Acid Process

The leach section of the nitric acid miniplant consisted of a stirred cold-mix pot where cold calcined clay was slurried with nitric acid before entering a series of three stirred reactors. The slurry produced in the cold-mix pot overflowed into the first reactor, a jacketed vessel constructed of Carpenter 20 stainless steel and heated with a circulating heat transfer fluid to maintain a minimum temperature of 100° C. The vent of the reactor was fitted with a water-cooled condenser to return any acid vapor to the reaction chamber. The reaction, $A1_2O_3 + 6HNO_3 \rightarrow 2Al(NO_3)_3 + 3H_2O$, represents the leaching step. The slurry leaving the first reactor overflowed and cascaded through two additional insulated, stirred 316 stainless steel reactors to complete the leach section.

The discharge from the third reactor flowed into a drag classifier that separated the coarse sand residue from the slurry. The sand discharge after one washing was stored for further testing. The pool overflow from the No. 1 classifier went to a 3- by 3-ft-diameter thickener. The thickener underflow was stored for future testing, and the clear overflow raw, pregnant solution was filtered in a 12-sq-ft plate and frame filter to remove suspended solids. This yielded a polished pregnant solution from which the iron was removed by solvent extraction.

To remove the iron, the polished pregnant solution was heated to 40° C and treated with organic extractant in three cells of an 11-cell, countercurrent, mixer-settler system. The organic phase consisted of 59 percent kerosine, and 41 percent DEHPA. The iron-laden organic phase was washed with water in the fourth and fifth cells prior to being contacted with 17 percent HCl in the sixth through ninth cells to strip the iron. The organic solvent was washed free of chloride in the 10th and 11th cells before being recycled.

The purified pregnant solution was pumped to an evaporator operated at 120° C, in which the concentration of the aluminum nitrate solution was increased by evaporating some of the water. The concentrated solution was then cooled to about 55° C, which resulted in the crystallization of aluminum nitrate nonahydrate. A drag classifier was used to separate the crystals from the solution, and the mother liquor was recycled to the evaporator and crystallizer. The drained crystals were stored for subsequent use in studies involving thermal decomposition to produce alumina.

Thermal decomposition of the aluminum nitrate nonahydrate was accomplished by several methods including a direct-fired, fluidized-bed roaster and a heated-tube roaster. Decomposition produced nitric acid, oxides of nitrogen, and hydrated alumina. The hydrated alumina was then fired at 1,050° C to produce alumina that was about 50 percent in the alpha form, the crystalline form desired by part of the aluminum smelter industry. Most of the decomposition of the $Al(NO_3)_3 \bullet 9H_2O$ took place at about 400° C according to the following equation:

$$Al(NO_3)_3 \bullet 9H_2O_{(s)} \rightarrow 1/2\ Al_2O_{3(s)} + 7\ 1/2\ H_2O_{(g)} + 3HNO_{3(g)}.$$

The miniplant evaluation of the nitric acid extraction of alumina from clay shows it to be a technically viable process. Data from these operations, supplemented by laboratory tests, have shown that 97.8 percent of the extractable alumina can be recovered. This represents 93.0 percent of the total alumina in the clay tested. The acid loss, which is about 7 percent, results primarily from nitrates left in the trailings and from oxides of nitrogen formed during crystal decomposition, which were not recovered by scrubbing.

Theoretical energy data summarized in Table 5 show that, for the conditions listed, there is a theoretical heat requirement of 651,000 British thermal units (Btu) to produce 40.4 pounds of Al_2O_3. This is equivalent to 32.2×10^6 Btu/ton of Al_2O_3 produced. Consideration of the theoretical energy requirements serves two functions: (1) It provides a basis on which various processing techniques can be compared, and (2) it points out the energy intensive process steps where efforts to minimize energy requirements should be made by either modification of the processing steps or by recovering or saving process energy.

Hydrochloric Acid Process

The flowsheet for the hydrochloric acid leach process for the treatment of calcined clay is very similar to that presented in Figure 1. The process differs from the nitric acid technique in that the crystallization of the aluminum acid salt in the nitric acid process is temperature dependent because the crystals melt at 73° C in their water of crystallization, whereas in the HCl process, crystallization is concentration de-

TABLE 5 — Theoretical energy data*, nitric acid miniplant

Process step	Flowsheet materials, lb	Theoretical energy requirements, Btu	
		Needed	Available reclaim
Calcination	143.7, raw clay	118,000	—
	100, calcined clay	—	35,000
	43.7, H_2O	—	74,000
Leaching	324.3, 50 percent HNO_3	—	—
	100, calcined clay	—	—
	424.3, slurry	—	45,000
Liquid-solid separation	424.3, slurry	—	—
	131.3, wash water	—	—
	336.2, pregnant solution	—	—
	115.6, sands	—	2,000
Solvent extraction	440.2, pregnant solution	—	—
	437.0, purified solution	—	—
	2.0, $FeCl_3$		
Evaporation/ crystallization	437.0, purified solution	195,000	—
	134.9, vapors	—	148,000
	311.8, crystals	—	66,000
	5.0, bleed stream	—	—
Decomposition	311.8, crystals	338,000	—
	271.4, acid vapors	—	252,000
	40.4, Al_2O_3	—	22,000
Acid recovery misc.			3,000
Total	40.8 lbs Al_2O_3 in		
	40.4 lbs Al_2O_3 out	651,000	

Energy needed = 651,000 Btu per 40.4 lb Al_2O_3 produced or 32.2 x 10^6 Btu/ton alumina produced.

*For ease of comparison these data and the data presented later for the clay/HCl process are based on a feed rate to the leach section of 100 pounds of calcined clay per hour.

pendent. There are two ways of effecting crystallization of $AlCl_3 \bullet 6H_2$. One is through evaporation to increase the concentration to saturation thereby causing crystallization. The second, referred to as gas sparging, utilizes the addition of hydrogen chloride gas to raise the acid concentration in order to depress the solubility of aluminum chloride, thus resulting in crystallization.

The hydrochloric acid miniplant equipment was fabricated from fiberglass and plastics or metal coated with glass, rubber, and/or Plastisol.

The leach section equipment consisted of a cold-mix chamber in which a clay/acid slurry was formed, four jacketed, glass-lined Pfaudler reactors, and a heat exchanger. The coarse waste in the slurry discharge from this section was separated and washed in two spiral classifiers while the slimes (or fines) were separated from the pregnant solution in two 5- by 5-feet thickeners. Final polishing of suspended solid from the solution was accomplished by a plate and frame filter prior to treating the solution by solvent extraction for the removal of iron. The polished pregnant solution is contacted with an organic phase, consisting of 16-percent Alamine 336 stabilized with 11 percent

TABLE 6. – Theoretical energy data, hydrochloric acid miniplant sparging process

Process step	Flowsheet materials, lb	Theoretical energy requirements, Btu: Needed	Theoretical energy requirements, Btu: Available for reclaim
Calcination	143.7, raw clay	118,000	–
	100, calcined clay	–	35,000
	43.7, H_2O	–	74,000
Leaching	361.0, 26% HCl	–	
	100, calcined clay	–	–
	461.0, slurry	–	13,000
Liquid-solid separation	461.0, slurry	–	–
	57.8, wash water	–	–
	403.2, pregnant solution	–	–
	115.6, sands	–	8,000
Solvent extraction	403.2, pregnant solution	–	–
	401.2, purified solution	–	–
	2.0, $FeCl_3$	–	–
Sparging crystallization	401.2, purified solution	–	68,000
	106.3, HCl gas	–	–
	191.1, crystals	–	7,000
	20.5, bleed stream	–	2,000
	295.9, recoverable acid 1/	–	30,000
Decomposition	191.1, crystals	221,000	–
	150.7, acid vapors	–	153,000
	40.4, Al_2O_3	–	22,000
Acid recovery	Miscellaneous	12,000	26,000
Total	40.8 Al_2O_3 in		
	40.4 Al_2O_3 out	351,000	

Energy needed = 351,000 Btu per 40.4 pounds Al_2O_3 or 17.4 x 10^6 Btu/ton of Al_2O_3 produced by sparging.

1/Estimated magnitude for cooling acid to base temperature prior to recovery system.

n-decyl alcohol in 73-percent kerosine, in two mixer-settler cells to remove the iron. The iron in the loaded organic phase is stripped with 3-percent HCl in three additional cells before the organic phase is recycled.

The leaching, liquid-solid separation, and solvent extraction sections of the hydrochloric acid miniplant have been operated as an integrated continuous process. The purified aluminum chloride solution produced has been stored for crystallization studies by both the evaporation and sparging techniques.

A review of the basic thermochemical data for the HCl process, incorporating the sparging option, is presented in Table 6. Table 7 presents thermochemical data specific to the evaporation/crystallization option. These data show that the production of 40.4 pounds of Al_2O_3 by the sparging crystallization tech-

TABLE 7. — Theoretical Energy Data for Evaporation/Crystallization Option

Process step	Flowsheet materials, lb.	Theoretical energy requirements, Btu	
		Needed	Available for reclaim
Calcination	143.7, raw clay	118,000	—
	100, calcined clay	—	35,000
	43.7, H_2O	—	74,000
Leaching	361.0, 26% HCl	—	—
	100, calcined clay	—	—
	461.0, slurry	—	13,000
Liquid-solid separation	461.0, slurry	—	—
	57.8, wash water	—	—
	403.2, pregnant solution	—	—
	115.6, sands	—	8,000
Solvent extraction	403.2, pregnant solution	—	—
	401.2, purified solution	—	—
	2.0, $FeCl_3$	—	—
Evaporation crystallization (double effect)	401.2, purified solution	195,000	—
	105.1, crystals	—	4,000
	258.2, vapors	—	120,000
	86.0, crystals	—	2,000
	87.0, vapors	—	96,000
	8.9, bleed stream	—	700
	7.5, bleed stream	—	350
Decomposition	191.1, crystals	221,000	—
	150.7, acid vapors	—	153,000
	40.4, Al_2O_3	—	22,000
Acid recovery	Miscellaneous	—	25,000
Total	40.8 Al_2O_3 in 40.4 Al_2O_3 out		

Energy needed = 534,000 Btu per 40.4 pounds Al_2O_3 or 26.5 x 10^6 Btu/ton of Al_2O_3 produced.

nique would have a theoretical energy requirement of 351,000 Btu. This is equivalent to 17.4 x 10^6 Btu/ton of Al_2O_3 produced. By comparison, to produce 40.4 pounds of Al_2O_3 by the evaporation/crystallization option would require 534,000 Btu or 26.5 x 10^6 Btu/ton of Al_2O_3 produced. In addition, the sparging technique offers the greatest potential for energy recovery, followed by the evaporation/crystallization technique, and then the nitric acid process.

The alumina industry today is principally based upon the treatment of bauxite by the Bayer process. Therefore, a similar theoretical energy tabulation has been developed for the Bayer process (Table 8), which shows an energy requirement of 28.4 x 10^6 Btu/ton of Al_2O_3. However, the processing generates considerable steam, which theoretically can contribute up to 1,087,000 Btu per 100 pounds of Al_2O_3 produced or 21.6 x 10^6 Btu/ton of Al_2O_3. This would reduce the energy needed to 7.87 x 10^6 Btu/ton of Al_2O_3 if all the steam were used. Since the steam cannot be recovered completely, it is felt that a more realistic comparative value would be about 10 x 10^6 Btu/ton of Al_2O_3.

TABLE 8. — Theoretical Energy Data for Bayer Plant

Process step	Flowsheet materials, lb.	Theoretical energy requirements, Btu: Needed	Theoretical energy requirements, Btu: Available for reclaim
Digestion	2095.3 caustic	386,000	—
	258.2 bauxite	—	—
	3.9 impure lime	—	—
	218.6 H_2O	—	—
	2134.9 digest liquor	—	—
	351.1 steam	—	–397,000
Washing and separation	835.4 wash water	—	—
	1234.9 digest liquor	113,000	—
	2772.0 green liquor	—	—
	198.3 red mud	—	–17,000
Precipitation of gibbsite	2772.0 green liquor	—	–106,000
	101.0 wash water	—	—
	2697.8 mother liquor	—	—
	175.2 gibbsite	—	–12,000
Mother liquor recovery and evaporation	2697.8 mother liquor	748,000	—
	7.5 soda ash	—	—
	2095.3 caustic to digestion	—	—
	610.0 water vapor	—	–682,000
Gibbsite decomposition	175.2 gibbsite	215,000	—
	79.2 stack loss	—	—
	101.0 impure Al_2O_3	—	–202,000
Limestone calcination	1.0 limestone	10,000	—
	3.5 stack loss	—	—
	3.9 lime	—	–4,000
Al_2O_3 produced	100.0	1,472,000	—

Energy needed @ 1,472,000 Btu per 100 pounds Al_2O_3 or 29.4 x 10^6 Btu/ton of Al_2O_3 produced.

Conclusions

The production of alumina from clay by either of the acid-leaching processes outlined appears to be technically feasible. Two options are available for effecting crystallization within the hydrochloric acid process. Crystallization by evaporation is endothermic, whereas the gas-sparging method is exothermic, and therefore is less energy intensive. A review of the theoretical energy requirements of each, without regard for engineering energy efficiencies of process steps or the quantity of the energy that is practicable to recover, indicates that the hydrochloric acid process using the gas-sparging option is the most promising. From a standpoint of theoretical energy requirements, the HNO_3 process is less promising than either of the HCl process options, primarily because 3 more moles of water must be removed during the decomposition step.

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GEOLOGICAL CLASSIFICATION AND EVALUATION OF HEAVY MINERAL DEPOSITS

Thomas E. Garner. Jr.
E. I. duPont de Nemours and Company
Lawtey, Florida

Introduction

Heavy minerals have been important to the world over the past 50 years, and increasing demand for them makes it essential that new deposits be discovered and developed. Traditionally, heavy minerals have been defined as discrete, liberated, sand-sized mineral particles with specific gravities over 3.2. They have generally been classed as beach sand deposits because of their association with shore line activity. The heavy mineral suite of most economic deposits consists of the more stable minerals such as ilmenite, rutile, zircon, staurolite, garnet, epidote, aluminum silicates, etc.

The first beach sand mining operation in this country was located at "Mineral City" on the Atlantic coast just south of Jacksonville Beach, Florida. During World War I, rutile was mined for use in productING titanium tetrachloride for tracer bullets. Following the war, this operation became inactive and the site is now the Ponte Vedra Country Club. There was very little activity in heavy minerals during the intervening years until World War II. The Humphreys Mining Company operated a heavy minerals mine just east of Arlington near Jacksonville from 1944 through 1964. Hobart Brothers operated a mine near Vero Beach to produce rutile for their welding rods during the 1946-1963 period. Both of these are inactive at the present time.

When the Arlington deposit was mined out, Humphreys moved their equipment to Folkston, Georgia, to mine a deposit owned by E. I. du Pont de Nemours & Company. In 1974, the Folkston deposit was mined out and Humphreys moved their wet mill across the St. Marys River to mine a deposit just west of Boulogne, Florida. The wet mill concentrates are trucked to their dry mill at Folkston where they are processed into titanium minerals, zircon, and monazite.

A deposit at the south end of Amelia Island owned by Union Carbide was scheduled for mining in the late 1950's. Plans to mine it were abandoned, and the property was sold in 1970 for development as a recreational area.

Three other heavy mineral mines operate in Florida: two Du Pont mines at Starke and Lawtey; and one just south of Green Cove Springs near Penny Farms which is operated by Titanium Enterprise, a joint venture between Union Camp Corporation and American Cyanamid. In addition, heavy mineral mines are operated for ilmenite in New Jersey by the Glidden Company and the American Smelting and Refining Company (ASARCO). The output from ASARCO's plant is purchased by E. I. du Pont de Nemours & Company under long term contract. Other known heavy mineral occurrences include several small deposits in southeast Georgia, northeast Florida and two large deposits in Tennessee. The Tennessee deposits are owned by Ethyl Corporation and Kerr-McGee. None of these are in operation at the present time.

The purpose of this paper is to review geologic history of heavy mineral deposits and discuss economic evaluation of heavy minerals contained therein.

Titanium Minerals

About 95 percent of the United States ilmenite, leucoxene, and rutile production is used to manufacture titanium dioxide white pigments. The remaining 5 percent is used to manufacture titanium metal. The two processes by which white pigments are manufactured from ilmenite and rutile are described by Lynd and Lefond (1975). In general, the higher the $Ti0_2$ content in titanium minerals, the more value they have to pigments manufacturers.

In the case of sulfate pigments manufacture, the degree of solubility in sulfuric acid is an important consideration. Ilmenite and altered ilmenite are soluble and can be used successfully in sulfate pigment manufacture. Rutile and high $Ti0_2$ leucoxene cannot be used in sulfate pigment manufacturing because they are insoluble in sulfuric acid.

E. I. du Pont de Nemours & Company pioneered the chloride process which can be used successfully with all titanium minerals. Because of the superior pigment quality produced by chlorination of titanium minerals and because there are less pollutants produced by their process, Du Pont has phased out all of their sulfate plants and produces pigments exclusively by chlorination.

Classification of Deposits

Traditionally, titanium mineral deposits have been regarded as either rock or sand deposits (Lynd and Leford, 1975). Hard rock deposits occur throughout the world producing ilmenite for use in manufacturing white $Ti0_2$ pigments. Ilmenite from the Adirondack deposit is used in manufacturing $Ti0_2$ pigment by the sulfate process. Ilmenite produced from the Allard Lake deposit is shipped to a smelter at Sorrel, Quebec, where a titanium-rich slag is produced for use in sulfate pigment manufacture.

TABLE I – Magnetic Opaque Minerals

Mineral	Composition	Sp. G.	Color	Distinguishing Optical and Physical Properties	Use
Magnetite	Fe_3O_4	5.2	Black	Ferromagnetic	Ore of iron
Chromite	$FeCr_2O_4$	4.6	Black	Chemical analysis may be required to distinguish from ilmenite	Ore of chromium; foundry sand
Ilmenite	$FeO \bullet TiO_2$ $Fe_2O_3 \bullet 3TiO_2$	4.7 4.3	Black to dark reddish brown	Chemical analysis may be required to identify	TiO_2 pigment manufacture $TiCl_4$
Leucoxene	$Fe_2O_3 \bullet 4TiO_2$	3.9	Dark red, tan, yellow, white	Light color, waxy luster, some grains translucent	Same as ilmenite
Magnetic Non-Opaque Minerals					
Monazite	(Ce, La, Y, Th) PO_4	5.2	Yellow to colorless	High relief and index of refraction	Rare earth and thorium ore
Xenotime	Y PO_4	5.1	Yellow to colorless	High relief and index of refraction; high birefringence; relief changes noticeably on rotation of 90°	Rare earth ore
Staurolite	$FeAl_4Si_2O_{10}(OH)_2$	3.8	Red	Color and relief	Portland cement manufacture; sandblast abrasive
Garnet	Complex FeMg Al, silicate	3.8	Red	Isotropic - red color and relief	Abrasive
Spinel	$ZnAl_2O_4$	3.6	Green to colorless	Isotropic-green color and relief	No commercial value
Epidote	Ca_2 (Al, Fe, Mn)$_3$ (OH) Si_3O_{12}	3.4	Green	Green color, particle shape and birefringence	No commercial value
Tourmaline	complex borosilicate	3.1	Pink to black	Low relief, index just below oil, pleochoism	No commercial value
Mica (Biotite)	Complex aluminum silicate of K, Mg and Fe	3.0	Black to golden	Uniaxial figure on basal plates	No commercial value
Non-Magnetic Opaque Minerals					
Gold	Au	17	Gold (R)	Soft, malleable, high sp. g.	Precious metal ore
Pyrite	FeS_2	5	Brass yellow	Cubic crystal habit	Ore of sulfur

con't.

TABLE 1 con't.

Non-Magnetic Translucent Minerals

Mineral	Composition	Sp. G.	Color	Distinguishing Optical and Physical Characteristics	Use
Zircon	$ZrSiO_4$	4.3	Colorless to pink	High index of refraction and relief - shape	Foundry sand, refractory, zirconium metal ore
Rutile	TiO_2	4.2	Red	High relief and index of refraction, birefringent with no interference - colors	TiO_2 pigment manufacture, welding rod flux
Corundum	Al_2O_3	4.0	Blue to colorless	High index of refraction, relief and irregular shape distinguishes from kyanite	Refractory
Anatase	TiO_2	3.9	Colorless to blue	Rectangular grain shape in sand deposits	Potential TiO_2 pigment raw material
Kyanite	Al_2SiO_5	3.6	Colorless to blue	Grain shape and cleavage	Refractory
Diamond	C	3.5	Colorless	Octahedral shape - high index of refraction	Abrasive
Sillimanite	Al_2SiO_5	3.2	Colorless	Matches in bromonapthylene cleavage	Refractory
Apatite	$Ca_5(F,Cl)(PO_4)_3$	3.2	Colorless, brown, black	Index of refraction below liquid. Collophane is amorphous	Fertilizer mineral

Ilmenite from hard rock source is less attractive than sand ilmenite because $Ti0_2$ content is normally too low for economic use in chloride pigment manufacture. An idealized depositional sequence for titanium mineral deposits is shown in Figure 1.

Altered Hard Rock Deposits

Titanium bearing hard rock deposits may undergo extensive alteration, producing new species of titanium minerals substantially different in character from those in the original rock. For example, Brazilian jacupirangites have undergone extensive alteration producing anatase and ilmenite at the expense of the original perovskite. Large octahedral crystals of secondary anatase may be found in the tropically weathered carbonatite near Tapira in Minas Geraes, Brazil. Anatase concentrates assaying up to 86 percent $Ti0_2$ can be produced from these deposits, using physical and chemical methods (Lynd and Leford, 1975).

Residual Deposits

Hard rock deposits may be broken down physically by weathering processes without alteration of the ilmenite. In many cases, weathering produces ores which can easily be mined without blasting and do not require extensive grinding, since most of the ore minerals have been liberated. These deposits can be classed as "residual" deposits if the minerals have not been transported. Unless there has been secondary alteration, the $Ti0_2$ content in the ilmenite is still relatively low (45 – 48 percent), thus making it unattractive to many pigments manufacturers at the present time.

Sedimentary Deposits

If minerals derived from weathered hard rock sources are transported, they may then be classed as sedimentary deposits. These may be sub-divided into continental alluvial, deltaic, and marine deposits.

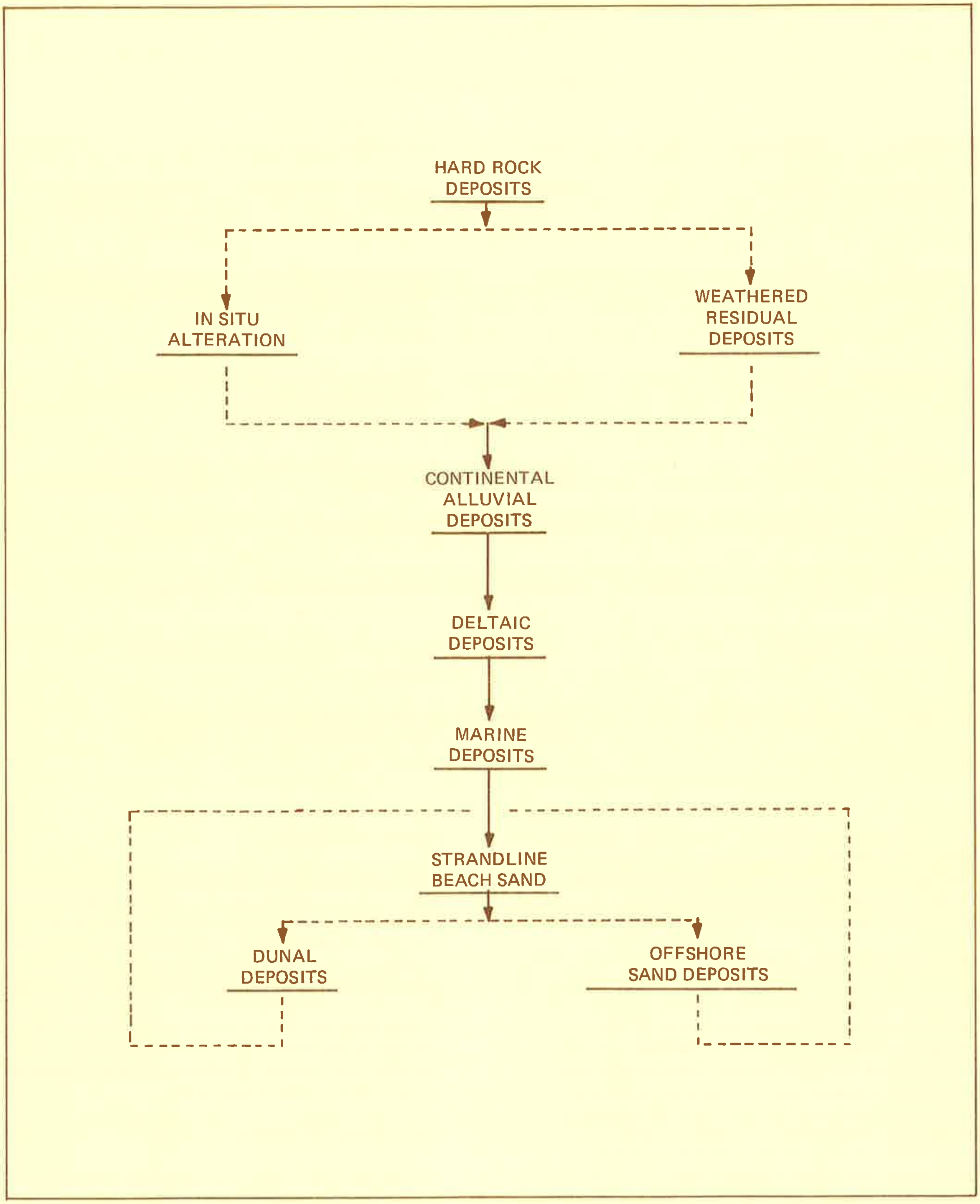


FIGURE 1 – Idealized depositional sequence for titanium mineral deposits

a) Continental Alluvial Deposits:

The liberated and weathered mineral particles may be transported and redeposited by means of rain water run-off and streams. During this process the ilmenite grains may undergo alteration through oxidation and leaching to upgrade the TiO_2 to 60 percent or higher (Garner, 1971 and Temple, 1966). Ilmenite from these deposits may be dull, irregular shaped fragments with a "shaly" structure. Such a grain is shown in Figure 2. The shaly structure may be due to exsolution or to basal parting often noted in ilmenite. Because of this structure, ilmenite from some continental alluvial deposits tends to break down physically producing excessive fines during processing. Alluvial deposits have the advantage that most of the mineral particles are completely liberated sand-sized grains and can be mined using inexpensive methods without crushing and grinding. Continental alluvial deposits are first associated with flood plans. Further reworking and transport downstream may lead to deposition in deltaic sediments. These deposits may be classed as "fresh water deltaic".

b) Marine Deposits:

Heavy minerals from continental alluvial or deltaic mineral deposits may be carried to the ocean by rivers and streams where they are deposited in deltas and along coastlines. Reworking of deltaic deposits by waves and tidal currents may redistribute the sand along coastlines.

In some cases, grains may be washed seaward and deposited on the sea floor contiguous to shore lines. The potential importance of these offshore deposits has been recognized and work is underway to investigate the concentration of heavy minerals off the southeast Atlantic coast and in the norther Gulf coast areas.

Heavy mineral grains may reside for eons in any of the environments discussed, then be reworked and redeposited many times before being concentrated into commercial deposits. During this process the ilmenite and other grains frequently become rounded with highly polished grain surfaces. Ilmenite particles from the Trail Ridge deposit shown in Figure 3 are typical. For example, heavy mineral bearing Citronelle sands of Miocene age are believed to have been reworked during the Pleistocene (Pirkle and Yoho, 1970). Heavy mineral content of the Citronelle is too low to be considered economic by present day standards. However, during the formation of Trail Ridge, the narrow feature formed at the north end of the Lake Wales erosion remnant, these heavy minerals were redeposited and concentrated

FIGURE 2 - **Electron micrograph showing ilmenite grain from continental deposit. Note "shaly" character. Grain is about 115 microns across shorter dimension. (ASARCO, Lakehurst, New Jersey).**

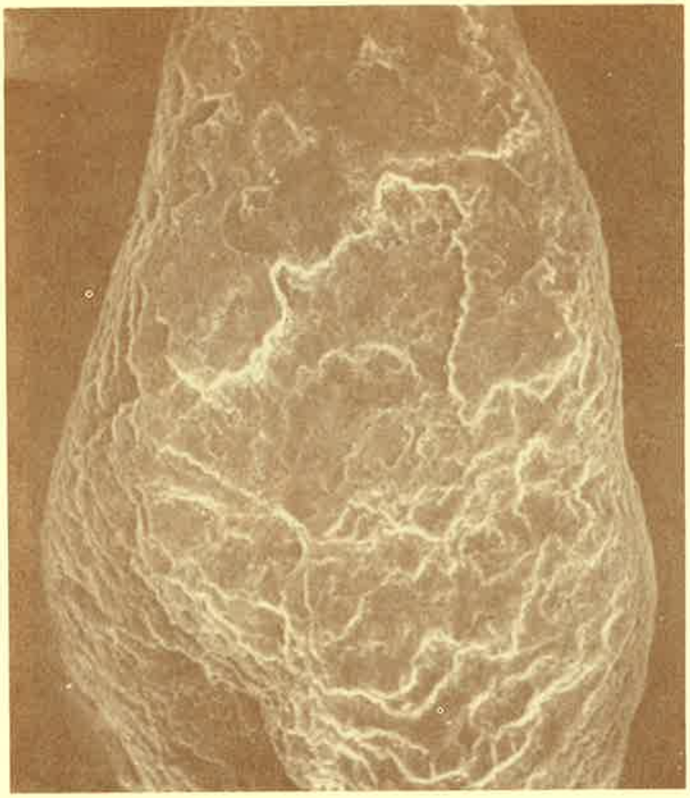
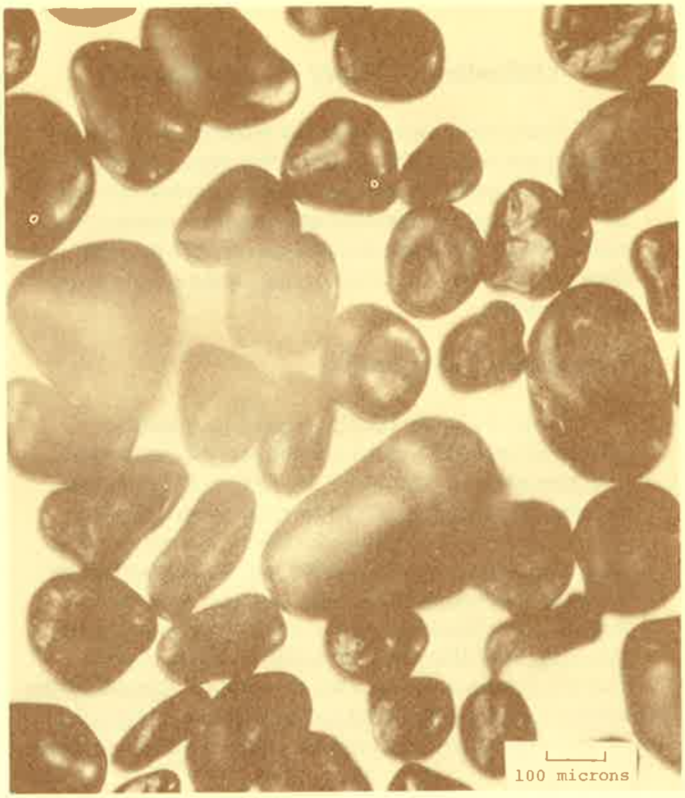


FIGURE 3 - **Photomicrograph of ilmenite grains from marine shoreline-dunal deposit. Note grain roundness and surface polish. (E. I. duPont de Nemours & Co., Inc. Trail Ridge Plant, Starke, Florida).**



to 3 — 4 percent heavy mineral in the deposit. Early scientists believed the cross-bedding features noted in the Trail Ridge heavy mineral deposit were foreset bedding of a huge delta. More recent studies indicate the angles of the cross-bedding are those of coastal dunes. The cross-bedding is shown in Figure 4.

Marine heavy mineral beach sands have the advantage that grains are completely liberated, and in many cases, extensive alteration through oxidation and leaching has upgraded ilmenite particles to as high as 90 percent TiO_2. The unconsolidated mineral particles in beach sands are easily mined by low cost methods (dredging, draglining, pan scraper, bucketwheel excavators, etc.), with essentially no crushing or grinding required to liberate mineral particles.

Heavy Mineral Silicates

The heavy mineral assemblage in a deposit depends entirely on the nature of the source rocks. The New Jersey heavy minerals are made up of 80 percent ilmenite with very little zircon (MacKewicz, 1969). Other deposits contain a variety of silicate heavy minerals which were deposited with the titanium minerals. Most of the unstable minerals weather out leaving the more resitant minerals. Younger continental deposits often contain hornblende, augite, and other pyriboles. Older marine beach sand deposits normally shown only traces of pyriboles. The more stable beach sand minerals include epidote, garnet, staurolite, tourmaline, zircon, kyanite, sillimanite, andalusite, corundum, spinel, monazite, and xenotime. Some of these have commercial value, whereas others do not. Heavy mineral uses are shown in Table 1.

Evaluation of Heavy Mineral Samples

Exploration techniques for finding heavy mineral deposits are outside the scope of this paper. However, a general outline for determining heavy mineral composition and economic potential of prospect samples is discussed below. No single method can always be used since samples may be different depending on the origin of the deposit.

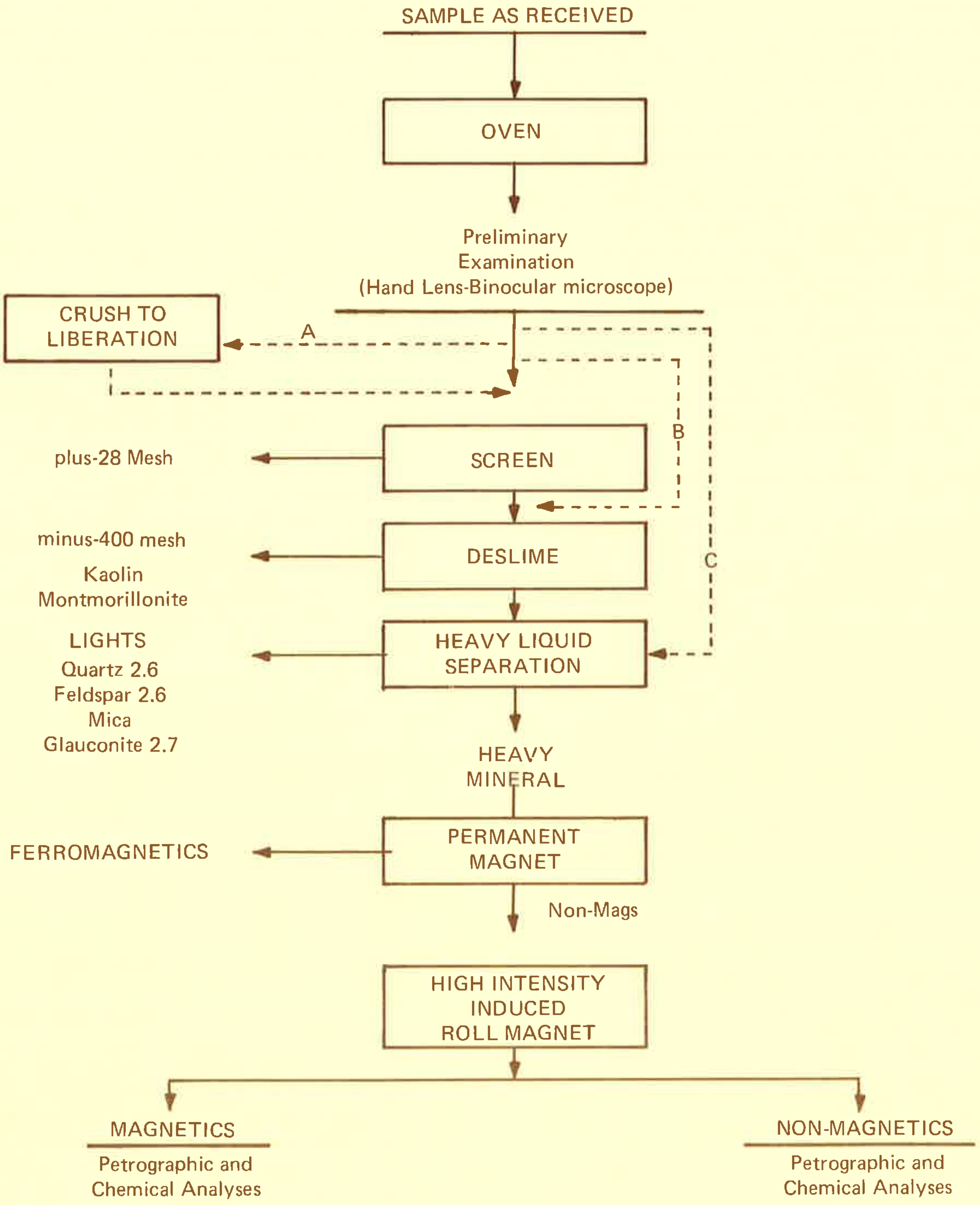
Preliminary Examination

After drying at 100°C, samples received for evaluation should be given a preliminary examination to determine best laboratory procedure. If the sample contains aggregates it should be crushed to liberation size. If the sample contains plus-28 mesh material, this should be removed by dry screening, the percentage determined and recorded for later use. If the sample is granular, free flowing, without significant amounts of clay, the evaluation continues with a heavy mineral determination using heavy liquids discussed below. If the sample contains significant amounts of caly, it should be slurried and wet screened on a 325 or 400 mesh sieve. Because of the very fine wire used in 400 mesh sieves, it is extremely important that care be taken not to break the wires while screening. Small amounts of slurry (no more than 300 grams of sand at a time) are introduced onto the 400 mesh screen. Water is used to wash the fines through the screen. Gentle rubbing in circular motion on the bottom of the screen may facilitate screening. The plus-400 mesh is transferred to a pan, dewatered, dried, then weighed, and the percent of sample determined.

Heavy Mineral Separation

The minus-28 plus-400 mesh fraction should be riffled into a 100 gram sample using a Jones type splitter. The sample is poured into a pear-shaped, 500 ml separatory funnel filled with acetylene tetrabromide which has specific gravity 2.96. If the minerals are extremely fine, bromoform (specific gravity 2.87) may be necessary for complete heavy mineral recovery. The sample and heavy liquid are stirred together using a stirring rod and the heavy minerals allowed to settle. The separatory funnel is agitated and stirred to make sure that all heavy minerals which may have been entrapped in the float are released and allowed to sink. This is repeated several times. The sink fraction is drawn

FIGURE 4 - Truncated dune bedding in marine shore-dunal deposit. Section is about 20 feet. Heavy minerals are more abundant in the cross-bedded portion. (E. I. duPont de Nemours & Co., Inc., Trail Ridge Plant, Starke, Florida).



A — Use if rocks or aggregates present.
B — Use if no coarse particles noted.
C — Use if no clays or coarse particles in sample.

FIGURE 5 — General outline for laboratory evaluation of heavy mineral prospect samples.

off into a funnel lined with Whatman #4 filter paper. The separatory funnel is again stirred and the heavy minerals allowed to settle to the bottom. The second crop is drawn off after the solution between the heavy minerals has cleared. (If the sample contains collophane or other minerals with specific gravity near that of the liquid, they may suspend.) This procedure is continued until no more heavy minerals are detected. The float is collected in a separate funnel lined with filter paper. After the heavy liquid is recovered, sinks and floats are washed using either alcohol or Solox®. (Solox is a trade name of U. S. Industrial Chemicals, Inc.) The sinks and floats are rewashed several times to completely remove all of the heavy liquid. Both products are then dried in an oven at a temperature of 100°C, weighed, and their percentages calculated. It should be noted that both acetylene tetrabromide and bromoform are toxic. Care should be taken to avoid skin contact and the separation should be carried out under a hood.

Identification and Evaluation of Heavy Minerals

Heavy mineral content of a sample may range from 0 to 100 percent. The value of the heavy minerals depends on their mineral composition. Mineral composition may be determined by petrographic analysis using bromonapthalene or other oil with index of refraction of 1.66 as an immersion fluid. This analysis may be supplemented with chemical analysis, x-ray diffraction, scanning electron microscope, electron probe, etc. If the sample contains minerals of potential economic value such as ilmenite, leucoxene, rutile, staurolite, zircon, monazite, xenotime, garnet, further evaluation is warranted. If the heavy minerals contained only non-economic heavy minerals with no value such as hornblende, augite, epidote, further examination is probably unwarranted.

Detailed Laboratory Analysis of Heavy Minerals

If the optical examination shows significant amounts of opaques (potential titanium minerals), zircon, staurolite, monazite, etc.), further laboratory analysis should be undertaken using the method outlined in Figure 5.

Ferromagnetic Minerals

Magnetic separation using a permanent horseshoe magnet removes magnetite and titaniferous magnetite. Silicates with magnetic inclusions may also be removed. Magnetite and titaniferous magnetite both have potential markets. The magnetics are weighed and recorded as percent of heavy minerals.

Paramagnetic Minerals

The non-ferromagnetic fraction may be further separated into paramagnetic and non-magnetic using high intensity laboratory magnet. Products from this separation should be examined petrographically to determine composition. The paramagnetic fraction may be made up of the opaque or non-opaque minerals shown in Table 1. The paramagnetic opaques may be high grade TiO_2 ilmenite type minerals. To determine this, analyze the fraction for TiO_2, FeO, and Fe_2O_3 and estimate TiO_2 in ilmenite as shown below:

$$\frac{\%TiO_2 x95}{TiO_2+FeO+Fe_2O_3} = \text{Estimated } TiO_2 \text{ in ilmenite}$$

If the estimated TiO_2 is less than 55 percent, it has only marginal value in the present day market. If it is 60 percent or higher, it has good potential value.

Staurolite by itself has potential economic value as a sand blast abrasive or constituent in Portland cement manufacture. In the presence of low Al_2O_3 garnet and epidote, staurolite has not value to the cement industry which specifies 45 percent Al_2O_3 or more. On the other hand, garnet by itself has potential value as an abrasive. In the presence of epidote, this value is minimized. At the present time epidote alone has no market value. Monazite is used to manufacture rare earth oxides and thorium.

Non-Magnetic Minerals

In general, the non-magnetic fraction will contain the more valuable minerals rutile, zircon, leucoxene, etc. The composition of the non-magnetic minerals may be determined by mineral count or chemical analysis. Rutile and leucoxene have value to TiO_2 manufacturers. Zircon is used as a foundry sand, as an opacifier, and to manufacture zirconium metal.

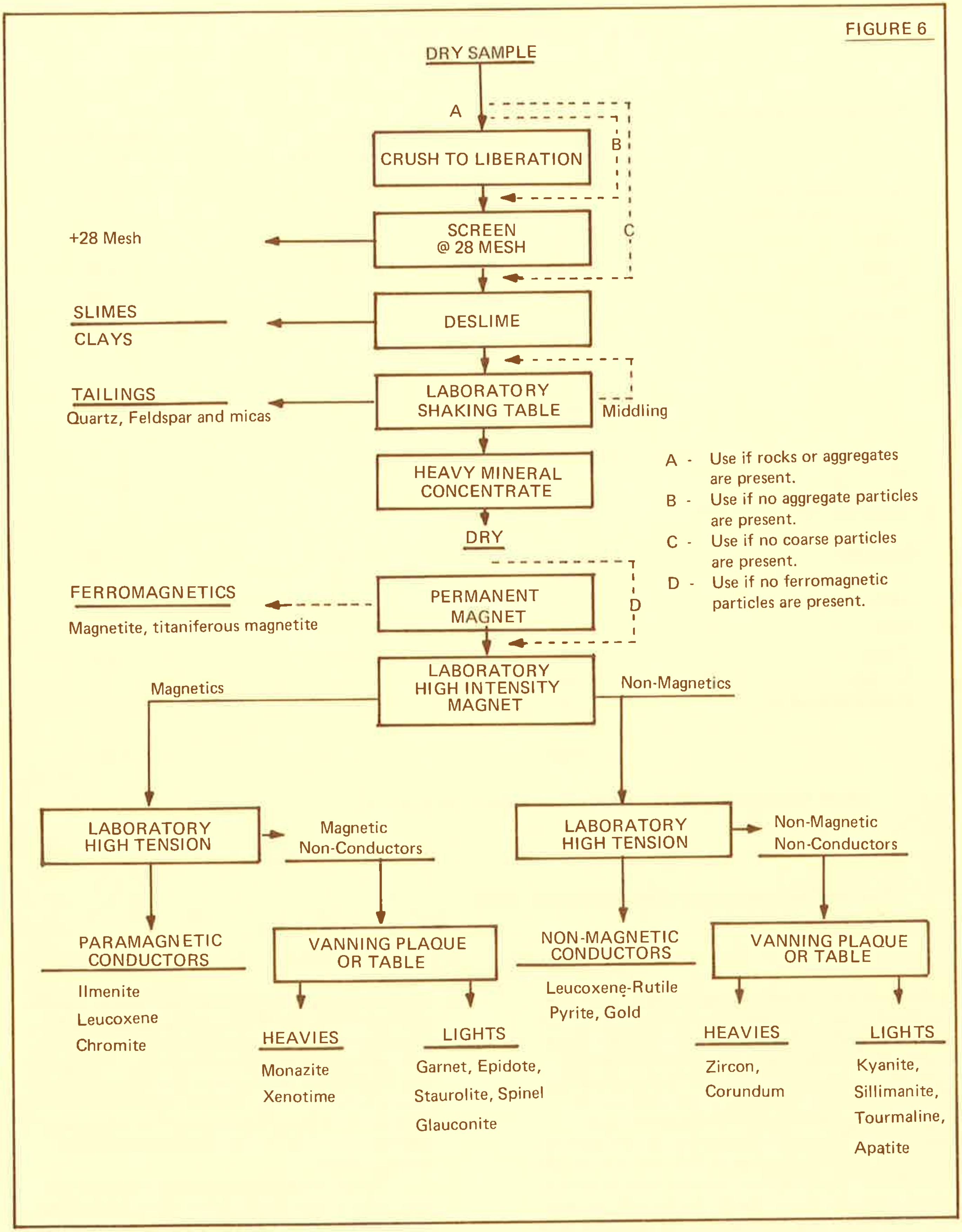
Preliminary Economic Evaluation

The value of a heavy mineral suite can be estimated by multiplying the percent of each heavy mineral constitutents by its present market value. These values can be found in "Engineering and Mining Journal" and "Industrial Minerals" magazines. If there is sufficient value indicated for the heavy mineral suite and there are enough heavy minerals in the original sample to warrant, then more detailed testing should be undertaken to investigate the separation characteristics of the heavy minerals. Examples are shown in Table 2.

Determination of Separation Characteristics

A large sample representing the prospective ore is obtained using composite drill samples, pitting, or trenching. The sample size should be large enough to furnish at least five pounds of heavy mineral concentrate. Using the laboratory procedure outlined in Figure

FIGURE 6



A - Use if rocks or aggregates are present.
B - Use if no aggregate particles are present.
C - Use if no coarse particles are present.
D - Use if no ferromagnetic particles are present.

FIGURE 6 - **Outline for laboratory determination of mineral separation characteristics.**

TABLE 2 — Examples of Heavy Mineral Value Calculations
(Hypothetical Cases)

Potentially Economic Ore:

	% of H. M.	x	Current Market Value	=	Heavy Mineral Value
Ilmenite @60% TiO2	45		$ 29*		$13.05
Leucoxene @ 87% TiO2	5		213*		10.65
Rutile	2		313*		6.26
Zircon	5		175*		8.75
Staurolite	20		5**		1.00
Monazite	1		219*		2.19
Garnet-Epidote	15		—		—
Kyanite-Sillimanite	7		—		—
	100%				$41.90

4% H. M. in ground x $41.90 = $1.68 value per ton of ore in ground.

Marginal or Uneconomic Ore:

	% of H. M.	x	Current Market Value	=	Heavy Mineral Value
Magnetite	2%		$ 18**		$ 0.36
Ilmenite @54% TiO2	20		19*		3.80
Leucoxene	1		213*		2.13
Rutile	—		—		—
Zircon	2		175*		3.50
Pyriboles	60		—		—
Kyanite-Sillimanite	15		—		—
	100%				$ 9.79

10% H. M. in ground x $9.79 = $0.98 value per ton of ore in ground.

* "Industrial Minerals" magazine No. 100, January, 1976.

** Estimated

6, the heavy minerals are concentrated on a laboratory-size shaking table. The table concentrate is driend, then split magnetically at 7,000 gauss to separate the ilmenite. The magnetic portion is passed over a laboratory high tension unit and the conductor fraction recleaned. The final conductor fraction is examined under the microscope, and if it contains 98 percent or more opaques, that is a good indication that mineral particles are free from surface coatings. If there are large amounts of silicates in the conductors, this may indicate presence of surface coatings which could interfere with dry milling. Tests should then be undertaken to determine what can be done to remove these surface coatings. In some cases high energy attritioning alone may do it. Some ores will require chemical treatment. Most acids have been used with some degree of success to remove coatings. Hydrofluoric acid has been very successful in removing surface coatings; however, it is extremely hazardout and should be considered only if all others fail. NaOH may be useful in removing clays and organic surface coatings.

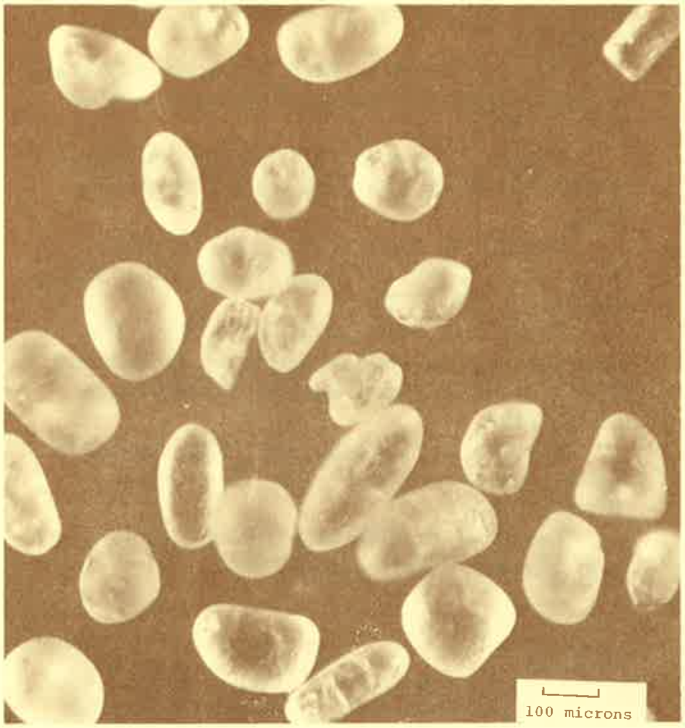
The non-magnetic fraction is run through the same series of tests. The presence of surface coatings, particularly on zircon, can be detected by calcining the non-magnetic non-conductors in a laboratory muffle furnace at 1000°C for 30 minutes. This treatment produces a bright orange color on grains with extensive surface coatings. It is important to have clean grain surfaces because surface coatings on zircon grains may not only effect their dry milling characteristics, but may also make the product unacceptable to the foundry industry where wettability and coatability are essential. Zircon is separated from lighter aluminum silicates using a shaking table if there is enough sample. If there is not enough sample, zircon can be upgraded with a vanning plaque.

Particle Size Analysis

Industries using the heavy mineral products usually specify the particle size distribution desired. Therefore, each product from the laboratory testing should be screened. In the event that none of the laboratory products are pure, then screen the entire sample, do petrographic analysis on each size fraction, and then calculate the size distribution for each mineral in the composite sample.

About 85 percent of the zircon produced goes into the foundry industry. Foundrymen prefer rounded grains with size range of 80 to 150 mesh. Du Pont zircon produced at Starke, Florida, is shown in Figure 7. A sample of zircon from a continental deposit is shown in Figure 8 for comparison.

FIGURE 7 - Photomicrograph showing zircon grains from marine shoreline-dunal deposits. Note grain roundness. Grain surfaces show frosting due to wind action be possibly chemical solution. (E. I. duPont de Nemours & Co., Inc., Trail Ridge Plant, Starke, Florida).



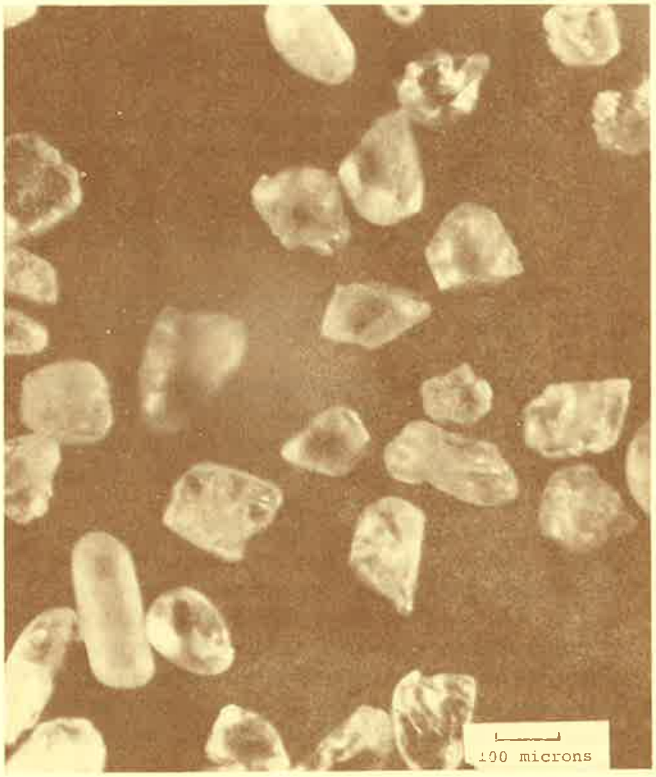


FIGURE 8 - Photomicrograph showing zircon grains from continental deposits. Note angularity and lack of surface rounding on most grains. (ASARCO, Lakehurst, New Jersey).

Finer zircon (say – 200 mesh) cannot be used in most foundry sand applications. If zircon quality is good enough, it may be used in manufacturing zirconium metal.

Final Evaluation

Decision to do further work leading to feasibility studies and economic evaluations will depend on the following:

- Laboratory work must show sufficient valuable minerals to be economic and no milling problems detected.
- Deposit must be located in favorable location with respect to potential markets.
- There must be roads, railroads, and labor supply.
- Environmental impact.
- Land use after mining.

Summary and Conclusions

This paper shows:

- Heavy mineral deposits have extremely complex geologic histories.
- Mineral samples vary in character and composition and reflect the type of deposit from which they were derived.
- The value of a heavy mineral suite may depend more on composition than on concentration in the ground.
- Extensive laboratory work is required before potential value of an ore can be accurately determined.

Perhaps the most important points are: No single laboratory method will fit all samples and no single set of rules can be set forth to accurately evaluate all heavy mineral ore deposits. Each deposit is different and each must be evaluated on its own merits.

There is no doubt that demand for heavy mineral products will continue through the foreseeable future, in fact demand will probably increase over the next two decades. To meet these needs the industry must prepare to meet increasing labor and energy costs with new, innovative methods to more cheaply mine and separate heavy minerals, and be prepared to restore land to 100 percent usefulness and leave no hazardous conditions for future generations.

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RECONNAISANCE INVESTIGATIONS OF OFFSHORE PHOSPHATE DEPOSITS OF GEORGIA AND SOUTH CAROLINA

James L. Harding
Marine Resources Center
John E. Noakes
Geochronology Laboratory
University of Georgia
Athens, Georgia

Introduction

An important mineral deposit consisting of bedded marine phosphorite is known to exist in the subsurface strata underlying the marshlands, estuaries and barrier islands of coastal Georgia. As it is centered in the eastern portion of Chatham County (which is the northernmost of Georgia's coastal counties), this deposit is usually referred to as the Chatham County Phosphate Deposit. The stratigraphy and economic geology of the deposit were presented by Furlow (1969) in a comprehensive study conducted by the Georgia Department of Mines, Mining and Geology. The objectives of his study were: 1) to determine if the mining of phosphate in the eastern portion of the county would jeopardize the fresh water aquifer which underlies the phosphate-bearing zone and 2) determinations of the depth, grade, volume, value, etc. of the ore.

Although Furlow concluded that mining done under strict supervision of regulatory agencies would not imperil the integrity of the aquifer, the proposed exploitation of these deposits led to an uproar on behalf of the environmentalists so the plans for mining were shelved. While such plans were still viable, interested parties drilled well over one hundred exploratory holes in the general area of interest. Data from these holes indicated that the phosphate-bearing strata (matrix) might extend several kilometers seaward of the Georgia coast.

In 1962, two test holes were drilled 16 kilometers (10 miles) offshore of Savannah Beach for engineering purposes coincident to the emplacement of a light tower designed to replace the Savannah lightship. The holes penetrated strata varying in age from Recent to late Eocene (McCollum and Herrick, 1964). These authors found a continuation of the subsurface stratigraphic sequence that exists onshore due west of the tower site.

According to Pickering (1976) re-examination of the Miocene section of cuttings and cores from the offshore test holes, now retained by the State of Georgia, showed the presence of the Duplin Marl, the same formation which contains the phosphate matrix onshore beneath eastern Chatham County. A columnar section — drill log prepared by Pickering showed the seafloor at the tower test hole #1 site to be about 17.5 meters (58 feet) below mean sea level, with the Duplin Formation being overlain by 0.9 to 1.2 meters (3 to 4 feet) of undifferentiated Recent to Pleistocene sands. The sands contained accessory quantities of shell hash and brown or black phosphate pebbles. The seafloor here is well within wave-base range, so that these few meters of unconsolidated coarse sands probably suffered constant re-working.

Beneath the sands, Pickering showed about 7.3 meters (24 feet) of the phosphate-bearing Duplin Marl Formation. Whereas the Chatham County deposit as described by Furlow (op. cit.) was overlain by up to 15.2 meters (50 feet) of non-orebearing Duplin Marl, the barren portion of this upper Miocene formation was non-existent in the area of the light tower, apparently having been removed by erosion.

Investigative Methods

The reconnaissance investigation described in this paper was based in large part on extrapolations from onshore data and the results of the analyses made on the drill hole samples from the tower site.

In 1970, the investigators began to develop applications of nuclear technology for adaptation to problems involving marine mineral exploration. The initial efforts (1970-73) were aimed primarily at evaluating the use of neutron activation analysis, first on the seafloor using a manned submersible and then on shipboard through measurements made on cores and water samples. The results of these research efforts were described by Noakes and Harding (1971) and Noakes, et al (1972).

During the work with the neutron activation analysis systems, new tools were developed for radiation detection in the marine environment. Although the work had involved the utilization of a man-made isotope, Californium 252 (^{252} Cf), it became apparent that the same tools could be modified to detect any naturally occurring radioactivity associated with seafloor sediments.

Previous studies, summarized by Emery and Uchupi (1972), indicated that the most common occurences of natural radiation in marine sediments were associated with heavy mineral and phosphorite deposits. Heavy minerals such as monazite, zircon, sphene and epidote may contain thorium and its decay daughters, whereas phosphate deposits may contain uranium and its decay products.

The potential of a large deposit of phosphate situated in dredgeable depths within continental waters was of far greater interest and import than was the proposed mining beneath the marshlands of the mid- and late 1960's. It was therefore decided to focus research efforts toward the development of a system to detect and differentiate between the gamma activity associated with any heavy minerals and/or phosphate in the surficial sediments of Georgia's continental shelf.

Instrumentation

The radiation survey system consisted of 1) a towable underwater sled and 2) a static radiation analysis device. The sled (Figure 1) was constructed of heavy stainless steel and contained four gamma-ray detectors consisting of 7.6-centimeter (three by three-inch) thallium-activated sodium iodide NaI(T1) crystals. It was similar to the one developed by Summerhayes, et. al. (1970) for marine phosphorite prospecting. Also, Bowie and Clayton (1972) and Miller and Symons (1973) developed and tested parallel systems. Preamplifiers and high voltage electronics encapsulated within the sled transmitted electronic pulses via coaxial cable to the surface vessel. There a dual channel recorder traced the gamma activity in terms of background and variations from background (anomalies) while the vessel was underway.

The static detector was designed to rest directly on the seafloor and consisted of a tripod arrangement housing a cylinder which contained a single large lithium-drifted germanium Ge(Li) detection crystal (Figure 2). The Ge(Li) detector was maintained at an operating temperature of minus 180 degrees Centigrade by a cryogenic coolant.

The static detector was a specific isotope detection system, which by obtaining spectrum analyses on a multi-channel analyzer and plotter, could differentiate between uranium and thorium and their respective decay products.

Operational Procedures

The radiation detection sled was towed astern the survey vessel with 2.2 centimeter (7/8 inch) stainless steel cable as tow cable, to which the coaxial signal cable was taped for the first 15 to 30 meters (50 to 100 feet) underwater. At the onset of the program, while still in the testing phase, tows were restricted to the intracoastal waterway and the various sounds along the Georgia coast. It was then taken into the open waters of the continental shelf where parallel traverses approximating the length of the Georgia coast were then made with the sled in tow (Figure 3). Although much of the work was weather-dependent as the research vessel *Kit Jones* was a 17 meter (57-footer), the two speeds averaged between three and six knots, depending upon sea state and bottom topography.

Horizontal positioning was accomplished by a combination of Loran-A fixes and radar bearings. Time,

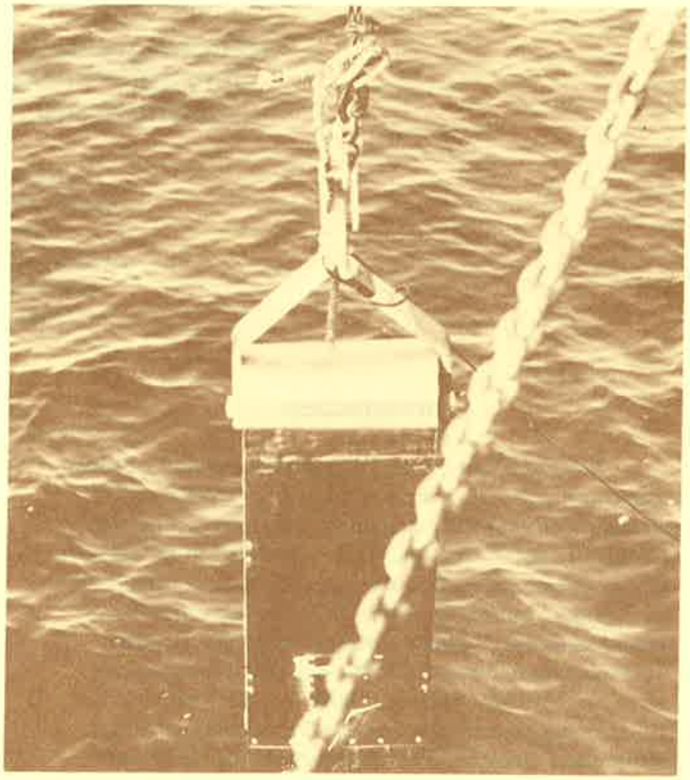

FIGURE 1 Gamma-ray detection sled, housing four Na I (TL) detectors, preamplifiers and associated electronics.

FIGURE 2 - Static underwater gamma detection system. Base of unit must be in direct contact with seafloor to register readings for spectrum analysis.

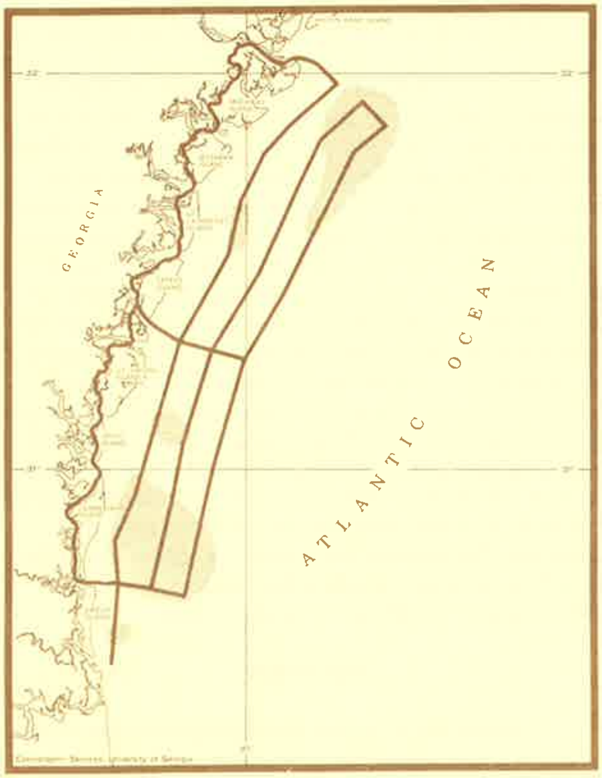

FIGURE 3 - **Original traverse lines in Georgia coastal waters.**

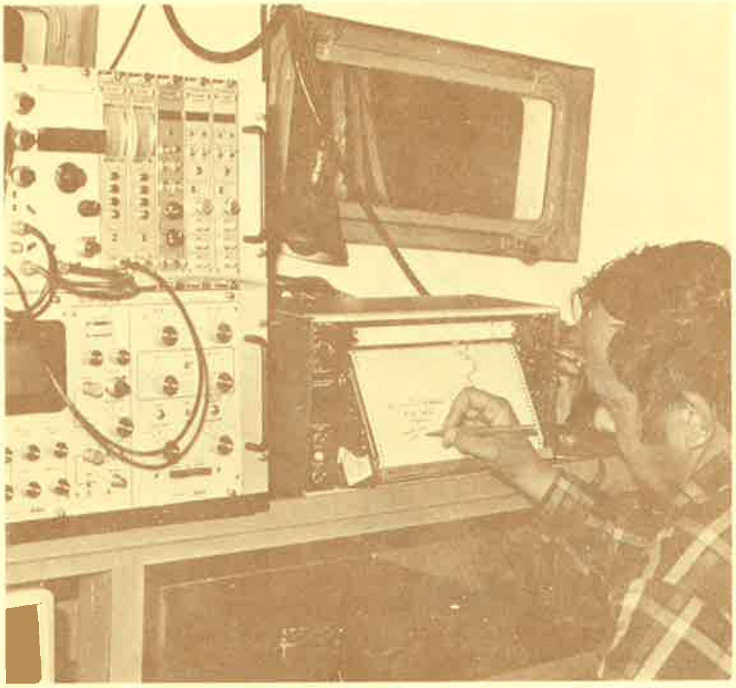

FIGURE 4 - **Radiometric readings recorded by shipboard electronics from towable underwater sled.**

compass bearing and Loran fix positions were marked simultaneously on the dual-channel ship recorder (Figure 4) and on the ship's depth-recorder charts. This was done as a matter of course every ten minutes. When anomalies were apparent on the sled-recorder, additional fixes, etc., were logged.

The original intent was to complete the series of tow traverses, keeping track of all anomalies and then return to such sites to utilize the static, specific isotope detector. This latter unit required that the vessel anchor at each site, because the electronics were such that between 30 and 45 minutes of direct bottom contact were required in order to record ample spectra data. The R/V *Kit Jones* carried only one anchor and after several frustrating attempts, it became clear that at least a two-point moor was necessary to achieve stabilization of the unit with regards to contact with the seafloor. Its use was therefore abandoned in favor of obtaining samples and doing the spectrum analyses in the laboratory.

Toward the end of the field phase of the investigation, a high-resolution acoustic seismic profiler unit was towed in conjunction with the sled. Both were towed by the surface vessel moving at a speed of about five knots. A sample of a seismic record thus obtained is shown in Figure 5.

Results

The original offshore traverses paralleled the coast and extended from abeam Mayport, Florida (just north of Jacksonville) to Port Royal Sound, north of Hilton Head Island, South Carolina. These lines were conducted in water depths ranging from 3 to 30 meters (10 to 100 feet).

As the work progressed, a definite geographic zonation of gross anomalies appeared, wherein the southern portion of the survey area (south of Sapelo Island) had but one large anomalous area, located directly offshore of the southern half of Cumberland Island. Subsequent sampling indicated that this one anomalous area was due to localized concentrations of heavy minerals in the surface sands. The middle portion of the survey area (between Sapelo and Wassaw Islands) was essentially barren with respect to anomalies, whereas the readings in the northern portion (between Tybee and Hilton Head Islands) were generally the highest encountered. The survey line direction, therefore, was changed to one which ran essentially east-west, in an attempt to grid the area, using the position of the light tower as a control point. The results seemed to indicate a slight increase in above-background readings towards the northeast. The final grid pattern is that illustrated in Figure 6, which also utilized the navigational tower as a reference point.

During the running of the grid pattern shown in Figure 6, the high-resolution sub-bottom profiling unit and the gamma detector sled were being towed from the surface vessel at a speed of about five knots. Radioactive anomalies located by the sled detector could be readily correlated with undulations in the bottom and near

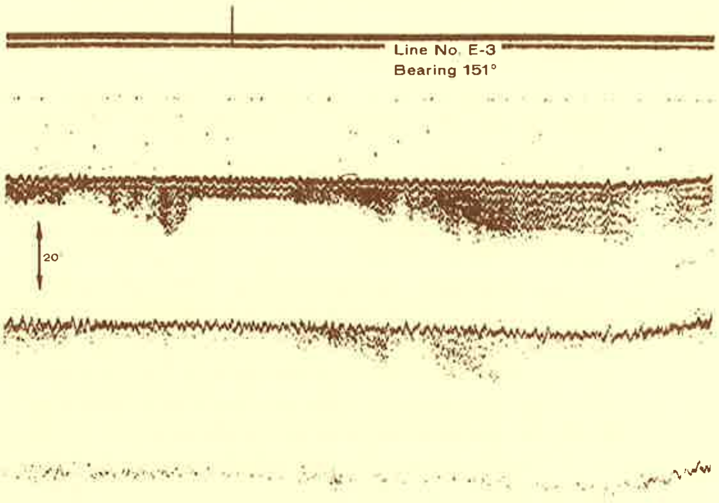

FIGURE 5 - **Sub-Bottom profiler (Elac-Echograph Laz-17) record of portion of line E-3.**

bottom strata as shown on the profile recorder. In places, where a sub-bottom reflector would rise to intersect the seafloor, an anomaly would also appear on the shipboard instrument obtaining signals from the sled. Figure 5 shows the sub-bottom profile record taken on line E-3. There are between 1.5 and 6 meters (5 and 20 feet) of fairly well-stratified unconsolidated sediments lying atop a horizon which is acoustically opaque at the frequencies employed. Undulations in the sub-bottom strata are also apparent. It is possible that this opaque surface represents the top of the Miocene which lies a few feet below the seafloor at the tower site according to Pickering (1976).

Grab samples taken in the grid area were analyzed for P_2O_5 and percent BPL, the results of which are given in Table I. As these samples represent only the top several inches of the seafloor sediments, any concentrations of phosphatic material could merely reflect wave-base concentrations.

Suggestions for Future Work

The gamma detection sled and the static spectrum analysis system described herein are applicable for gross reconnaissance work only, because their detection limits are confined to the upper few inches of seafloor material. By themselves, they are excellent guides; when combined with high resolution seismic profiling, the system improves. Additional substantiation is needed in the vertical, however, and this can only be done by a coring program.

The work done to date was conducted under the auspices of the Georgia Sea Grant Program, and the investigators were limited to using vessels within the Georgia Marine Program, none of which are capacle of the type of coring required. Any further work should utilize a combination of tools consisting of (1) high resolution seismic profiling which should be either a mini-sparker or a Uniboom, so that at least 30 meters (100 feet) of data can be obtained on the sub-bottom strata; and (2) precise horizontal positioning, accurate and reproducible to within plus or minus one meter; (3) side-scan sonar for mapping the micro-bathymetry; (4) the gamma detector sled system and (5) a survey vessel capable of at least a two-point anchoring system in the case of vibracores or a drill barge with support tug for deployment of a three or four point anchor spread in the case of a rotary rig. The detailed sub-bottom profiling should be done first, in order to reduce the number of drill hole sites necessary to define the prospect area.

Summary

It should be reiterated that the gamma detection system is a reconnaissance tool and is only capable of detecting radioactive material in the surface sediments. The sub-bottom profiler, where used, gave the only clues as to the composition and altitude of the subsurface

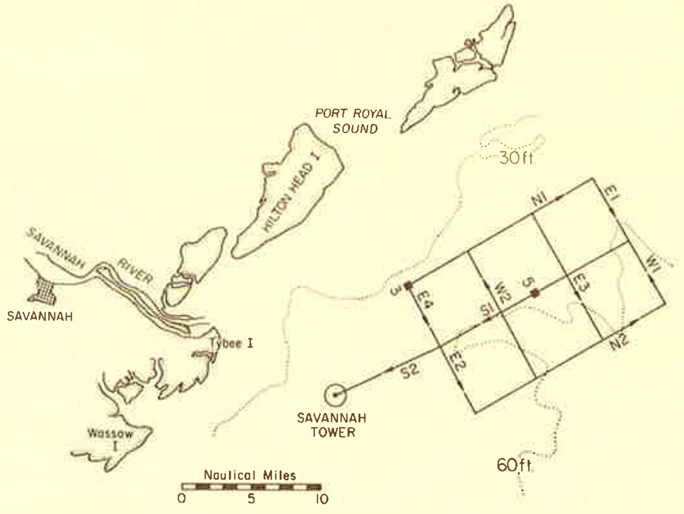

FIGURE 6 - **Final grid pattern run off Georgia-South Carolina coast, using tower location as a control point.**

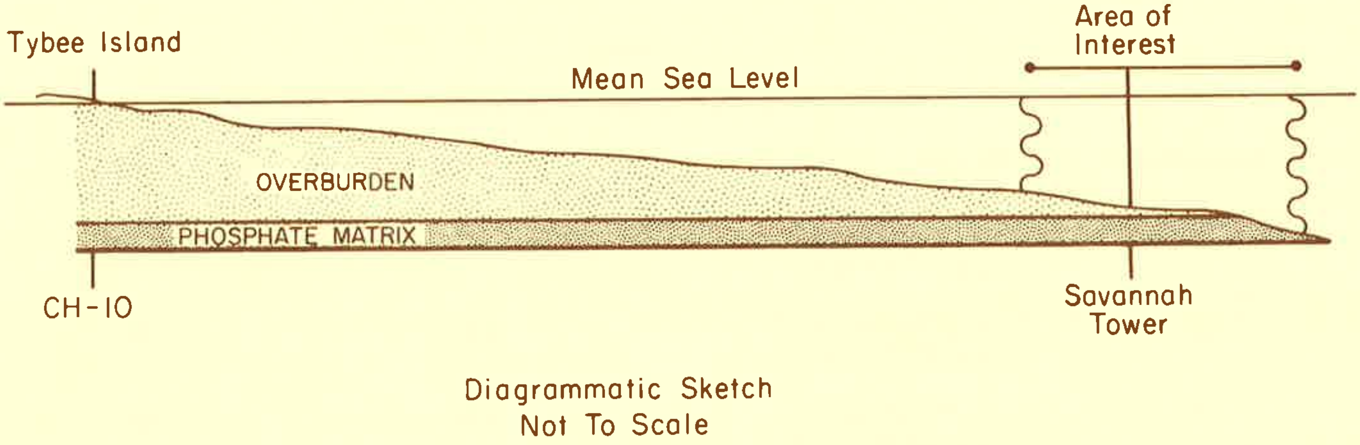

FIGURE 7 - Diagrammatic cross-section showing gross relationships between matrix zone and shelf strata.

FIGURE 8 - Diagrammatic fence diagram showing gross relations of near sub-surface strata.

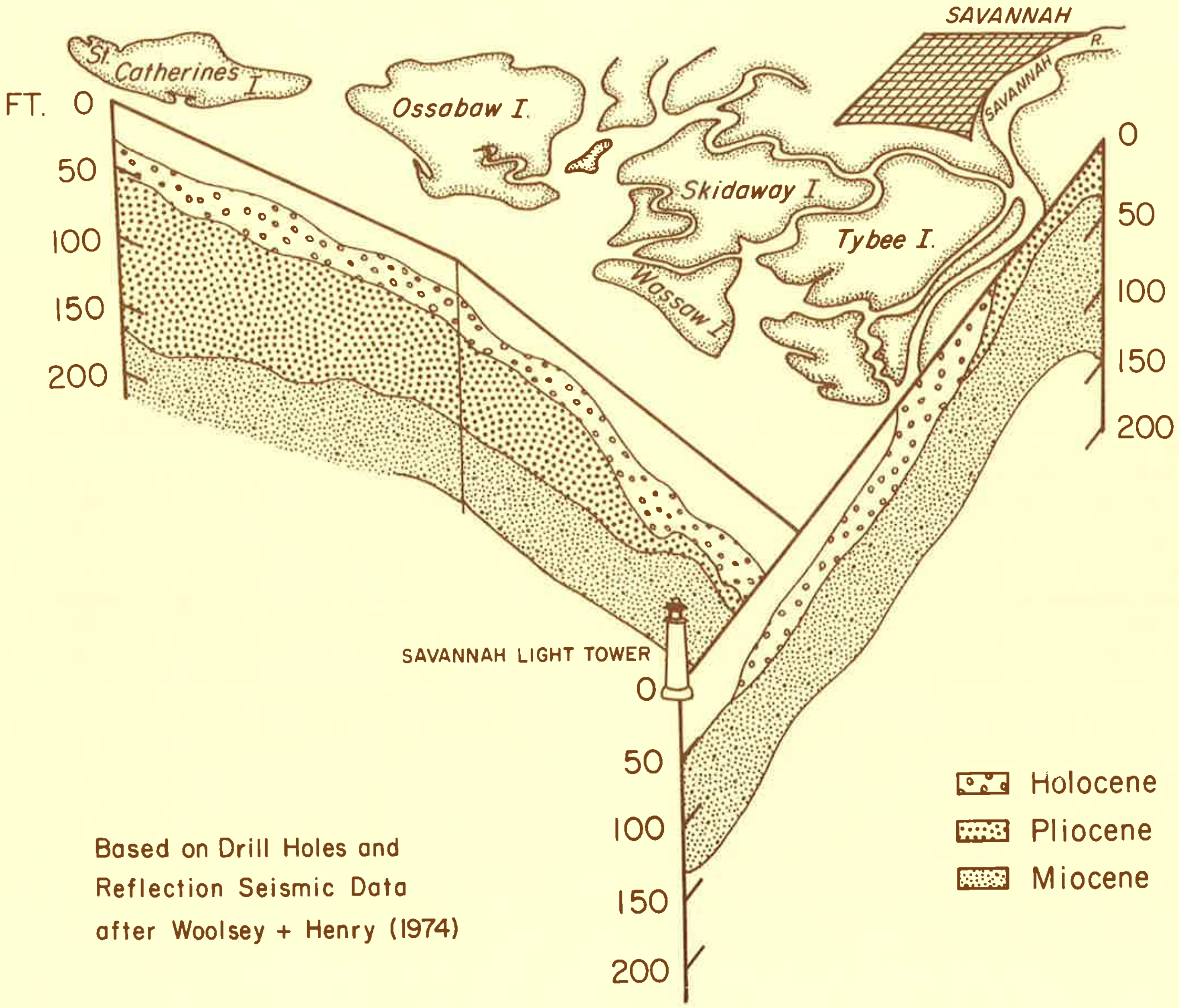

TABLE I

Phosphate Analysis of Grab Samples

Sample No.	Location	$\%P_2O_5$	%BPL
1	Lat. 31° 51′ 45″ N Long. 80° 52′ 00″ W	0.15	0.327
* 3	Lat. 32° 05′ 00″ N Long. 80° 34′ 10″ W	0.57	1.24
* 5	Lat. 32° 04′ 33″ N Long. 80° 25′ 30″ W	1.65	3.60
8	Lat. 31° 40′ 20″ N Long. 80° 54′ 00″ W	1.19	2.60
11	Lat. 31° 12′ 30″ N Long. 81° 04′ 45″ W	0.12	0.26
17	Lat. 31° 00′ 00″ N Long. 81° 12′ 30″ W	0.70	1.53
22	Lat. 31° 43′ 10″ N Long. 80° 59′ 00″ W	1.64	3.58

Vanomolybdate colorimetric method used.

*Sample stations are in Grid area depicted in Figure 6.

strata. Nonetheless, the authors feel that because the investigation showed a definite geographic pattern to at least surficial concentrations of phosphatic material, and that pattern coincided with the data obtained from the tower borings, the inner Continental Shelf off Chatham County, Georgia, and the adjacent counties in South Carolina may contain a potentially economic deposit of phosphate within dredgeable depths. According to Pickering (1976), the matrix (ore-bearing) section of the Duplin Marl is 7.3 meters (24 feet) thick at the tower site where the BPL content ranges from 19 to 40 percent.

Vertical data are insufficient to identify the regional dip the strata involved, but it is reasonable to assume that somewhere to the east of the tower site, the seaward slope of the shelf causes the Miocene section to thin (see Figures 7 and 8). It is also possible that a significant portion of the ore-bearing section has been removed by erosion.

In order to fully evaluate the ore potential of the offshore deposits, information on the thickness of the matrix, overburden characteristics, and the relationship to the underlying strata must be obtained. The former two are imperative for design of the recovery system whereas the latter must be firmly established in order to guarantee the integrity of the aquaclude and the principal aquifer beneath.

Coincident to any additional prospecting, information should also be gathered on the current regime, water mass movement and water chemistry in the general prospect area, as such data will be necessary for the preparation of the environmental impact statement certain to be required by the regulatory agencies prior to the issuance of any leasing, etc. Portions of the area are currently under study by scientists of the Skidaway Institute of Oceanography in Savannah under the auspices of ERDA, and some information will therefore be available for baseline purposes. Although the offshore area is far less critical to the ecological balance of the species which are known to spawn and spend a portion of their juvenile life-cycles in the estuaries and marsh-dominated tidal creeks, the effects of suspended sediments and detritus stirred up by dredging operations must be established in advance.

Acknowledgements

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DEVELOPMENT OF HIGH EXTRACTION MAGNETIC FILTRATION BY THE KAOLIN INDUSTRY OF GEORGIA

Joseph Iannicelli
Aquafine Corporation
Brunswick, Georgia

Status Of Development

Between 1973 and 1975, each of the 5 major kaolin producers in Georgia put hugh 20 kilogauss filters into service. These filters are unique as to size, power, capacity, and novelty of design.

Each magnetic filter weighs 250 tons and can process its own weight of kaolin slurry each hour when operated at the maximum throughput (20 seconds retention time). Under these conditions, flow rates are about 5425 liters (1,400 gallons) per minute or about 100 tons per hour on a dry basis.

These five installations represent a total investment of over 8 million dollars (1975 dollars) and can process nearly all of the waterwashed kaolin in the United States if operated at maximum throughput. This commercialization constitutes the most rapid acceptance of novel magnetic separation technology by a mining industry in recent times.

Full scale commercial use of magnetic separation is the culmination of a 10 year effort almost entirely supported by the kaolin industry. It is the only commercial use of this technology at this time. However, the experience gained in the kaolin industry is now a good springboard for practical applications of this process in other industries.

Evolution of the Process

High extraction magnetic filtration was carried from initial research to pilot production in 1968 after a 5 year research and development effort on kaolin. During this period the importance of 3 key concepts were identified and successfully combined into a practical prototype separator.

An early breakthrough in HEMF was the finding that separations on micron and submicron kaolin suspensions required very slow transit velocities and maximum exposure to collecting elements to allow capture of paramagnetic particles.

To achieve particle capture, the magnetic attraction of a collecting element must overcome viscous drag of the transporting fluid. Therefore, it was necessary to operate at much lower velocities (less than 2 cm/sec.) than had ever been employed in practical separations.

Discriminating control of slurry exposure in the collecting zone of a magnetic filter is best expressed by the concept of retention time. It is the subject of a basic process patent on kaolin.

Evolution of Small Scale HEMF Equipment

To exploit the concept of retention time, it was necessary to devise a new magnetic separator which for the first time combined high gradients of the Frantz Separator (Figure 1) with high intensity fields of the Jones Separator. (Figure 2). This critical fusion of separator technology was achieved in 1967 when J. Iannicelli constructed a hybrid separator (Figure 3) utilizing a canister filled with high gradient Frantz screens installed in the high field gap of a Jones machine. This hybrid device generated an average field of nearly 10 kilogauss throughout a relatively large volume of 38 liters (10 gallons). The new hybrid separator furnished a startling increase in efficiency over the previous Frantz and Jones Separators. For the first time, the equipment and process had reached the stage of commercial practicality.

At this point, further refinements were added as a result of contacts with consultants retained by the J. M Huber Corporation. This effort led to the design of prototype equipment generally resembling a Frantz Ferrofilter, but modified to generate fields of up to

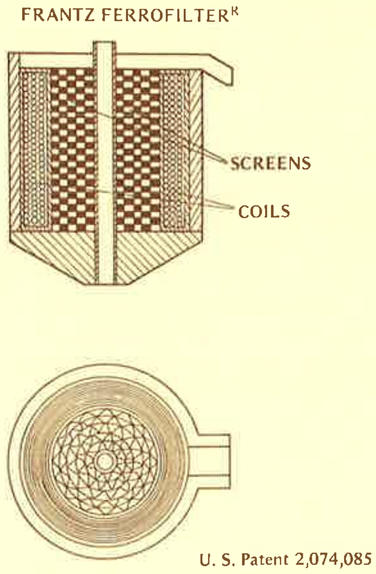

FIGURE 1 - Frantz Ferrofilter^{R}. Section elevation view (top) and plan view showing Frantz screen matrix and solenoid coil construction.

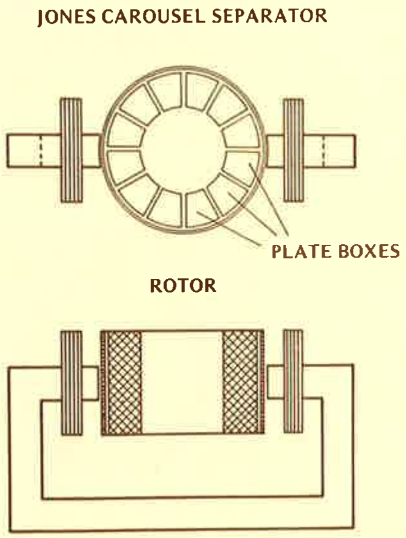

FIGURE 2 - Jones Carousel Separator. Schematic plan view (top) showing compartmented rotor, magnetic coils and pole pieces. Elevation view (bottom) shows steel yoke return circuit for flux.

20 kilgauss. Stainless steel woood had been shown at J. M. Huber to be more effective on kaolin than coarser ribbon Frantz screens, so the prototype incorporated this refinement.

The first production prototype high extraction magnetic filter (HEMF) was placed on stream in 1969 at J. M. Huber. It consisted of a hollow conductor solenoid surrounding a canister 50 centimeters (20 inches) in diameter by 30 centimeters (12 inches) high. The solenoid was surrounded by a heavy box-like enclosure which served as a return circuit for magnetic flux generated by a short coil equal in height to the canister (Figure 4). This small filter generated a field of 20 kilogauss and required 300 kilowatts of power. Throughput was about 2 tons per hour (dry basis). Estimated processing cost was about $2 per ton which made the process of interest only for speciality kaolin applications.

A second small filter rated at 10 kilogauss and 4.8 tons per hour was delivered and placed on stream at J. M. Huber in 1970. In 1971, a third small unit of this type was acquired by the Industrial Minerals Department of Cyprus Mines.

Development of this equipment is the subject of numerous apparatus patents issued to J. M. Huber as well as to Magnetic Engineering Associates and M.I.T.

Modern HEMF Production Units

In 1971, the Pacific Electric Motor (PEM) Company (Oakland, California) agreed with Aquafine to undertake design of full scale HEMF equipment using experience gained through 20 years of construction of large high field magnets in their own shops.

Because of extensive experience in the design and fabrication of large high field magnets, PEM was able to proceed directly with design of very large magnetic separators for full-scale use by the kaolin industry. Efforts were devoted to design of the largest machine whose components would be shippable by ordinary carriers.

Design and manufacture of the first 84-inch (213-centimeter) separator required solutions to numerous problems that had never been faced before in construction of magnetic separators.

A key feature in the design of these new separators was the long coil devised by J. Allen of PEM. In the long coil separators, the coil axis is 50 to 100 percent longer than the magnet gap or canister height. (Figure 5).

Since the long coil had a smaller mean diameter (compared to the prototype units), the long coil required less conductor than a short coil having the same number of turns. This saving in copper signigicantly reduced the capital cost of separators. Comparison of the two types of coils is shown in Figures 4 and 5.

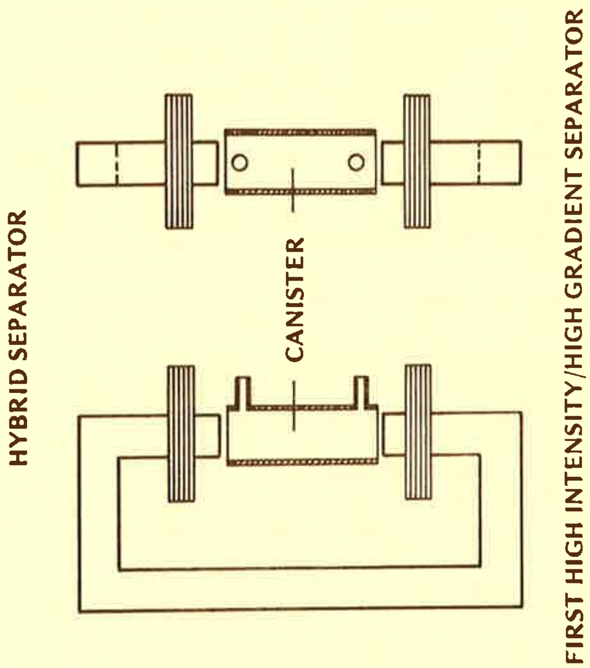

FIGURE 3 - Hybrid Separator. Schematic elevation views showing coils, steel yoke magnet with canister in magnet gap.

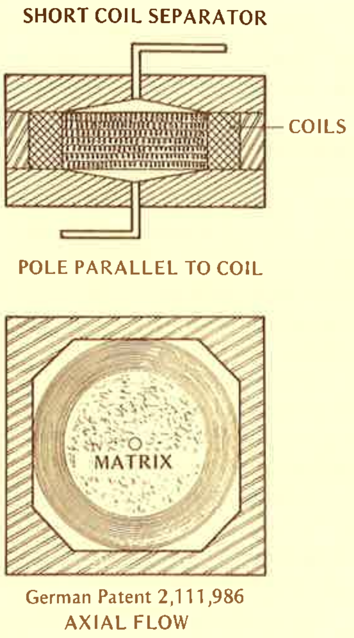

FIGURE 4 - Short Coil Separator. Schematic cross section in elevation (top) and plan (bottom) views showing matrix, canister, coils and steel return circuit.

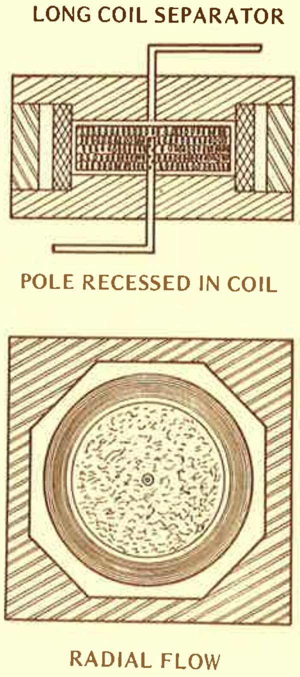

FIGURE 5 - Long Coil Separator. Schematic cross section in elevation (top) showing coil taller than canister and pole pieces fitting into the coil cavity. Fluid flow is parallel or axial to the magnetic field.

The first commercial HEMF unit was constructed by PEM in 1972 (Figure 6) and represented a 13 fold scale-up over the largest filter then in use at J. M. Huber. This PEM machine went on stream at Freeport Kaolin (Gordon, Georgia in the first quarter of 1973. A second PEM 84-inch machine went into production at American Industrial Clay Company in Sandersville, Georgia, during the fourth quarter of 1973, and a third PEM 84-inch machine was delivered to Thiele Kaolin in Sandersville, Georgia in early 1974. In addition, a 76-inch (193 centimeter) magnetic separator was delivered to J. M. Huber by Magnetic Corporation of America in 1973. In the second quarter of 1975, Engelhard Minerals and Chemicals placed the fourth PEM 84-inch unit in operation.

Description of HEMF Equipment

These PEM 84-inchseparators, which are the largest HEMF machines now in service, are far more efficient than any magnetic separation equipment that has been available in the past. The nucleus of the HEMF units consists of a canister which is (84-inch) 213 centimeters in diameter by 50 centimeters (20 inches) high and filled with compressed mats of 430 magnetic stainless steel wool. In essence, the unit is a powerful magnetic filter. The canister is surrounded by hollow conductor copper coils which energize the entire void volume of the canister to 20 kilogauss. Water cooled copper coils in turn are surrounded by a steel magnetic return circuit 3.6 x 3.6 x 2.4 meters (12 x 12 x 8 feet). Efficiency of the magnetic circuit is such that power consumption is only 400 - 500 kilowatts per hour.

Description of Process

HEMF units are operated in the manner of depth filters on a batch or cyclic basis. The filter bed matrix is magnetized and the slurry is pumped through to yield a filtered or nonmagnetic product. When the matrix is saturated, water is fed into the filter at a comparable rate to displace the nonmagnetic product.

The magnet is then de-energized and the bed flushed with high velocity water to remove magnetic contaminants. Processing of product is resumed after reenergizing the magnet. In normal operation, the duty or product cycle is 70 - 80 percent. Flow velocity can be varied from 102 centimeters (40 inches) per minute to 25 centimeters (10 inches) per minute, corresponding to a slurry retention time of 30 seconds to 120 seconds in a magnetic field. Filtration rate at 30 seconds retention corresponds to 90 liters per 0.09 square meter per minute (24 gallons per square foot per minute).

Utilization in Kaolin

HEMF equipment is employed to brighten kaolin by extraction of submicron feebly magnetic contaminants. Removal of only 1 - 2 percent of discolored ferruginous contaminants, based on original kaolin, typically brightens kaolin 2 - 4 General Electric (GE) Brightness units. In extreme cases, brightness improvement may be as low as 1 or as much as 30 GE units, depending on the nature of mineral contaminants in the kaolin.

These improvements can be utilized to (a) mine lower quality crude kaolin, thereby extending reserves (b) reduce the consumption of leaching chemicals (c) produce super-brightness clays (d) allow use of hard media to delaminate kaolin without discoloration[9] (e) produce clay products having superior end use performance in ceramics, glass, and catalysts because of reduced mineral contaminants.

Figure 7 shows the response of a good quality Georgia kaolin (80 percent minus two microns) as a result of HEMF processing. Even though the Fe_2O_3 content was reduced only 10 percent and the TiO_2 content only 25 percent the brightness improvement of 3 points is commercially significant. A retention time of 60 seconds was sufficient to raise unbleached brightness from 84.7 to 86.6. Normally, this would require treatment of this kaolin with from 1.7 to 3.6 kilograms (4 to 8 pounds) of zinc or sodium dithionite leaching chemical per ton of clay. Magnetic separation allows this brightness level to be reached without leaching reagents. In practice, 0.5 to 0.9 kilograms (1 to 2 pounds) of leaching chemical may be used merely to improve shade of kaolin. In any case, there would be a savings of leaching regent ranging from 0.9 to 3.2 kilograms (2 to 7 pounds) per ton of clay. HEMF treatment of kaolin under these conditions costs less than 0.9 kilograms (2 pounds) of dithionite reagent.

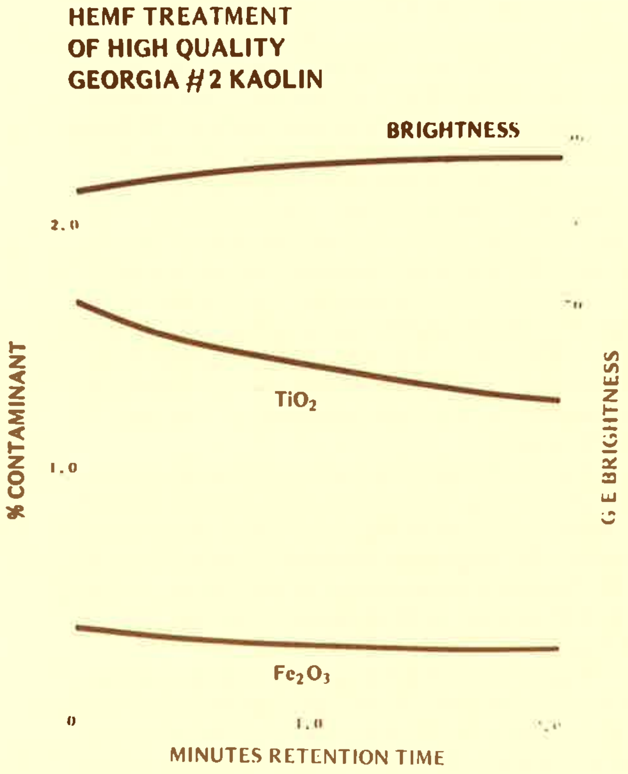

FIGURE 7

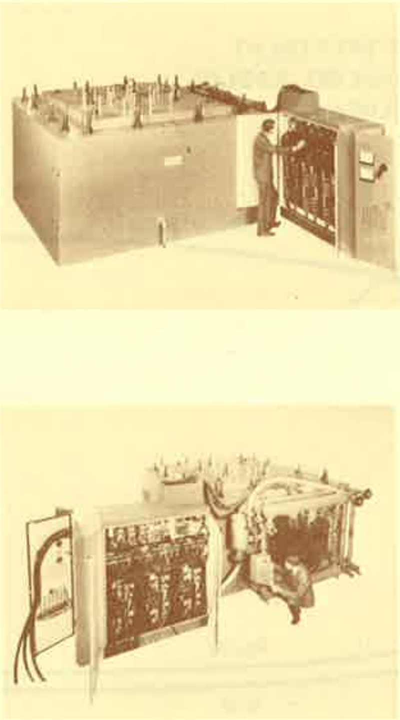

FIGURE 6 - 84-Inch Magnetic Separator with D. C. power supply.

With 2.0 minute retention time, brightness of this clay reached the 88 range. This can be converted to an 89 brightness after leaching, which allows the clay to meet premium or super brightness specifications and command a significant increase in price.

To illustrate the power and versatility of HEMF, submarginal kaolin samples from Ione, California, were subjected to magnetic separation as shown in Figures 8, 9, 10, 11, and 12. As a result of HEMF processing, it was possible to convert all of these clays from unusable submarginal deposits to usable kaolin reserves.

Figure 8 shows the response of a submarginal kaolin to HEMF at 20 kilogauss, with 0 to 2 minutes retention time. The original GE brightness of this clay was about 73. Crude selection normally requires a minimum brightness of 78 - 79 incommercial kaolin processing. After 30 seconds retention, the brightness of this sample rose to 79. This was accompanied by a decrease in iron content from 0.58 to 0.47 and a dramatic reduction in TiO_2 content from 1.9 to 1.2. After 2 minutes retention, brightness of this clay was 82 and the final TiO_2 analysis was 0.8 or less than half of the original.

Figure 9 shows a submarginal kaolin of 75 brightness with an unusally high TiO_2 value of 3.36. After 30 seconds magnetic retention, brightness reached 78 and the TiO_2 content was reduced to a value of 2.70. After 2 minutes retention, and brightness of the clay was about 82, allowing it to qualify for use as a paper filler without further processing. An interesting by-product of this treatment was reduction of K_2) from 0.30 to 0.22 and finally to 0.19. This represents a 37% decrease in mica content and has a very desirable improvement in the fired properties of this clay in ceramic outlets.

Figure 10 shows the dramatic reduction of TiO_2 in a sideritic clay. The original TiO_2 content of 2.38 decreased to 1.1 and finally to 0.71 after two minutes retnetion. GE brightness increased 12 points from 62 to 74. Potassium oxide content of this clay was reduced from 0.21 percent to 0.14 percent after two minutes retention.

Figure 11 shows the unusual and spectacular response of a very discolored kaolin which would never be considered a useful reserve prior to the advent of high extraction magnetic filtration. After one minute retention time, brightness rose from 49 to 70. Fe_2O_3 content decreased from 2.3 to 0.7 and TiO_2 content decreased from 1.9 to 0.8. After 4 minutes retention, this kaolin achieved a remarkable brightness of 79 and Fe_2O_3 and TiO_2 contents of 0.5 and 0.4 K_2O content decreased from 0.3 to 0.18 after 4 minutes, indicating significant removal of mica. As a result of magnetic treatment, an utterly unusable kaolin was converted to filler brightness. Subsequent chemical leaching of this whole fraction clay resulted in an 89 brightness which would put this clay in the premium brightness range.

Figure 12 shows the response of a gray lignitic kaolin (containing pyrite) to magnetic separation. Although the brightness increase was only 14 points over a retention span of 4 minutes and the final product submarginal with respect to brightness, the results are nevertheless significant. The Fe_2O_3 content decreased from 2% to 0.5% and the TiO_2 reduction was parallel. K_2O content was halved from 0.28 to 0.15.

The most significant aspect of this experiment was the striking reduction in the sulfur content of this clay. The initial sulfur value was 0.72 percent and the final sulfur content after 4 minutes retention was 0.08 percent. This represents a 90 percent reduction in the sulfur content of this clay through the nearly complete removal of micron and submicron pyrite by high extraction magnetic filtration. This experiment has enormous significance in that it demonstrates removal of pyrite in a very difficult, fine-particle size range. This and other results indicated that magnetic separation

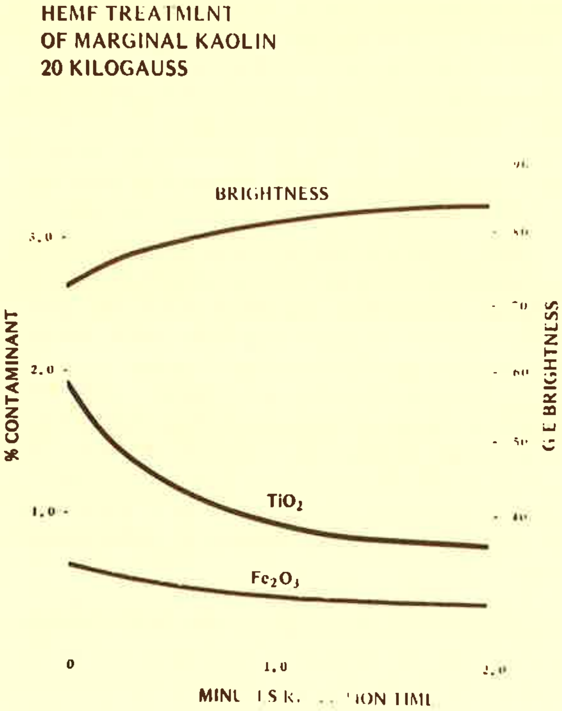

FIGURE 8

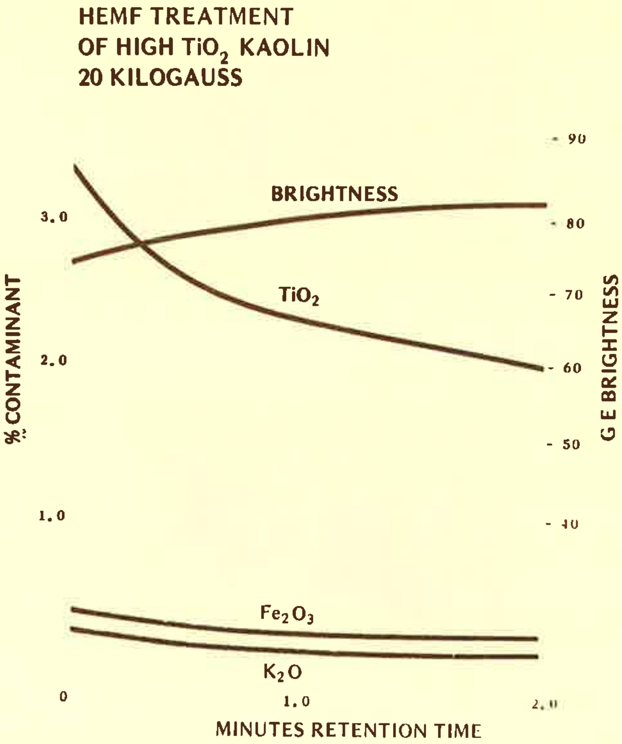

FIGURE 9

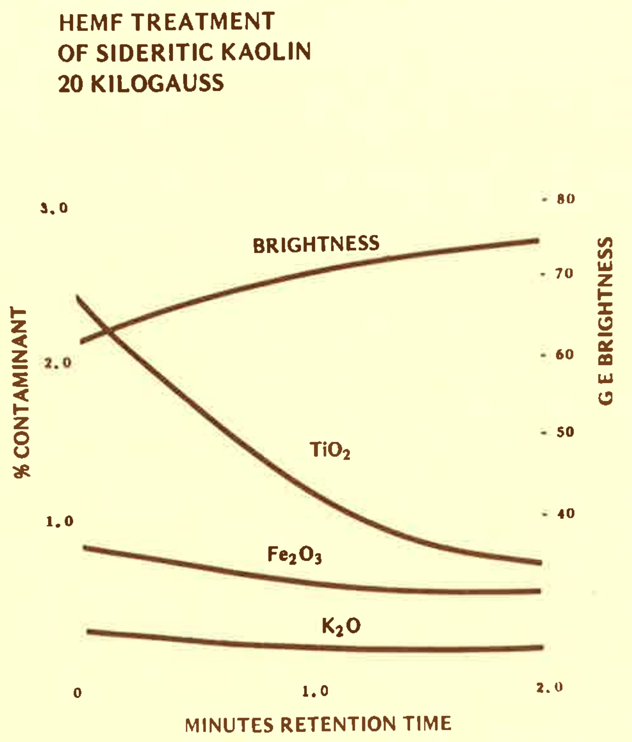

FIGURE 10

of pyrite from coal is feasible and that high extraction magnetic filtration of coal is a viable method of coal desulfurization.

Removal of iron stained anatase, siderite, and pyrite as shown in Figures 8-12, as well as removal of hematite, tourmaline, and other ferruginous minerals from kaolin has enormous significance to the benefication of other industrial minerals ranging from asbestos to zircon.

Nature of Magnetic Fractions Removed

Mineral slimes extracted by HEMF from kaolin include iron-stained anatase and quartz, hematite, mica, tourmaline, siderite, and pyrite. Depletion of iron values in clay varies from 10 to 90 percent of original iron content. Bulk magnetic susceptibility of magnetic fractions is as low as 35×10^{-6} emu/cc, and iron content of magnetic fractions is usually less than 5 percent (expressed as Fe_2O_3).

Figures 11 and 12 show HEMF response of kaolins containing relatively high amounts of Fe_2O_3 as well as TiO_2 while Figures 7, 8, 9, and 10 show effects on kaolins with a much higher content of TiO_2 than Fe_2O_3. In all of these cases, iron-stained anatase is the principal mineral contaminant extracted by HEMF.

Magnetic separation in kaolin processing is a comparatively low cost and versatile technique. By variation of retention time, it is possible to achieve differing degrees of magnetic extraction at acceptable variations in costs. The HEMF process is not only low in operating cost, but also furnishes high yield (98-99 percent) of kaolin product based on feed.

Economics

The economics of magnetic separation on large (84-inch) HEMF equipment are extremely attractive compared to that of small prototype units. This is due to efficient design coupled with the fact that the capacity of HEMF separators increases with the square of the diameter while the amount of copper conductor increases linearly with the diameter. Operating costs for an 84-inch HEMF unit are estimated in Table 1.

Processing of Other Industrial Minerals by HEMF

HEMF processing has been applied to a series of 40 industrial minerals in a study conducted under the direction of H. H. Murray at Indiana University. This study was conducted on a 20 kilogauss mobile PEM 5-inch pilot plant unit leased from the Aquafine Corporation. A report of this work shows that HEMF is an economical benefication technique for many of the industrial minerals evaluated.

Figure 13 shows the degree of iron removed from a group of industrial minerals after HEMF processing at (B) 30 seconds (C) 60 seconds (D) and 120 seconds

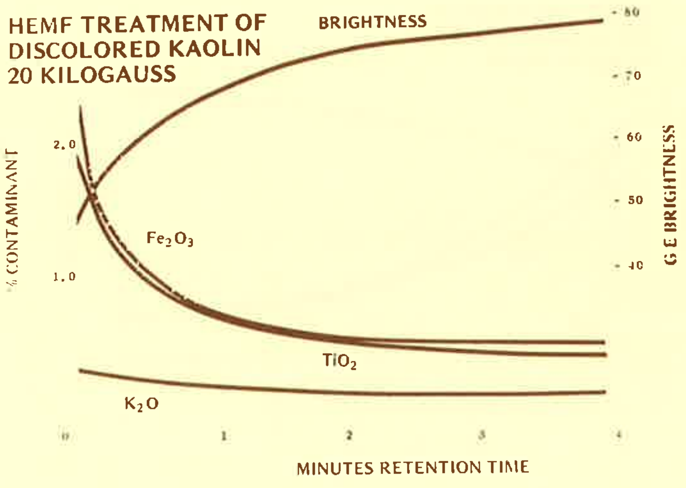

FIGURE 11

TABLE 1 – Estimated Operating Cost of 84-inch Pem Separator

	Cost per hour	General Purpose Use - Cost per Ton 66 tons per hour (30 seconds retention)	Specialty Purpose Use - Cost per Ton 15 tons per hour (120 seconds retention)
Amortization of installed separator ($1,600,000) over 10 years (80,000 hours)	$20.00 $20.00	$0.30 $0.30	$1.33 $1.33
Magnet Power (400 KW @ 2¢ KWH (switching magnet off for flushing easily offsets rectification losses).	8.00	0.12	.53
Pumping and Flushing Power (200 KW @ 2¢ KWH)	4.00	0.06	.27
Labor (incl. benefits)	5.50	0.08	.37
Maintenance	4.00	0.06	.27
TOTAL	$41.50	$0.62	$2.77

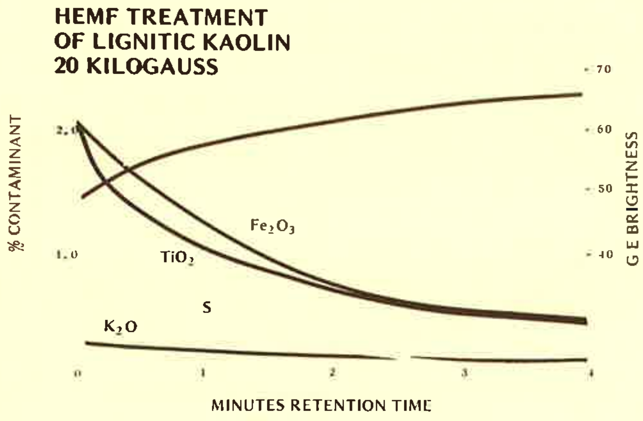

FIGURE 12

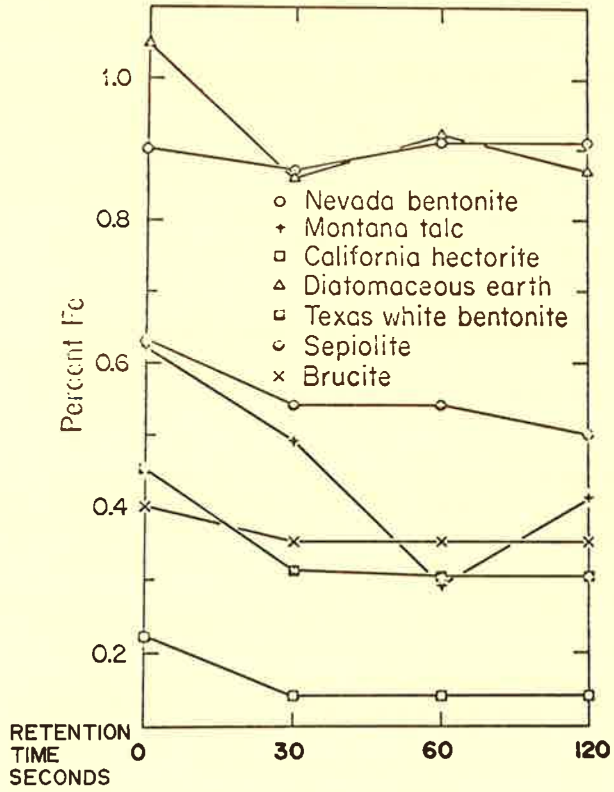

FIGURE 13

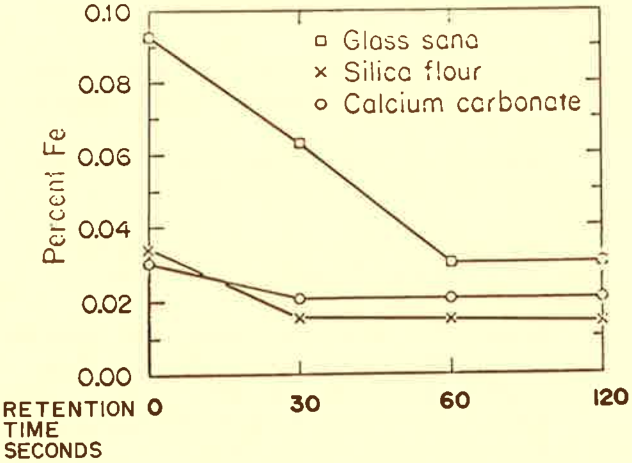

FIGURE 14

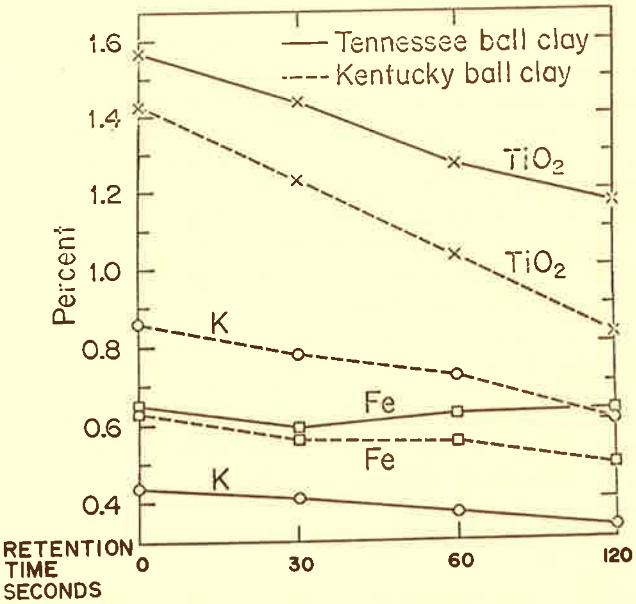

FIGURE 15

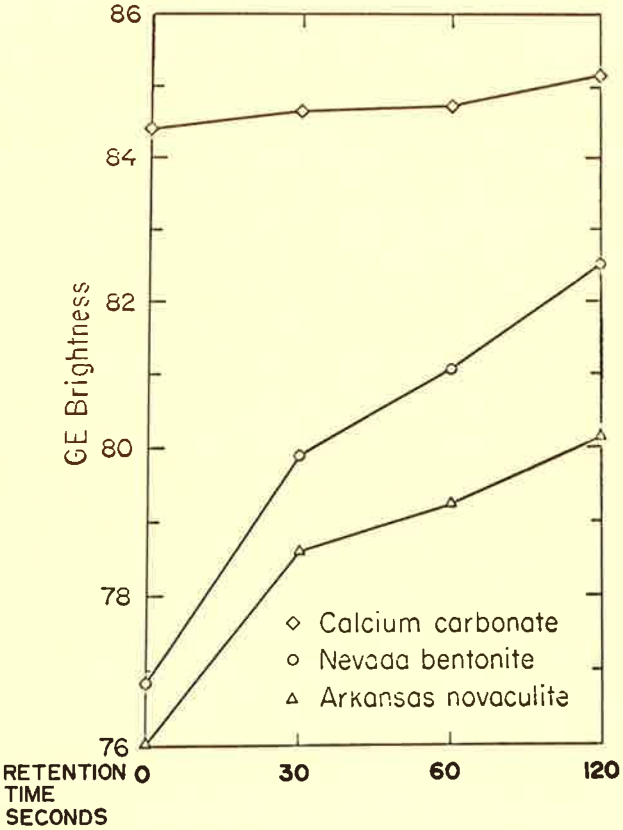

FIGURE 16

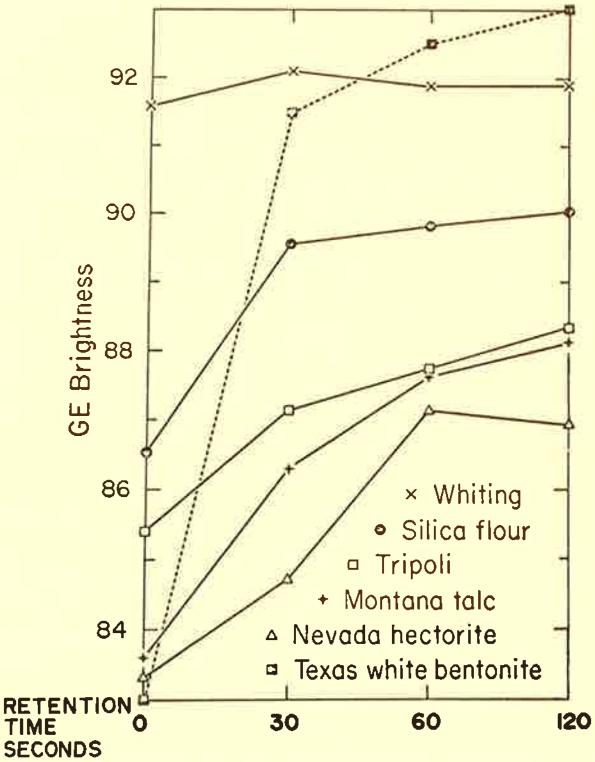

FIGURE 17

retention time at 20 kilogauss. The untreated sample is shown by A. Of this group, diatomaceous earth, sepiolite, and Montana talc show signigicant reductions of iron content.

HEMF processing of glass sand was successful in reaching optical glass standards after 60 seconds retention time (Figure 14). Treatment of Kentucky and Tennessee ball clay achieved only small reductions in iron content, but significant reductions in TiO_2 and K resulting from removal of deleterious anatase and mica, (Figure 15).

Even though Nevada bentonite (Figure 13), and calcium carbonate showed only slight removal of iron, these removals resulted in sharp and significant upgrading of brightness (Figure 15).

Likewise, Montana talc, whiting, and silica flour showed significant increases in brightness as a result of HEMF processing. Texas white bentonite gave a spectacular brightness response going from 83 brightness to 91.5 after 30 seconds, and finally to 93 brightness after 120 seconds retention time, (Figure 16).

Utilization in New Areas

The kaolin separation problem represents the most difficult application of magnetic separation ever commercialized. This technology can now be applied advantageously to beneficiation of other industrial minerals, concentration of metallic ores, desulfurization of coal, and clean up of effluent streams, Programs on magnetic separation of metallic ores, and desulfurization of coal are currently in progress at Indiana University under the direction of Dr. H. H. Murray. An NSF sponsored study on desulfurization of solvent refined coal is also under way at Auburn University under the direction of Dr. Y. A. Liu.

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Factors for Converting English Units to International System (SI) Units

The following factors may be used to convert the English units published herein to the International System of Units (SI).

Multiply English units	By	To obtain SI units
Feet (ft)	.3048	metres (m)
Miles (mi)	1.609	kilometres (km)
Square miles (mi^2)	2.590	square kilometres (km^2)
Gallons (gal)	3.785	litres (l)
Million gallons (10^6gal)	3785	cubic metres (m^3)
	3.785×10^{-3}	cubic hectometres (hm^3)
Gallons per minute (gal/min)	.06309	litres per second (l/s)
	.6309	cubic decimetres per second (dm^3/s)
	6.309×10^{-5}	cubic metres per second (m^3/s)
Million gallons per day (Mgal/d)	43.81	cubic decimetres per second (dm^3/s)
	.04381	cubic metres per second (m^3/s)

DEFINING A COMMERCIAL DIMENSION STONE MARBLE PROPERTY

Lance Meade
Vermont Marble Company
Proctor, Vermont

A few years ago there was a popular song that asked — "Where have all the young men gone?" This of course referred to the lack of young men at the end of World War I. Today through, the producer of Dimension Stone Marble, Granite, Slate or Limestone can be heard singing "—Where have all the building projects gone?"

There are a few mid-western banks putting up an addition or two, and an occasional insurance company refurbishing a lobby, but for the most part, the area in the construction industry that uses a large volume of dimension stone is in a very depressed state of activity. What is more, the domestic industry has gone through a decade of devastating Italian competition. With a diminished market and cheap imports, it is a quiet time for most domestic stone producers.

There are areas in the third World Nations, however, who have invested development funds in resource studies and have been told that marble lies waiting to be marketed in their hills. They have been reading up on the amount of cubic feet of marble that Italy has exported to the big market place of North America in the recent past, and how many dollars have been flowing back into that greatly subsidized Italian industry. These developing nations have wanted to enter the marble mining industry. These nations have since learned that it takes more than a Unido-financed report to make a potentially commercial marble deposit located far from population centers and transportation into a reality.

The question remains: what makes a marble property commercially viable? The answer could be either a good government subsidy program or an extensive marble deposit that could be economically extracted and marketed in an acceptable form to return a profit to the owners.

Discussion, here, is limited to the type of stone deposit that would yield enough uniform material to satisfy the contemporary styled large structures that were being built prior to the economic slow-down. This would exclude the many small multi-colored deposits of decorative stone that are available in the world. Traditionally marble has been used in structures that use cubic pieces of stone that have been sawn from blocks. These cubic pieces normally are as thick as they are high in dimension. This style of architecture has been duplicated somewhat in using a thinner 1 1/4" veneer and fastened directly to the structural steel of contemporary buildings. Marble also is presently being used in an 1 1/4" or a 7/8" thick veneer combined with steel and glass to give an aesthetically pleasing building.

To satisfy the architects of these kinds of structures, the stone used has to meet certain requirements. The deposit has to be extensive enough in uniformity to supply consistent material that is marketable within a very restricted range of suitable color and texture.

The geological parameters of these kinds of deposits have been very adequately described in past reports of the Bureau of Mines, the U. S. Geological Survey and individual articles. A coherent core is needed when initially investigating the property by core drilling as well as a very favorable waste-to-ore ratio with a maximum utilization of the entire stone deposit. The deposit must yield consistently uniform blocks that will produce slabs of similar color and markings and be acceptable to an architect. An abundance of cracked, cross-clouded or variably colored blocks that will fall apart in the gang saws and yield unmatchable slabs in the shops is obviously undesirable.

As Robert Power stated in his 1972 article in "Mining Engineering, ". . . most cut dimension stone is sold by the cubic foot by vertically integrated companies". A well planned mill with maximum gang-saw space utilization and a well-run, efficient shop-layout to minimize handling and maximize utilization of stock is essential for an economically viable property.

With the filling of the above parameters, the dimension stone producer can expect to have a well defined, highly feasible dimension stone property. That is of course if he is a domestic producer and the E.P.A. and MESA and O.S.H.A. regulations don't impose such costly alterations to his original plan of production that his original cash flow analysis becomes negative rather than positive. Or, if he is on foregin soil and his operations aren't confiscated for the good of the people; or if he is in Canada and isn't taxed into oblivion. If he can surmount these perils and the economy really is on the upswing, and we start building again – using stone to conserve energy and create beautiful structures and use up our accumulated block and slab inventories, then maybe we can justifiably say we have a commercial dimension stone property.

Many authors have contributed to the various techniques that can be used in the exploration of mineral deposits in general, and a few authors have been more specific in covering dimension stone. Some of these sources are conveniently listed in the bibliography at the back. In any case, at the expense of being repetitive, I would suggest the following:

Phase I — Site Selection and Production Analysis

A. Determining the potential deposits to supply graded marble of uniform color, texture and size of blocks.

  1. Literature Search
     Overview the entire list of deposits and determine the best sources based upon the above criteria.

  2. Literature Search and On-Site Visits
     Review the present active quarry areas to determine the extent of present producers expertise and sources of material.

  3. Select target areas of investigation, looking for those sources of natural stone that best answer the architect's design characteristics on current and projected buildings.

  4. Select target areas based upon geographical locations of upcoming large contracts and building projects.

  5. Review the transportation costs from the selected deposits to proposed building projects.

  6. Select the Mill and Finishing Facility location in conjunction with domestic transportation and best location for importing of supplies and materials.

B. Determining the quarry feasibility of each individual target area.

  1. Determine the geological uniformity and structures.

  2. Determine horizon of most marketable material.

  3. Determine potential percent of marketability of various horizons

  4. Determine the percentage waste due to unsoundness, poor color, and non-uniformity of marble.

  5. Study and determine the overburden and waste rock removal costs.

C. Determining costs of quarrying and plant construction of Mill and Finishing Shop to determine competitive costs with established sources in the country and current importers.

  1. From selected sites outline various operating plans based upon geological parameters of deposit, available labor, equipment, power sources.

  2. Determine finishing characteristics of production stone and develop design and layout of Mill and Finishing Shop according to block size and slab handling facilities needed.

  3. Preliminary Costs:

     a. Procurement of required operating permits.

     b. Preliminary road construction and water supply, core drilling, test block, extraction.

     c. Quantify volume available from deposit measured against projected demand volume and volume of waste needing to be moved.

     d. Determine potential productivity factors, establish time parameters to develop deposit into full production cycle (establish site preparation costs and project potential quarry production costs, design plant facilities in conjunction with volume supply and projected demands).

  4. Site Preparation Costs:

     a. Prepare roadway systems, quarry buildings, water and power supply.

     b. Procurements to set up quarry equipment.

     c. Establish plant site and install buildings, procure and install Mill and Finishing equipment.

     d. Establish sources of supply of various supplies (abrasives, cutting tools, handling equipment).

5. Development Costs:

    a. Removal of overburden and waste rock

    b. Develop Keyways across production layers — open production zones.

    c. Initiate gang sawing and preliminary coping and finishing.

    d. Review production scheduling to establish anticipated lead times for production cycles.

Phase II — Market Analysis

D. Determine a five year and ten year projected building projects within the country.

1. Determine present suppliers and potential suppliers, both domestic and foreign.

    a. Study past, current and projected volumes of variety of stone of each.

    b. Determine present pricing and cost parameters of present suppliers.

2. Determine quantities of concrete, structural steels, and natural stones anticipated to be used within this time frame.

3. Determine the quantities of materials that will be imported versus that which can be supplied domestically. Identify the factors that will alter present development trends. Determine who the present Architects and Engineering Consultant Groups are. Determine the primary contractors operating and potential contractors operating within the market area.

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SOUTHEASTERN CERAMIC RAW MATERIALS

J. L. Pentecost
School of Ceramic Engineering
Georgia Institute of Technology
Atlanta, Georgia

Traditionally, the ceramic industry has used primarily mineral raw materials, often native materials with little beneficiation other than grinding and sizing. While many new ceramic products depart widely from this tradition, a large segment of the industry is still closely tied to its mineral raw material supplies.

Consider first the three large tonnage consumers of minerals for producing ceramics: the brick industry, the glass industry, and the portland cement industry. These industries thrive nationwide largely because clays, limestone, and sand deposits are abundant. The southeast is particularly fortunate in its native clays, shales, limestone, and sand deposits. The brick industry in this area, for example, has been the leader in innovative production techniques. There are also approximately 20 cement plants in the southeastern U.S. with some of the most modern technology for dust control and close-cycle material handling represented in these plants.

The glass industry deserves special note. Over 70 percent of most compositions is silica sand, which must be a minimum of 99 percent $Si0_2$ with less than 300 parts per million Fe_20_3 and 3 parts per million Cr, with a minimum of 85 percent plus 100 mesh, 0 percent plus 20 mesh. These are specifications for a material selling for $3.50 per ton at the plant. These specifications are necessary for ordinary clear container glass, which is most often used; for colored glasses the specifications are relaxed somewhat. To avoid confusion, most plants purchase only a single grade of sand.

The glass industry developed in the north and midwest, near population centers, using the St. Peter Sandstone (Illinois) and the Oriskany Sandstone (Virginia, Pennsylvania, and West Virginia). Both of these are excellent deposits and have served the industry well. Twenty-five years ago there were only a very few glass plants in the southeast so most of the glass we used was shipped into the area. With the growth of the soft drink industry, along with "no return" bottles, rapid expansion of southeastern markets occurred. This resulted in the development of excellent glass sand deposits in southern Georgia and in Florida, and today a wide variety of containers and glass products are made in the south for southern markets.

When considering the value of brick, glass, and portland cement on a per pound basis, the pressure for cheap raw materials is obvious. Brick sells for just 1¢ per pound. To mine the clay, do at least minimal benficiation, maintain constant water content, formulate the desired colors, shape, stack, dry, fire to 2000°F, package, move this heavy product a couple of times and sell the finished product profitably at $25/ton requires profound engineering and very large facilities. It is generally uneconomical to process less than 400 tons of brick per day.

Portland Cement is similar in value. The clay and limestone must be ground together, fused to a clinker at 2800°F (somewhat higher temperature than brick), ground to 325 mesh fineness, and sold for just over 1¢ per pound also.

By comparison, glass containers sell for about 10¢ per pound, with the cost of chemicals (soda ash), the accurate forming, and the inspecting all reflected in this cost. Even fiberglass insulation is only 17-24¢ per pound, and fiberglass filaments for textiles is approximately 35¢ per pound. With such value on a per pound basis, it is not surprising that these are large industries.

Other segments of the ceramic industry using primarily mineral raw material include the tile industry, sanitary ware industry, and the dinnerware industry. Kaolin clays, ball clays, talc, silica, pyrophylite, and feldspar are among the major raw materials utilized by these industries. Low iron content and low alkali impurities are usually desirable. The refractories industry also depends on native clays, magnesite, (often from seawater) dolomite, chromite, silica and alumina minerals. While much of the technical ceramic industry uses refined raw materials, the chemicals used are usually directly related to processed mineral raw materials.

Some of the specific minerals and their uses in the southeast deserve mention. Georgia and South Carolina's abundant kaolin clay and North Carolina's primary kaolin are widely used in dinnerware, refractories, sanitary ware, and tile. Ball clays from Kentucky and Tennessee are also necessary in most compositions for improved farming. In these same compositions are feldspars from North Carolina (about 70 percent of native production) and Monticello, Georgia. Feldspars are also important constituents in glass, with aplite and nepheline syenite also included in this group of fluxes. About 500 thousand tons are used annually in ceramic, glass and porcelain. Ceramists usually distinguish between high potassium and high sodium feldspars for specific uses. High potash feldspars are generally less available than soda or mixed feldspars.

Limestone for cement is produced widely since there is relatively little restraint on the iron and trace impurities it may contain. In contrast, limestone used in glass and ceramics must meet rigid specifications concerning iron content (less than 0.1 percent Fe_2O_3).

Ceramic talc, a major constitutent in steatite electrical insulators and wall tile, is produced primarily in Texas, California, Montana, and New York. Ground pyrophylite is also used in these products. Though Georgia has proven talc deposits several times the size of those being worked elsewehere, its iron content is too high for ceramic uses and would have to be removed. While such processing is practical, these deposits remain to be exploited as ceramic raw materials. Spodumene from Kings Mountain, North Carolina, is used as a major constitutent of many low expansion ceramics. As it occurs, this spodumene is too high in iron until beneficiated by hot chlorine leaching, volatilizing the $FeCl_3$.

TiO_2 is a constituent used in enamels and glazes to opacify the glass. It is a major constituent in special dielectrics and piezoelectrics. TiO_2 is refined from southeastern deposits of heavy mineral sands; currently only Florida deposits are being mined.

The ceramic industry is also a large consumer of fuel, primarily oil and natural gas. All ceramic products are heated above 1000° and often above 2000°F. The continued availability of these fuels is important, but conversion to coal could be accomplished in much of the industry, if this became a necessity.

THE FUTURE

As energy costs increase, some product displacements will occur due to resulting cost differentials; however, most ceramic producers are confident that there will be a reasonable balance and ultimately, due to bountiful raw materials, ceramic products will fare well in spite of their energy component. The southeastern market continues to grow as income levels increase and markets increase. This optimism provides the impetus in Georgia, South Carolina, and Florida for the continued education of ceramic engineers to serve this industry. Such trained personnel are essential for continued growth of the industry, and the ceramic industry provides excellent employment opportunities for the future.

The ceramic industry is a demanding customer of mineral suppliers; (this has been so in the past and will continue in the future). Improved mineral raw materials will be necessary for our improved products and dynamic technology, As we must control our products more closely, raw material will receive increasing scrutiny. We will need new material in large quantity and, with transportation costs an ever present concern, nearby mineral deposits will be developed with the lower transportation cost providing part of the differential needed to support the beneficiation of lower grade deposits and to justify the capital investment.

It is most disturbing to see the ceramic industry press mineral suppliers to a point of cost competition where inadequate research in beneficiation and exploration can be supported out of the price obtained for the mineral. This is ultimately destructive to both industries which are interdependent and ought to work together for improved ceramic products.

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ECONOMIC GEOLOGY OF THE GEORGIA MARBLE DISTRICT

W. Robert Power
Georgia State University
Atlanta, Georgia

Introduction

The Georgia Marble District, located in Pickens and Gilmer Counties, Georgia, has produced marble for more than one hundred and thirty years. For the first one hundred years practically all production was dimension stone, but since World War II a wide variety of crushed and ground marble products have been produced and sold. These include terrazzo and other decorative chips, agricultural lime, fillers, extenders, and similar products. Today the crushed and ground products account for a higher volume and dollar value than does dimension stone.

The Georgia Marble District includes three outcrop belts of the Murphy marble in Pickens and Gilmer counties, Georgia (Fig. 1). The largest is a hook-shaped area that occupies the valley of Long Swamp Creek at Tate, Georgia, and its East Branch near Marble Hill, Georgia. It was called the Marble Hill Belt by Bailey (1928) and the Tate Belt by Fairley (1965, p. 23). In this paper it is called the Tate-Marble Hill Belt. A second belt lies on the east side of Long Swamp Creek about two miles east of Jasper. It was called the Long Swamp Creek Belt by Bailey (1928) and by Fairley (1965). However, the ridge from which the stone is mined is locally known as Cove Mountain and the stone is known commercially as Cove Mountain stone. Therefore the outcrop belt is referred to in this report as the Cove Mountain Belt.

The third belt lies on the east side of Talona Creek at Whitestone, Georgia. It lies partly in Pickens and partly in Gilmer County. It is referred to as the Whitestone Belt in this report.

Parts of the three belts are now being mined by the Georgia Marble Company. Crushed and ground products are produced from all three. Dimension stone is produced from the Tate-Marble Hill Belt only.

Regional Geology

The Murphy Marble lies within the Murphy Marble Belt, a sequence of lower Paleozoic metasedimentary rocks occupying a long synclinal trough that extends from Bryson City, North Carolina to Canton, Georgia. The sequence was first described by Keith (1907) who described the rocks near Murphy, North Carolina. The marble and enclosing schists were traced southward into Georgia by LaForge and Phalen (1913), Bailey (1928), Hurst (1955), and Fairley (1965). Power and Forrest (1973) discussed the stratigraphy and correlations within the Murphy Belt.

The Murphy Marble overlies the Brasstown Formation (of Hurst, 1955) which consists of thinly bedded mica schist and feldspathic quartzite. The Brasstown Formation makes excellent flagstone and is quarried locally for that purpose.

There is some confusion regarding the nomenclature of rocks overlying the Murphy Marble. Keith (1907) described the Andrews Formation as a thin calcareous schist that overlies the marble. Hurst (1955) redefined the Andrews Formation to include several hundred meters of pelitic schist above the calcareous schist. Fairley (1965) introduced a new name — the Marble Hill Hornblende Schist — for a thin calc-schist that overlies the Murphy Marble at Tate and assigned overlying garnet-mica schist to the Andrews Formation. Power and Forrest (9173) pointed out that the original definition of the Andrews Formation was for a thin calc-schist that overlies the Murphy Marble and corresponds to the Marble Hill Hornblende Schist. The succeeding garnet-mica schist should therefore be called the Mineral Bluff Formation following Hurst (1955).

Whatever the nomenclature all writers have agreed upon the succession of lithologies. The succession with names used in this report given in parentheses is from oldest to youngest: (1) thinly bedded mica schist and feldspathic quartzite (Brasstown Formation), (2) marble (Murphy Marble), (3) calc-schist (Marble Hill Hornblende Schist), (4) garnet-mica schist (Mineral Bluff Formation).

Fairley (1965, 1969) reported that the Murphy Marble is not a continuous layer throughout the Murphy Marble Belt, but rather is discontinuous and locally interfingers with the enclosing schists. Power and Forrest (1973) agreed and interpreted the marble as original reefs.

Fairley (1965) distinguished between relatively pure calcite marble and dolomite marble, and impure micaceous marble and calc-schist. He included the former within the Murphy Marble, but considered the latter to be a calcareous facies of the Mineral Bluf (i.e. Andrews) Formation. Bayley (1928) had included both within the Murphy Marble.

The Murphy Marble as defined by Fairley occurs principally in two areas named the Tate Marble Hill Belt and the Cove Mountain Marble Belt. All commercial production of marble from the district has come from these two areas plus a third area outside the Tate Quadrangle at Whitestone, Georgia.

Tate-Marble Hill Belt

The Tate-Marble Hill Belt crops out in a hook-shaped area that occupies the valley of Long Swamp Creek at Tate, Georgia and its East Branch near Marble Hill, Georgia (Fig. 1). It is the area first exploited for marble and has produced practically all the dimension stone in the district. Today dimension stone is produced from the main valley of Long Swamp Creek at Tate; crushed and ground marble is produced from Marble Hill along the East Branch or "barb" of the hook.

Lithology

The marble is a medium to coarse-grained calcite marble that is magnesian or dolomitic in places. The dolomitic marble is generally finer grained. Individual crystals in the calcite marble typically are five to ten millimeters across

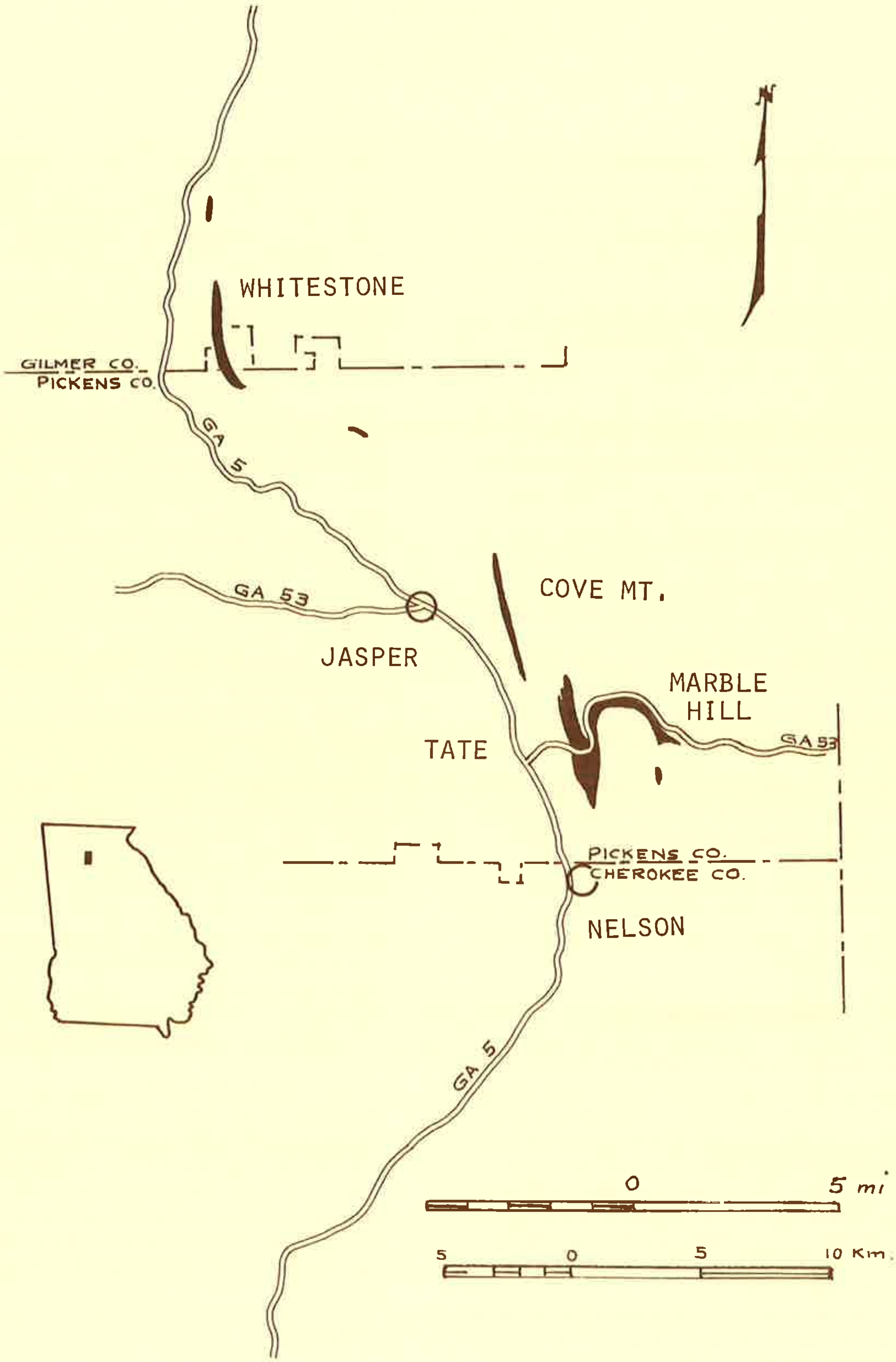

FIGURE 1. Location map of Georgia Marble District. Marble outcrops shown in solid black.

whereas crystals in the magnesian marble are typically less than two or three millimeters. The marble contains varying amounts of accessory minerals, chiefly graphite, mica, amphibole, pyrite, and epidote. These typically occur disseminated in layers or "veins" that give the marble its character as ornamental stone. A number of commercial varieties have been named based on color and nature of the veining. Mapping these varieties has proved useful in working out the local stratigraphy and structure of the marble. The more important varieties are described below.

Cherokee. Cherokee marble has a white background and gray markings. The markings are caused by disseminated graphite, black mica, and finely disseminated pyrite. They occur as layers or bands which are in places straight, but more commonly are disrupted and contorted, appearing as irregular veins and blotches. Markings make up as little as one percent to as much as fifteen percent or more of the rock. Cherokee marble is the principal variety of building stone quarried from the district.

Messotint. Messotint marble has a gray background with dark gray or black markings. Accessory minerals and the character of markings are the same as for Cherokee marble. The gray background is caused by finely disseminated graphite and pyrite.

Creole. Creole marble has a white background with black markings. Accessory minerals are the same as for Cherokee marble, but more heavily concentrated. The amount of dark colored marking is much greater, commonly exceeding 50 percent of the stone.

Silver Grey. Silver Grey marble is a uniform gray variety in which the gray color comes from finely disseminated graphite and pyrite. It is used extensively for monuments.

Etowah Pink. Etowah Pink marble has a light to deep pink background with green to greenish black markings. Accessory minerals are chiefly green biotite, but include amphibole and epidote. The pink color comes from hematite that is disseminated along grain borders. The markings are shaped similarly to those in Cherokee marble. They generally make up 10 to 30 percent of the stone.

Golden Vein. Golden Vein marble has a white background with golden brown markings caused by accessory phlogopite and less commonly, colorless to pale tan amphibole. The more highly prized Golden Vein has only the golden brown phlogopite set in a white background of calcite crystals. However, in places the layers containing the phlogopite also have a grayish background color, probably caused by finely disseminated graphite. In places Golden Vein type marble contains patches or layers with green mica and amphibole rather than tan and brown. This makes a striking looking stone, but it is not common enough to have been quarried and marketed as a separate variety.

White Georgia. White Georgia is nearly pure white marble with only a trace of color resulting from widely scattered grains of phlogopite and rare colorless or pale tan amphibole. It is the most highly prized marble in the district, but is relatively rare.

Rosepia. Rosepia is a fine-to medium-grained magnesian marble that is pale pink in color with amber colored streaks or "veins." The amber color is caused by disseminated grains of phlogopite.

Stratigraphy

The stratigraphy of the marble is best displayed at Marble Hill (Fig. 2) where the layering is regular, the dip rarely exceeds 20 degrees, and several deep core holes have penetrated the entire formation. The stratigraphy was first described by Reade (1965).

Detailed mapping and core drilling in the New York Mine (Fig. 2) has resulted in the recognition of many units, some of which are very thin and of very limited aerial extent. Other units grade into each other along the stratification and probably reflect local facies changes in the original sediments. In order to summarize the vertical and horizontal changes it is convenient to distinguish five units which will be called: (1) magnesian marble, (2) middle schist, (3) quartz sulfide marble, (4) silicate marble, and (5) graphitic marble or "Cherokee" type. All units are recognized at Marble Hill, but only the silicate marble (4) and graphite marble (5) have been recognized at Tate.

The contact between marble and the underlying schist (Brasstown Formation) is abrupt and apparently conformable. The schist is a medium to coarse-grained feldspathic, quartz, biotite schist.

Magnesian Marble. The magnesian marble is the lowermost marble unit. It is characterized by high magnesian content and dolomite layers. It consists of alternating layers of fine-grained white, gray, and tan dolomitic marble and coarse-grained white to light gray marble. Some layers contain abundant sulfide minerals. Rosepia marble comes from this unit. The thickness is variable, ranging from about 18 to 50 meters under the New York Mine, but it thins out and disappears to the southeast.

Middle Schist. A thin layer of hornblende, biotite, calcite schist overlies the magnesian marble. In hand specimen or drill core it is hardly distinguishable from the Marble Hill Hornblende Schist which overlies the Murphy marble. The middle schist ranges in thickness from about one and one-half to five meters (5 to 16 feet).

Quartz, Sulfide Marble. Quartz-bearing, gray marble containing disseminated pyrite overlies the middle schist. Pyrite also occurs in fractures that cut the regular stratification. No commercial stone comes from this unit.

Silicate Marble. The silicate marble unit contains the commercial varieties Etowah Pink, Golden Vein, and White Georgia. It is white, light gray, or pink marble with accessory silicate minerals (mica, amphibole, and epidote) that generally are disseminated in layers giving the rock a striped appearance. The stripes or layers commonly range in thickness from one to about fifteen centimeters (.5 to 6 inches). Colored layers contain anywhere from a few percent to as much as thirty percent or more accessory minerals. In extreme cases dark colored layers approach calc schist in composition.

The chief accessory mineral is mica which may be golden brown phlogopite or green biotite. Amphibole ranges in color from very pale tan to bright green. Epidote is less common, but is present in some layers. Quartz, pyrite, norbergite fuchsite, and chlorite occur rarely. Where the background color is white the accessory minerals are generally pale tan or brown in color. Where the background color is pink the accessory minerals are green. Green mica and amphibole occur less commonly in white marble.

The silicate marble is more than 75 meters (250 feet) thick in the New York Mine. The lower half is much darker or more heavily striped than the upper half of the unit. The upper half contains an elongate lense or prism-shaped body of almost pure white calcite marble. The commercial varieties Golden Vein and White Georgia are produced from this area. Pink marble is common in the lower half of the unit and becomes more abundant west of the New York Mine.

Pink silicate marble is also exposed in the central part of Long Swamp Valley at Tate, where it is quarried as Etowah Pink. Brown, striped Golden Vein marble occurs on both sides of the Etowah Quarry.

The silicate marble is thickest in the New York Mine, but it thins rapidly westward and is less than 100 feet thick a few hundred feet west of the mine.

Graphitic (Cherokee) Marble. The graphitic marble contains disseminated pyrite, black mica, and hornblende as well as graphite. Cherokee, Messotint, Creole, and Silver Grey Marble all come from this unit. The unit is less than 30 meters (100 feet) thick at the New York Mine, but it thickens westward and at Tate it is the dominant marble exposed in the valley. The structure at Tate is too complicated to work out detailed stratigraphy, but the darker varieties (Creole and Messotint) tend to be concentrated along the west margin of the valley. They grade eastward into typical Cherokee Marble and finally into silicate marble (Etowah) in the east central part of the valley. Cherokee Marble has been quarried from both Marble Hill and Tate, but Creole, Messotint, and Silvery Grey, from only Tate.

Transition Marble. The Murphy Marble is overlain by the Marble Hill Hornblende Schist. However, there is a transition zone about 15 meters (50 feet) thick between typical Cherokee type marble and typical hornblende schist. This consists of alternating layers one to ten centimeters (4 inches) thick of white marble and calc-silicate schist. The calc silicate layers contain hornblende, biotite, epidote, and calcite. They become increasingly silicate rich and darker in color upsection and grade into hornblende schist containing only minor epidote, biotite, calcite, and other minerals.

Structure

Layering. Layering is well developed in the marble at Marble Hill. The silicate marble shows it particularly well where colored layers containing abundant accessory minerals alternate with white layers of pure white marble. Similar layering is less spectacularly displayed by other units. Individual layers range in thickness from a centimeter (.4 inches) or less to several meters (feet). They can be traced tens of meters (feet) in the mine workings and groups of layers can be correlated several thousand meters (feet) by core drilling.

The present mineral assemblages are metamorphic in origin, but they undoubtedly reflect original chemical differences in sedimentary layering.

Layering is also present at Tate, but it is much distorted by deformation. Tight folds a meter (3 feet) or less in amplitude are common. Layers are discontinuous so that it is seldom possible to trace a unit more than a few meters (feet). Distorted and disrupted dark layers make the typical "veining" of the marble. Abundant barb-shaped veins probably are the hinges of small disrupted folds. These typically plunge about 15-20 degrees down the valley to the southeast.

Foliation. Foliation resulting from the sub-parallel alignment of micas and amphiboles in dark colored layers is almost invariably parallel to the layering. Petrographic studies by Fairley (1965) showed flattened calcite crystals and parallelism of calcite twin lamellae.

Folds. Several orders of folds exist in the area. The largest or first order folds include an open south plunging syncline at Marble Hill and a tight overturned southeast-plunging anticline at Tate. These two folds produce the hook-shaped outcrop pattern of the belt.

At Tate second order folds ten to fifty meters (30 to 150 feet) across can be seen in quarry walls. These are flowage (passive slip) folds that are greatly thickened along the axes or hinges of anticlines. The axial planes generally dip steeply east. Some of the best quality dimension stone produced in the area occurs in the thickened hinge areas of these folds and the quality stone can be followed down plunge to the southeast.

Third order folds probably show the nature of the deformation best. Individual layers are drawn out, thinned, and commonly disrupted along the limbs, but are greatly thickened at the axes where they form hook or barb-shaped features in the rock. The plunge of the third-order folds is the same as that of first and second order folds.

In the New York Mine at Marble Hill the structure is a very broad, shallow syncline that plunges about 15 to 20 degrees south. Second and third order folds have not been found within the mine; however, an elongate lense or prism-shaped body of Georgia White Marble lies with its long axis down the plunge. Correlation of layers or units is easier and more consistent southward down the plunge than in the east-west direction. Immediately east of the mine there appears to be a steep anticline that is overturned to the west and thrust faulted from east to west. There is further evidence from drill holes and mapping that a zone of tight folding exists immediately west of the mine.

Faults. Two faults have been recognized from mapping and core drilling. Both are east-dipping thrust faults that apparently grade along strike into overturned anticlines. One occurs southwest of the New York Mine and the other along the east side of the valley of Long Swamp Creek at Tate, east of the dimension stone quarries (Fig. 2).

Structural Interpretations. Previous workers have proposed two different structural interpretations to account for the distribution of marble and enclosing schists in the Tate-Marble Hill Belt. Bailey (1928) proposed that older rocks (his Carolina Gneiss and Roan Gneiss) had been thrust over the marble from the east leaving the marble at Marble Hill as a partial window. Bailey (1928, p. 55, 56, 122) recognized no clear criteria for the fault within the Tate Quadrangle, but apparently considered it a necessary extension of a fault previously mapped in the Ellijay Quadrangle by LaForge and Phalen (1913).

Fairley (1965) interpreted the structure as a syncline overturned to the west and later subjected to a second period of deformation that twisted the eastern limb of the

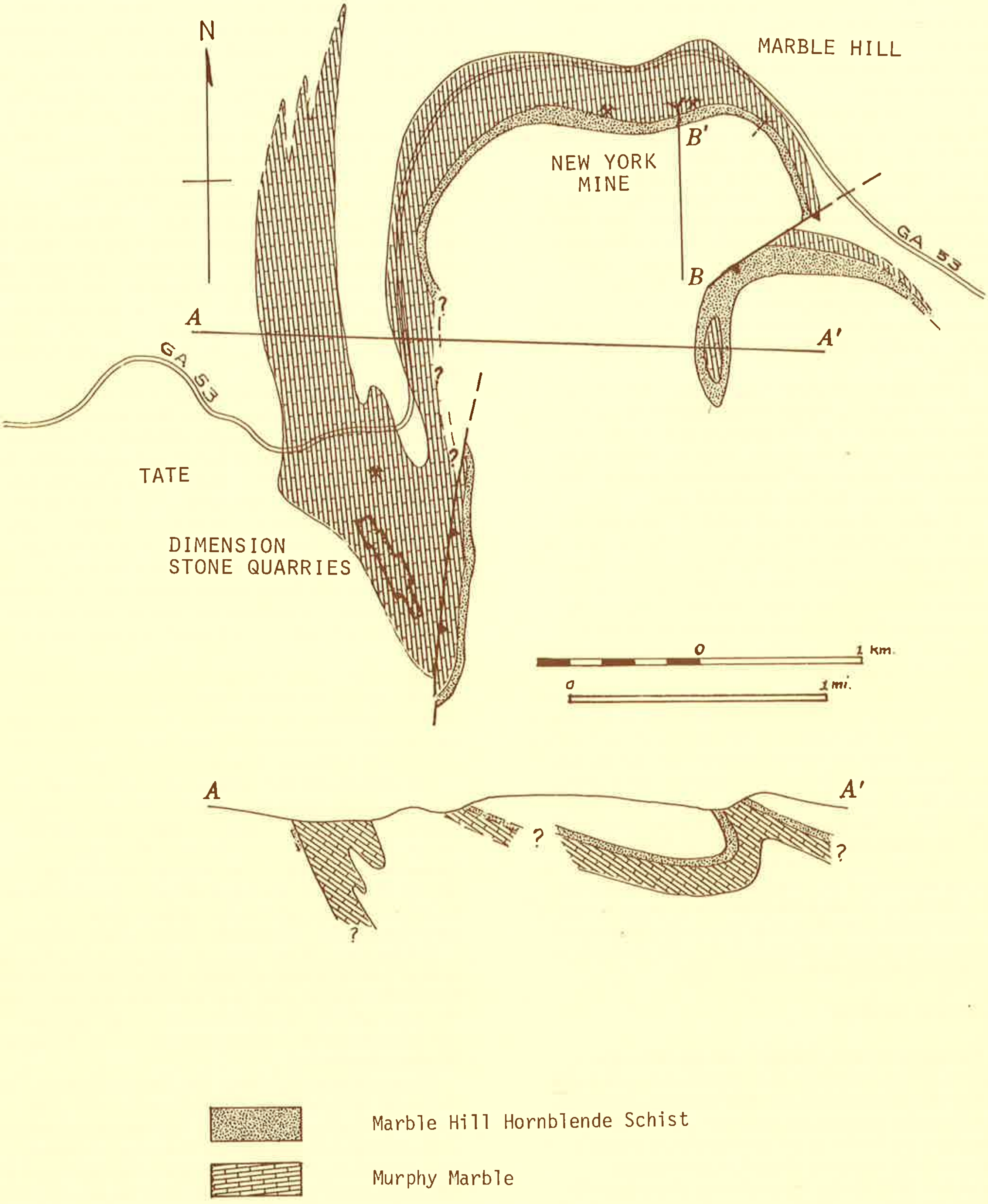

FIGURE 2. Geologic map and section of the Tate-Marble Hill Area.

fold into the sickle or hook-shaped map pattern displayed at Marble Hill. He further suggested that the marble passes laterally into schist by facies changes and interfingering so that, in effect one limb of the syncline is exposed as marble, the other as schist.

Earlier structural interpretations of the marble were part of larger pictures that involved regional patterns, the mapping of schist units, and long range correlations. The difficulty of the mapping and correlation is attested to by Bailey's statement (1928, p. 56) that "there seems to be no way of distinguishing between rocks belonging in the Great Smoky Formation and those belonging in the Carolina Formation." Lithologic descriptions of the various schist units sound monotonously alike and it is difficult to find clear-cut criteria for distinguishing them. Except for the marble and the overlying hornblende schist there seems to be no good marker bed within the Murphy Belt rocks in the Tate-Marble Hill area.

In this report a structural interpretation of the Murphy Marble and overlying Marble Hill Hornblende Schist is given independently of the enclosing schists. It is assumed that the marble and overlying hornblende schist are derived from original sediments that had lateral continuity within, but not necessarily outside the Tate-Marble Hill area.

The interpretation is summarized in the cross-section (Fig. 2). Long Swamp Valley at Tate is underlain by a south plunging anticline. The hinge of a second anticline is exposed at the Amicalola Quarry about one mile east. The area between is underlain by a broad syncline interrupted by minor folding. Both anticlines are overturned to the west and are cut by thrust faults that strike and dip sub-parallel to the axial planes of the folds.

A key to this interpretation is the rocks at the Amicalola Quarry. Bailey's map (1928) shows the marble at this quarry to be completely surrounded by hornblende schist. Fairley's map (1965) shows the hornblende schist on the west side only. Recent core drilling at the Amicalola Quarry shows that the hornblende schist does indeed crop out on both sides of the marble and further that the schist on the west side dips eastward demonstrating that the fold is overturned.

A small valley runs north from the Amicalola Quarry to join the west branch of Long Swamp Creek at Marble Hill. Core drilling on the west side of this valley shows marble at least 200 meters (650 feet) lower in elevation than marble on the east side at the Amicalola Quarry. This can only be explained by a steep fold or fault, or both.

Economic Geology

The Georgia Marble Company operates mines and quarries for the production of dimension stone and a wide variety of crushed and ground stone products. Dimension stone has been produced for more than a hundred years. Significant production of crushed and ground products began with the formation of the Calcium Products Division in 1947. Today the production of crushed and ground products exceeds that of dimension stone in both volume and value.

Dimension Stone. The Georgia Marble District is famous for dimension stone which was used for the state capitol buildings at Rhode Island, Minnesota, the New York Exchange and many other famous buildings. Since 1947 the production of dimension stone at Marble Hill gradually diminished and finally ceased. Today all dimension stone is produced from open quarries in Long Swamp Valley at Tate.

Most production is of Cherokee Marble, but Messotint and other graphitic varieties are also produced. There is no current production of Etowah Pink, Golden Vein, or White Georgia as dimension stone; however, these varieties are produced in a unique process for split-face ashlar.

Dimension blocks averaging about 16 tons are cut by line drilling using conventional quarry bars on flat floors. Blocks are broken free with wedges and feathers (Power, 1975) and moved across the quarry floor with tractors and loaders to where they can be lifted out with fixed derricks. The blocks are then graded and sent either to storage, the mill, or to waste. Some of the "waste" blocks can be partially salvaged for split-face ashlar, but many are a total loss and must be dumped.

Quarry blocks may be rejected for color or, more commonly, structural unsoundness. Most dimension marble for buildings is cut into thin slabs (7/8 inch to 1-1/4 inch thick) for use as veneer or in curtain wall construction. No cracks, fractures, or other weakness along which the slabs can break or part can be tolerated. It is important to the producer that such weakness be discovered before money is wasted in sawing the block. The result is that commonly more than one-half the blocks removed from the quarry are rejected. When this loss is combined with kerf cuts and other inevitable losses it is common that only 25 percent or less of the stone removed from the quarry is actually sent to the mill.

Some of the rejected blocks can be partially salvaged for split-face ashlar. Blocks are sawed into slabs of modular thickness (generally 2-1/2 inches or 5 inches thick), which are then split into oblong slabs that can be laid brick-like in courses with the broken face of the slab showing.

The price of split-face ashlar is such that it is uneconomic to quarry blocks solely for the production of split-face and therefore production is viable only from blocks that are "waste" as dimension stone. However, the demand for Etowah Pink, and White Georgia ashlar is so high that Georgia Marble Company has developed a special technique for producing these varieties even though there is no current production of dimension stone in these varieties.

Circular diamond saws mounted in a special frame cut a series of parallel grooves or kerfs in the quarry floor spaced 2-1/2 inches or five inches apart and about four inches deep. The stone between the kerfs is broken out and becomes split-face ashlar. Pink stone is produced at Tate; the White Georgia at Marble Hill.

Calcium Products Division. The Calcium Products Division of Georgia Marble Company produces crushed and ground marble at Marble Hill. Two abandoned and one active underground mine are in a lense or prism-shaped layer of stone approximately 60 meters (200 feet) thick and 600 meters (2000 feet) wide (east-west) that dips southward about 18 degrees. The mines are of the room and pillar type. The active New York Mine is multi-level with stacked pillars (Fig. 3).

The older workings have random pillars and varying room height. Current mining plans call for 12 meter (40 foot)

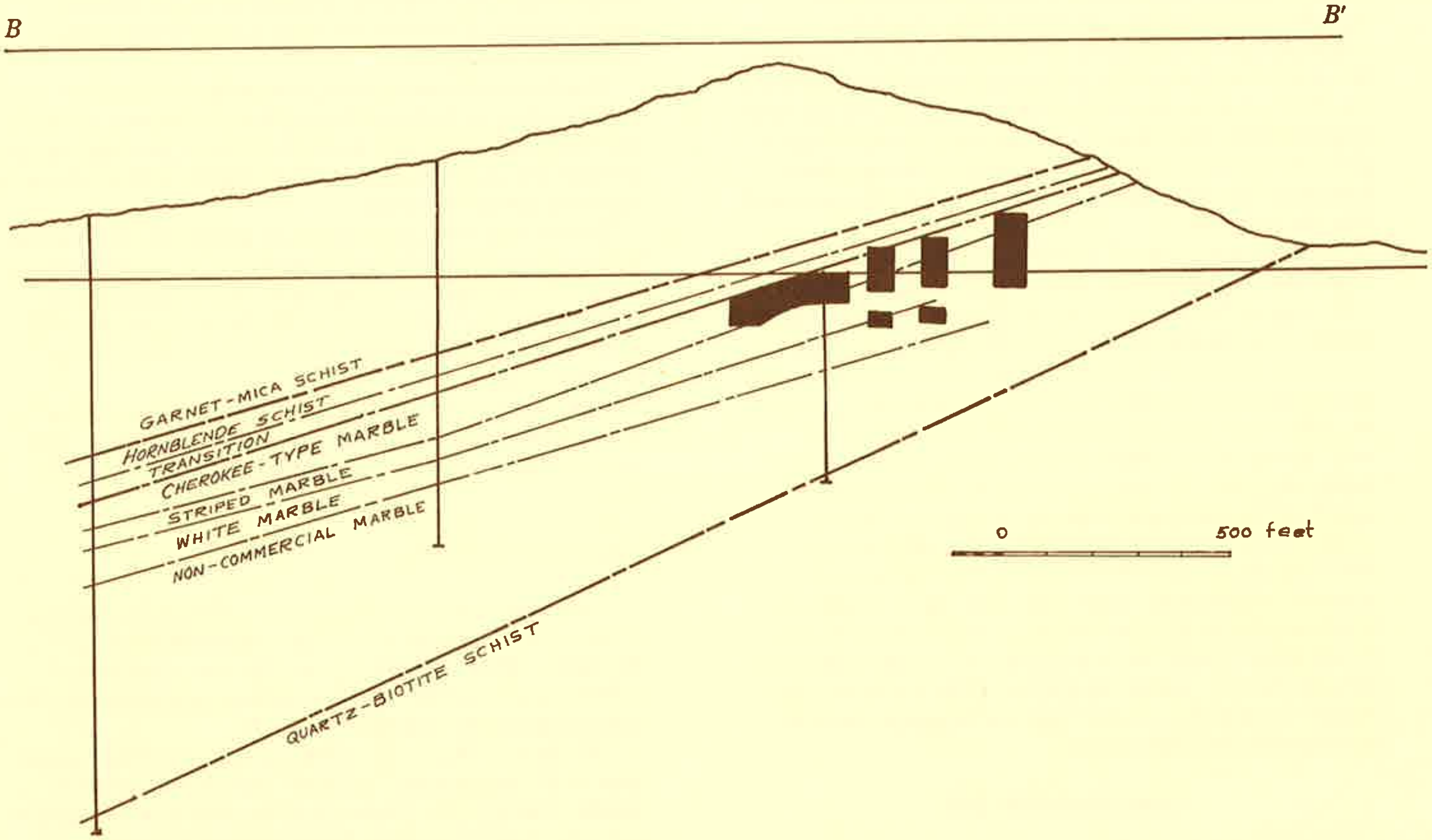

FIGURE 3. North-south cross section through New York Mine.

pillars on 24 meter (80 foot) centers lined up along strike. Initial openings are 7.5 meters (25 feet) high and plans call for lifting floors until the total room height is 36 meters (60 feet). Succeeding levels will be spaced 30 meters (100 feet) apart thus leaving 24 meters (40 feet) of stone between levels. Because of the dip of the stone only two-thirds of each level will directly underlie the level above. Access to deeper levels will be by minus 10 percent ramps. All haulage at the present time is by diesel trucks.

The stone varies in color and quality both across and along the layering, but chiefly across. Each level will extend from foot wall to hanging wall thus ensuring accessibility to all grades of stone.

Stone from the mine is delivered to one of three primary crushers feeding three plants or plant compexes. One complex manufactures a complete range of products requiring high reflectivity and whiteness; a second produces dry-ground products with less stringent color requirements; the third produces screened products only and has the least stringent color requirements. Because the footwall rock is generally off-color marble it is always possible to obtain additional "no color standard" stone by extending any level into the foot wall.

Products are produced and sold on the basis of size and brightness (i.e. whiteness). The division produces marble in sizes ranging from less than ten microns to boulders weighing several tons. Boulders and pebble-size particles are sold for decorative landscape purposes. Finer screened products are used in synthetic marble, floor tile, welding rods, aggregate, and other uses. Ground products are used as fillers and extenders in paint, rubber, plastics, putty, chewing gum, paper, and a host of other products. As grinding and classification techniques improve new uses are found.

Ground products are produced dry in roller or tube mills and wet in ball mills. Generally the finer ground products require higher brightness. The higher brightness is achieved partly by controlling the feed, but partly also by the grinding itself because there is an inverse relationship between particle size and brightness. Price is also inversely related to particle size, the finest and brightest material commanding the highest price.

Cove Mountain Belt

The Cove Mountain Belt lies on the east side of Long Swamp Creek about 3 km (two miles) east of Jasper, Georgia. Dolomitic marble crops out along a 5 km (three mile) strip on the east valley wall and dips about 20 to 30 degrees east. The layer reaches a maximum thickness of nearly 100 meters (300 feet) in the middle of the belt, but thins to the north and south finally pinching out.

Schist typical of the Brasstown Formation encloses the marble. Bailey (1928) and Fairley (1965) suggested that the belt is an infolded, isoclinal syncline because it is enclosed by similar schist both above and below the marble. Fairley also found possible graded bedding on the east side of the belt suggesting overturned beds.

Core drilling shows calcite marble in the central layer at several places. This is consistent with the synclinal hypothesis if the original stratigraphic suggestion was similar to that in the Tate-Marble Hill Belt, that is, calcite marble overlying dolomitic marble.

At the south end of the belt the layer is split by a ten meter (30 foot) layer of schist. This could be interpreted either as interfingering of original layers or as an overlying schist in the core of a syncline. Because the marble and schist clearly interfinger at the northern end of the Tate-Marble Hill Belt several kilometers (miles) south (Fairley, 1969) the author prefers this interpretation for the interlying schist at the Cove Mountain Belt.

Lithology

The marble is distinctly layered in most places. White to light gray, fine-grained, dolomitic marble is interlayered with brown mica schist and gray, medium to coarse-grained, calcite marble. Accessory minerals include brown mica, amphibole, and quartz.

The dolomitic marble is most abundant. It occurs in layers ranging in thickness from a few centimeters to nearly ten meters (30 feet). The thickest and most persistent layers are near the upper contact of the formation and are the only layers mined commercially.

Brown, calc-mica schist occurs in paper-thin laminae and layers up to a meter (3 feet) in thickness. In places brown mica is also disseminated in the dolomitic layers giving them a brown or tan tint. Schistosity is parallel to layering and in places the rock splits readily into thin slabs along planes of schistosity.

Gray, medium-to coarse-grained marble is interlayered with dolomitic marble in the central part of the belt. This rock strongly resembles the graphitic marbles of the Tate area.

Mines and Quarries

Dimension stone was quarried from the Lincoln quarry at the north end of the belt in the nineteenth century (McCallie, 1907). The quarry was inactive when Bailey (1926) studied the area and has not been worked since. It is now overgrown by vegetation.

The Georgia Marble Company operates an underground mine in the belt due east of Jasper. Only white dolomite marble is mined. It is crushed and screened to make terrazzo and other decorative chips. Waste material is sold for agricultural limestone.

The mine is room and pillar type and follows a layer averaging 6 to 7.5 meters (20 to 25 feet) thick down dip. Stone is hauled by truck to a crushing and screening plant at the mine portal. Finished products are then truck-hauled to a bagging and shipping plant on the railroad south of Jasper.

Whitestone Belt

The geology of the Whitestone area was described by Power and Reade (1962). The geologic map (Fig. 4) is from that report.

The Murphy Marble crops out in the valley of Talona Creek and along the east wall of that valley. It dips steeply eastward and is complexly faulted. It is overlain by quartz-biotite schist that is tentatively correlated with the

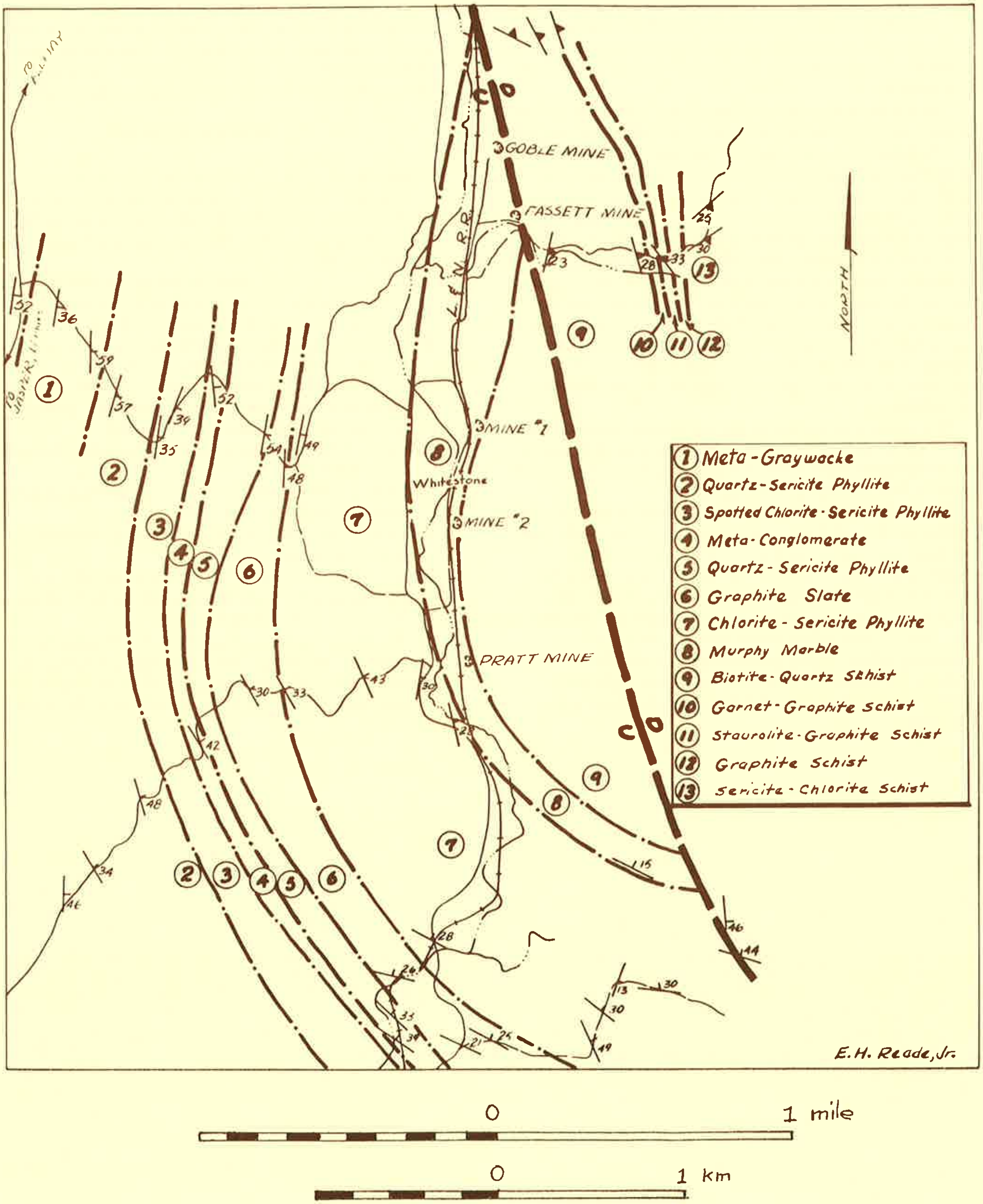

FIGURE 4 - Geologic Map of the Whitestone Belt. (Power and Reade, 1962)

Brasstown Formation and overlies chlorite-sericite phyllite that possibly correlates with the Mineral Bluff Formation. If these tentative correlations are correct the section is overturned to the west and is probably the overturned limb of a syncline.

The marble is divisible into two units: a medium to fine-grained dolomite marble and a coarse-grained calcite marble.

The dolomite marble overlies (but may be stratigraphically lower than) the calcite marble and is in contact with the quartz-biotite schist. Many thin interlayers of talc-phlogopite schist occur near the contact. The main body of the unit is a more-or-less massive, white to gray, fine-grained dolomite marble that is mined for decorative chips and agricultural limestone. Near the contact with the coarse-grained calcite marble, the dolomite is increasingly siliceous and contains numerous pods and laminae of white talc and scattered pale tan to colorless tremolite crystals. The dolomite marble appears to be 150 to 200 meters (500 to 600 feet) thick, but some of this thickness may result from repetition due to faulting.

The coarse-grained, calcite marble closely resembles dark Cherokee or Messotint Marble of the Tate area. It is several hundred meters (feet) thick and underlies most of the flood plain of Talona Creek at Whitestone.

The marble crops out along the east wall of the valley for a distance of more than 3 km (two miles). Most of the area has been open to mining due to the opening of eight connecting mines. The mines are random room and pillar type. Pillars are generally twelve to twenty meters (38 to 64 feet) in diameter with rooms about the same size. Practice is to take up floors after initial openings are made thus increasing ceiling height. Some ceilings in old, abandoned sections are more than 50 meters (150 feet) high, but current practice is to limit ceiling heights to 20 meters (64 feet).

The Georgia Marble Company is currently the sole producer in the area. The company operates three plants and four mines for the production of decorative chips (terrazzo, landscape chips, etc.), agricultural limestone, and dry-ground fillers for use in latex and similar products.

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GEOLOGY OF KYANITE

Dennis Radcliffe
Department of Geology
Hofstra University
Hempstead, New York

Introduction

Kyanite has the chemical formula Al_2SiO_5 with an equal ratio of alumina to silica ($Al_2O_3 \bullet SiO_2$). The theoretical chemical analysis is 63 percent Al_2O_3, 37 percent SiO_2. Kyanite, andalusite and sillimanite are all polymorphs of Al_2SiO_5 and occur under varying pressure and temperature conditions in regionally metamorphosed rocks (Figure 1).

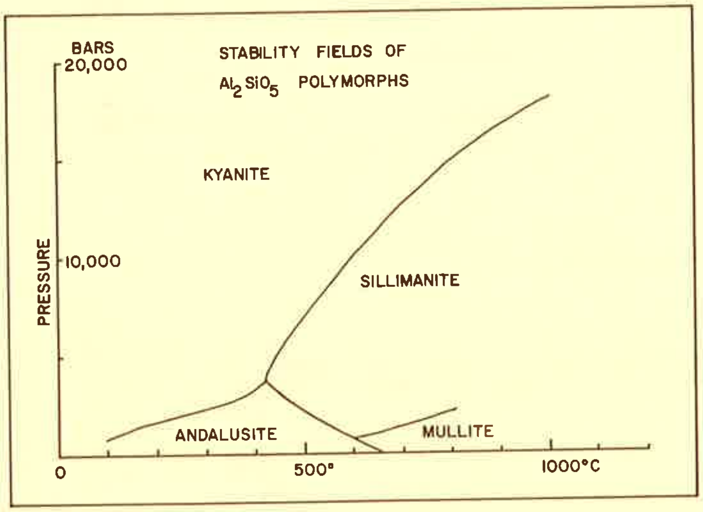

FIGURE 1 - **Pressure and temperature condition for kyanite formation.**

Kyanite occurs frequently in metamorphic rocks such as gneiss, schists and quartzites. In these rocks, kyanite normally does not exceed 5-10 percent of the total mineral abundance, but under certain conditions it may reach 40 percent. Rocks containing 30 percent kyanite are not uncommon from an exploration viewpoint. However, kyanite is produced commercially from only 2 small districts - Graves Mountain, Georgia (C-E Minerals), and Baker and Willis Mountain, Virginia (Kyanite Mining) (Figure 2). This is due not just to the relatively small market (estimated at 100,000 TPY), but to the specifications demanded by the consumer end use (Table 1). Kyanite also exhibits differing properties depending on the local geology of the deposit, and only certain classes of deposits meet industrial specifications. It is, therefore, necessary to thoroughly understand the end use in order to appreciate the commerciality of the deposits.

FIGURE 2 - **Location of the principal kyanite quartzite deposits.**

Industrial Use of Kyanite

Kyanite has a range of uses, but the largest tonnage is consumed by the refractory industry, particularly in specialty refractories including: gunning mixes, mortars, castables and ramming mixes. In most of these applications (Table 2) the expansion characteristics of kyanite (Table 3) are extremely important during firing, as it can offset the shringkage of other phases and produce a stress-free volume-stable refractory. In a related application, the utilization of kyanite in mortars causes mortar expansion during firing. This then locks the refractory bricks tightly in place within the furnace or kiln.

The expansion characteristic of kyanite is related to the generation of mullite plus cristobalite (Figure 3) beginning at 2678° F. The resulting mullite and cristobalite not only has a greater volume than the original kyanite (Table 3) but is extremely refractory. The refractoriness is measured as the softening and deforming point of a cone shaped aggregate. This is known as the pyrometric cone equivalent (PCE). For pure kyanite a PCE of 40 can be attained but the impurities in the commercial concentrate reduces the PCE to the 36-38 range.

Clearly the least amount of impurities is most desirable in any commercial concentrate. In the case of kyanite, any impurities must not tend to lower the softening and melting temperature of kyanite. Thus

TABLE 1 - **Typical composition of ore and specifications of kyanite concentrates.**

ORE*		CONCENTRATE**				
		Analysis			Characteristics	
Al_2O_3	34 5		Al_2O_3	56.6	ca.	91% Kyanite
SiO_2	52.3		SiO_2	40.0	ca.	5% Quartz
Fe_2O_3	2.5	(Total)	Fe_2O_3	1.0 (acid soluble)	ca.	2% Rutile and Fe minerals
TiO_2	1.7		TiO_2	1.0	ca.	1% Hydrous silicates
CaO	0.9		CaO	0.5		
MgO	1.4		MgO	0.5	PCE:	36-37
Na_2O+K_2O	4.7		Na_2O+K_2O	0.1	S.G.: raw calcined	3.6 3.0
L.O.I.	1.8		L.O.I.	0.3	Hardness:	5-7
					Shape:	Bladed

* Kyanite Quartzite contaminated with schist

** From C-E Minerals Sales brochure — Mineralogy calculated from analysis

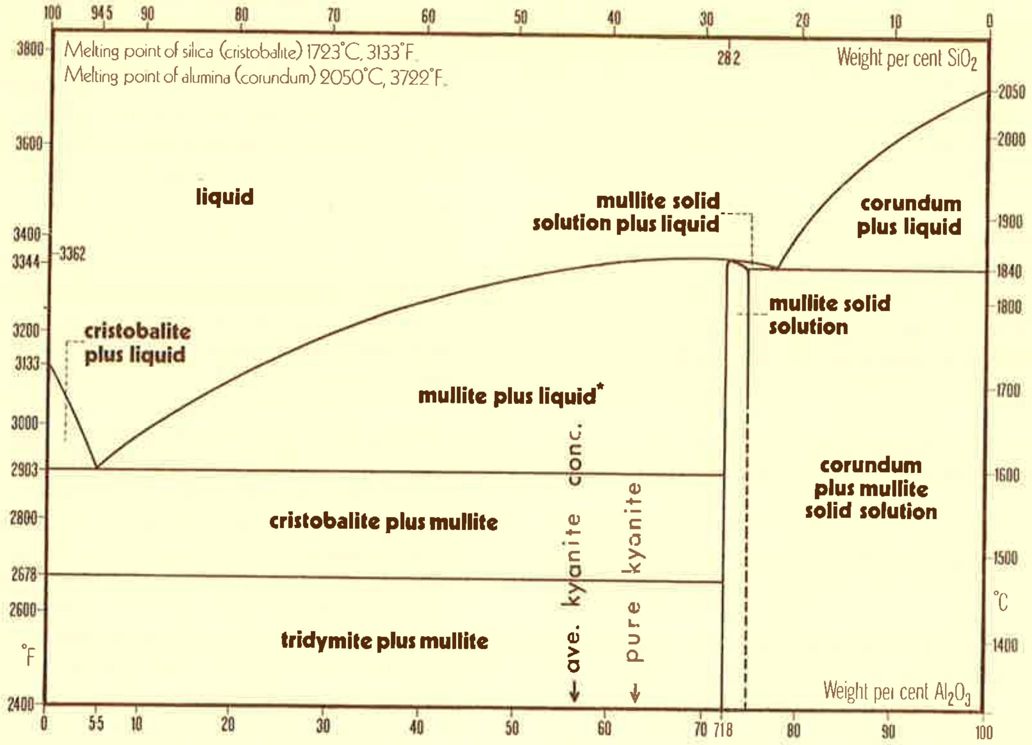

FIGURE 3 - Phase diagram for the Al_2O_3-SiO_2 system.

GENERAL EXTRACTIVE METALLURGY OF KYANITE

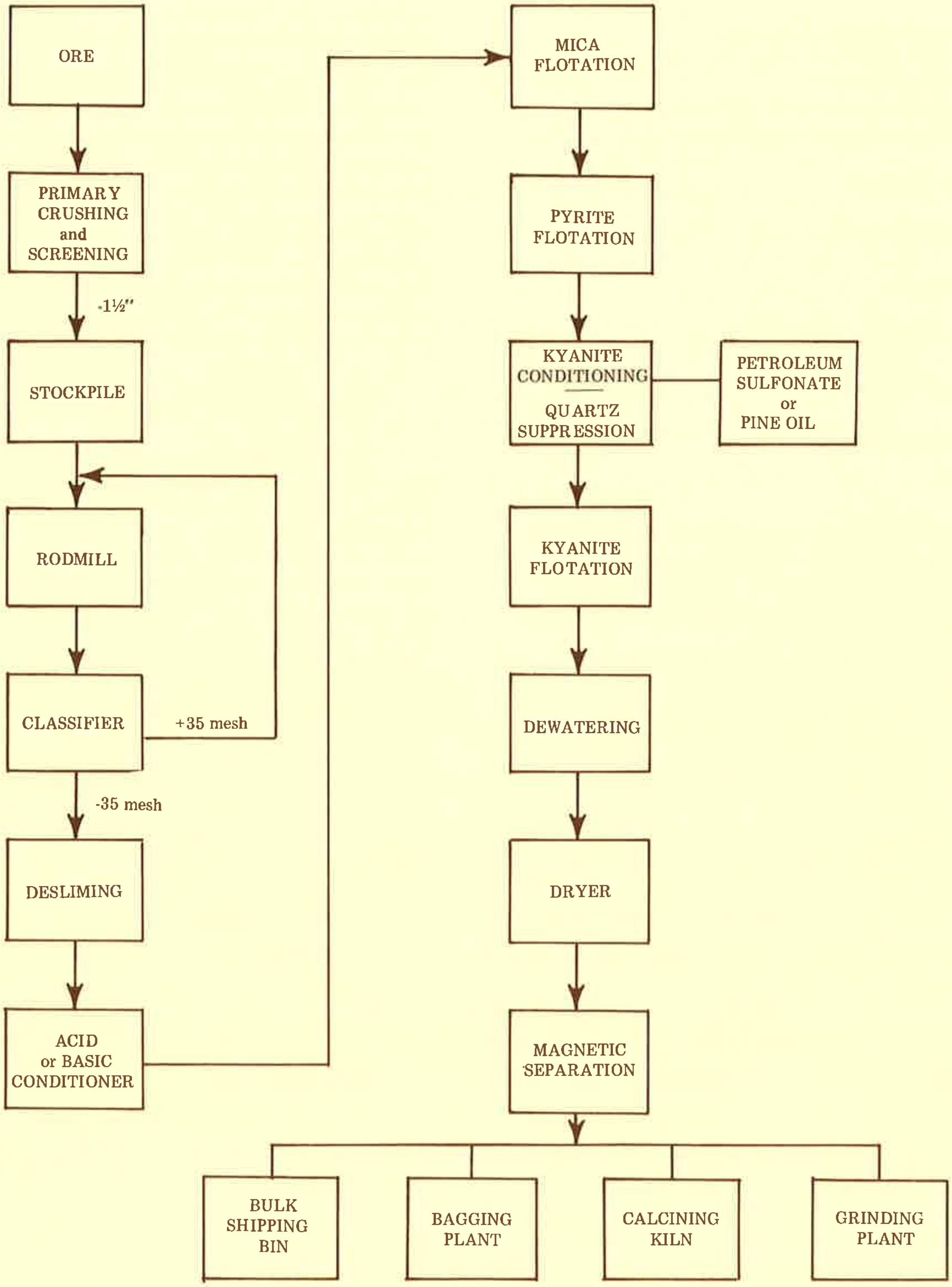

FIGURE 4 - General schematic for kyanite mining and beneficiation.

TABLE 2 - **Summary of industrial applications of kyanite.**

Uses of Kyanite

1. REFRACTORY SPECIALTIES
   Monoliths, Castables, Plastic Ramming Mixes, Cements, Mortars
   Kyanite constitutes 10 to 40% of mixture and offsets shrinkage of clay binder
2. Ceramic Bodies
3. Foundry Molds
4. Brake Shoes
5. Welding Rods
6. Catalytic Converters
7. Aluminum-silicon Alloys. Future Use?

alkalis, iron and other fluxes should be either absent or of very low concentration. Alkalis are particularly harmful in refractories, and their occurrence more than any other factor restricts the types of deposits which can be utilized by industry.

The shape of kyanite crystal is also important, and the sharp rectangular outline of the kyanite cleavage fragments lends itself to bonding with aluminous clays and other ingredients of refractory grogs. The effect is similar to the strength imparted by lath-shaped plagioclase feldspars to ophitic-texture basalts.

The largest size of domestic kyanite sold in the U. S. is 35 mesh. Many refractory manufacturers would prefer a coarser size, but the availability of size is controlled by metallurgical flotation technology. It is just not possible to float kyanite grains which are larger than 35 mesh. Since domestic kyanite must always be beneficiated by standard flotation metallurgy, a 35 mesh upper limit is the effective cut off.

Kyanite also has significant uses because of its chemical, as well as size or shape, characteristics. These applications include mostly refractory ceramics, such as: spark plugs, the substrate of automobile catalytic converters, porcelain enamels, and a host of other uses, but generally in small tonnages. In this application kyanite is normally sold calcined and ground to 100, 200 ro 325 mesh. Calcination is a solid state subsolidus recrystallization into different phases. For kyanite this occurs in the range 2678-2903^{o} F (Figure 3) when mullite and cristobalite are formed from kyanite in a process similar to metamorphism.

Specifications of Kyanite Concentrates

The commercial specifications of the Graves Mountain Kyanite are shown in Table 1. Of great importance is the low level of fluxing elements (alkalis, etc.) which otherwise degrade the refractory bodies. The effect of 1 percent TiO_2 is debatable as it can be a slight advantage or a slight disadvantage depending on the application. Iron oxide has a negative effect in the refractory, in that it causes "rafting" and "spotting" within the body, and can lead to carbon monoxide poisoning. The lowest possible iron contents are most desirable, but 1 percent Fe_2O_3 is not serious. Lower levels of iron are achievalbe but not economically viable. Thus, it is quite possible to generate a 0.5 percent FE_2O_3 concentrate, but the recovery loss is so large that price increases would be necessary and consumers are not willing to pay the price for the higher quality.

Maximum Volume Expansion of Al_2SiO_5 Minerals on Inversion

To Mullite and Cristobalite at 1300^{o} C (2372^{o} F)

$$\begin{array}{llll} 3A1_2SiO_5 \longrightarrow & 3A1_2O_3 \bullet 2SiO_2 & + & SiO_2 \\ (63.0\%\ A1_2O_3 & (71.8\%\ Al_2O_3) & & \\ 100\%\ \text{Kyanite} \longrightarrow & 88.4\%\ \text{Mullite} & + & 11.6\%\ \text{Cristobalite} \end{array}$$

	Cell Volume (A^{o3})	ΔVolume►Mullite	ΔVolume►Cristobalite	$\sum$ % Δ → Volume
Andalusite	342.26	– 2.36	+ 21.17	0.1
Sillimanite	331.42	+ 8.48	+ 32.01	3.4
Kyanite	292.90	+47.00	+ 70.53	17.1

TABLE 3 - **Illustration of the expansional characteristics of kyanite.**

(Realized volume expansions are less due to voids)

The alumina content is quite revealing as pure kyanite should contain 63 percent Al_2O_3, yet most commercial concentrates assay in the range of 56 to 57. This relates to the difficulty of floating kyanite (silicate) from quartz (silica). The normal method is to add a silica depressant to prevent quartz from floating, but the depressant also acts on kyanite. Therefore, only a small amount of suppressant can be used resulting in the floating of some quartz together with kyanite. As a result all kyanite concentrates are contaminated with quartz, generally in the range of 5 to 10 percent.

Supply and Demand of the United States Market

The actual size of the U. S. kyanite market is very difficult to determine accurately because sales figures are held as industrial secrets by the two producing companies. It is possible to estimate production based on the consuming industries, and Radcliffe (1976) estimated 100,000 TPY plus or minus 10 percent depending on the state of the national economy. However, this production figure includes export sales which is another "secret", variously estimated at 10 to 20 percent of total sales.

The future demands on the kyanite producers is also difficult to determine. While the U. S. steel industry is projected by economists to grow at an annual rate of 8 percent per year for the next 5 years, improvements in refractory technology tend to reduce the unit comsumption of refractory raw materials. The main hope for expanded demand is in the development of new applications. The most promising appears to be in the manufacture of Al-Si alloys by heating kyanite with coke and then converting the carbide to the alloying elements in an electric arc furnace.

The state of the supply precludes the entry of new companies into the field of production. The two current suppliers together have an approximate 100 percent excess production capacity. This overwhelming excess originated in 1973 when kyanite production fell well behind demand due to rapidly expanding sales in the pre-oil embargo economy. Both producers expected a continuation of strong and expanding sales and moved into a position to meet grossly exaggerated future sales forecasts. In addition to this excessive production capacity, both producers have very substantial ore reserves (Bennett, 1975, Espenshade, 1973, and Potter, 1975) suitable for at least 50 years of continuous operation.

Mining and Benefication of Kyanite Deposits (Figure 4)

Mining is by open pit, utilizing standard techniques. Ten to twenty foot benches are drilled with percussion equipment. The 6-inch holes are packed with an explosive mixture of fertilizer (ammonium nitrate) and fuel oil, and detonated with dynamite. The shot is loaded with shovels, transported by truck to a primary crusher, normally backed up with a secondary Symans Cone crusher. The minus 1 to 1½ inch rock is then stockpiled for feeding into the flotation plant.

The ore is fed by conveyer into a rod mill in closed circuit with a classification system which is generally a rake. The classification is aimed at 28 mesh but due to variation in ore characteristics may become 35 mesh. After desliming, the ore is conditioned (pine oil and xanthate at Graves Mountain – amine collector) for removal of pyrite, sericite and any other clay minerals. Kyanite can be flotated either in an acid or basic circuit. At Graves Mountain an acid circuit is employed, and after removal of pyrite and sericite, the slurry is reconditioned with sulfuric acid and petroleum sulfonate followed by normal counter current flotation circuitry.

The resulting kyanite concentrate contains clean grains and kyanite crystals coated with varying amounts of iron oxide contaminants. Wet or dry magnetics reduce the iron content to a level of approximately 1 percent Fe_2O_3 resulting in a recovery loss of kyanite. To achieve lower levels of iron, it is necessary to process very pure ore (an extravagence) or to magnetize the remaining iron by heat treatment. This latter is costly and in addition, the small gains in iron are offset by excessive losses in kyanite grains which, at this level, contain only very small patches of iron.

The iron coating on the kyanite crystals throughout the deposit is sufficiently tenacious such that a ferrugenous pseudomorph after kyanite is suggested. The nature of the iron contaminant is normally thought to be low temperature goethite (hydrous iron oxide), as it does not normally give an X-ray pattern. However, recent Mossbauer studies have shown it to be an aluminum-substituted hematite, perhaps supporting (?) a hydrothermal (rather than weathering) origin for most of the strong iron alteration products so prevalent in many parts of the deposits.

Petrography and Metallurgy

Kyanite crystals vary in texture from one deposit to another, and an elementary analysis of petrographic characteristics (Figure 5) quickly leads to a prediction of metallurgical behavior and quality of concentrates. The following types of information can normally be obtained from a standard 0.03mm petrographic thin section:

(a) The grain size of the kyanite crystal and probable liberation point of the intercrystalline rock aggregate.

(b) The frequency of inclusions within the kyanite grain. For example, the kyanite grains in Figure 5A are essentially free of inclusions, whereas those in 5C are loaded with impurity inclusions. In this latter case a theoretic 100 percent recovery and a

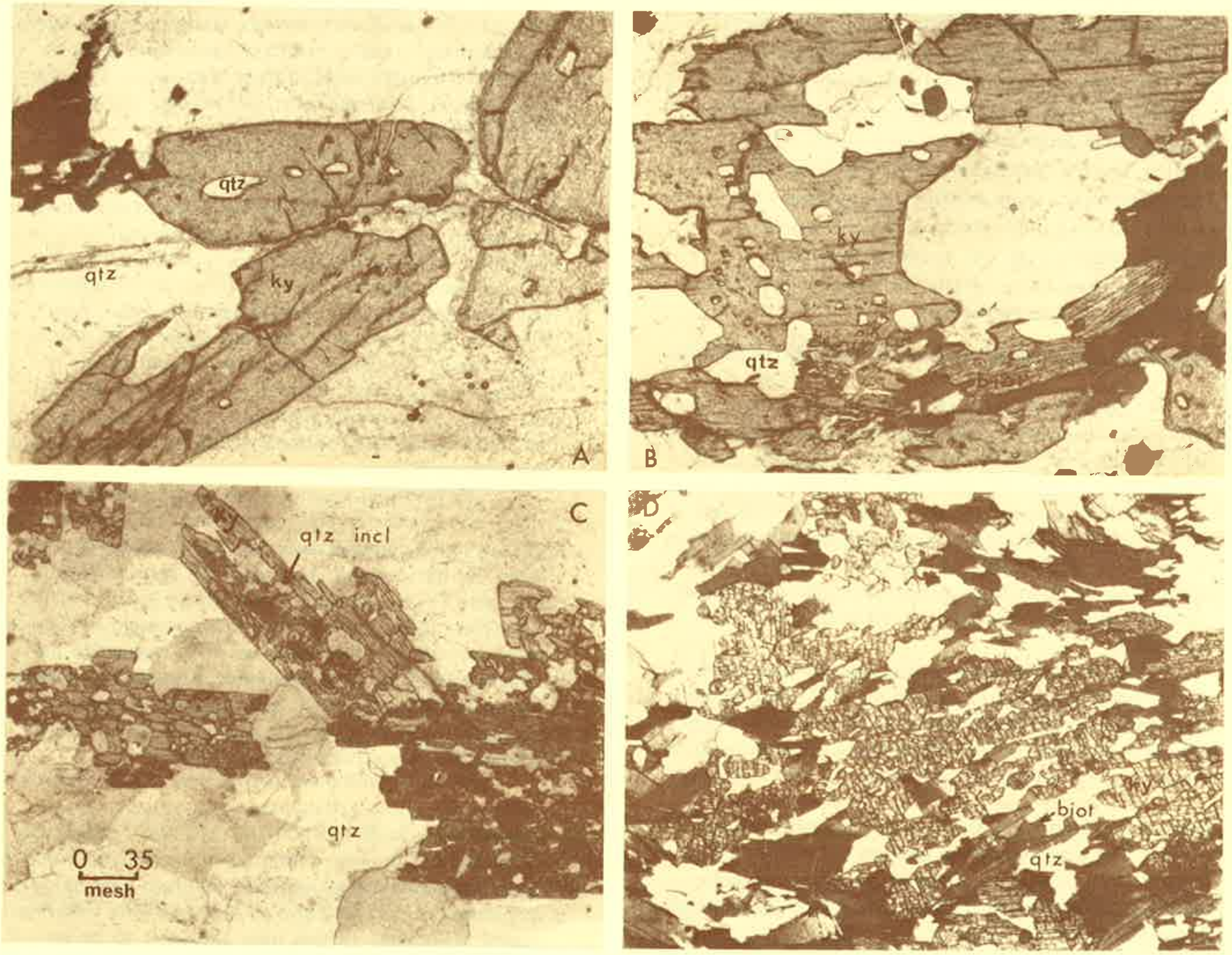

FIGURE 5 - **Photomicrographs of kyanite ore. Rock sections are 0.03mm thick and show the distribution of kyanite (ky), quartz (qtz), quartz inclusions (qtz incl in 5C) and biotite (biot).**

theoretic 100 percent kyanite grain concentrate would still only yield a substantially low 40-50 percent Al_2O_3 compared to 2A which could be essentially 63 percent.

(c) The nature of the distribution of undesirable phases, such as iron-bearing micas (biotite). In Figure 5D the micas are attached to the kyanite grains, and a minimum 48 mesh grind is necessary to liberate the two phases. In Figure 5B a relatively clean 35 mesh kyanite concentrate might be predicted, however, the biotite (lower right) is included within the kyanite grain and may well contaminate the final product.

Geology of Kyanite

Kyanite occurs principally in two petrologically distinct facies, both of which are metamorphic in origin. It occurs either in a gneiss (or schist) associated with feldspar, biotite and garnet, or in a quartzite where it is associated with pyrite, rutile and pyrophyllite.

Every metamorphic petrologist is familiar with kyanite grade metamorphism of pelitic assemblages, but such rocks are generally low in kyanite (less than 10 percent) because the available alumina is complexed in other phases. The principle deposits of kyanite in North America were described by Radcliffe (1975) as follows:

(a) Kyanite-quartzite belt of the southeastern states (C-E Minerals and Kyanite Mining).

(b) Kyanite-garnet-biotite gneisses of the Wanapitei-Crocan Lake-Timiskaming belt in Ontario-Quebec (Kyanite Mining).

(c) Kyanite-biotite-garnet gneisses of the Clearwater River area (middle fork) in north-central Idaho (Ethyl Corporation).

(d) Kyanite-quartzite near Ogilby, California.

TABLE 4 - Origin of Kyanite Quartzite

Hypothesis of Metamorphism of Transported Sandy Kaolin Deposit

	Typical Kyanite Ore	Typical Sandy Kaolin
Al_2O_3	34.5	34.2
SiO_2	52.3	47.5
Fe_2O_3	2.5	1.0
TiO_2	1.7	1.5
CaO	0.9	0.2
MgO	1.4	0.4
Na_2O+K_2O	4.7	1.1
L.O.I.	1.8	14.0
Mineralogy	KYANITE Al_2SiO_5 QUARTZ lazulite pyrite ilmenite goethite rutile muscovite pyrophyllite	KAOLINITE $Al_4Si_4O_{10}(OH)_8$ QUARTZ apatite pyrite ilmenite goethite rutile muscovite kyanite and sillimanite

The Geology of Graves Mountain

Graves Mountain (Figure 6) was first described by Shepard (1859), Watson and Watson (1912), then by Prindle (1935), followed by Furcron and Teague (1945), more recently by Crickmay (1952). However, Hurst (1959) provided the first comprehensive mapping and detailed analysis of the deposit. Espenshade and Potter (1960) provided the first regional analysis of the southeastern deposits, and Hartley (1976) gave the up-to-date account of the mining and processing operation. It is somewhat ironic that while Graves Mountain is one of the very best kyanite deposits, it is known primarily for its very fine, black lustreous multiple-twinned rutile crystals. These latter are illustrated in most modern mineralogy text books and also can be found in all good mineralogy museums anywhere in the world.

Graves Mountain is an elongate monadnock (Figure 6) reaching 900 feet above sea level and projecting 400 feet above the general level of the surrounding Piedmont. It is about 7200 feet long and 200 feet wide trending in a northeasterly direction, directly parallel with regional strike. The main ore zone however is about 800 feet wide. Graves Mountain occurs within gneisses and schists of the Little River Series which have been correlated with rocks of The Carolina Slate Belt and dated as Lower Paleozoic. These sequences are regarded as metabolcanics or at least interbedded metavolcanics and metasedimentary facies. Because of this association the Graves Mountain kyanite quartzite is frequently assigned as being volcanic in origin (Hurst, 1959, Espenshade, 1960, and Hartley, 1976), but this is not necessarily correct.

Graves Mountain itself has an average N70°E strike and consists of interbedded and somewhat lensoidal kyanite quartzite and sericite schist (with or without kyanite). The individual quartzite units vary in thickness from 6 to 50 feet. The zones dip northwesterly at varying angles, generally 60-70°. The quartzites are massive and the schistose rocks display cleavage parallel only with the apparent bedding direction of the quartzite bodies. Therefore, it is difficult to determine whether or not Graves Mountain is an inclined series of interbedded rocks, an overturned syncline or an over-turned anticline.

Examination of Graves Mountain rocks can

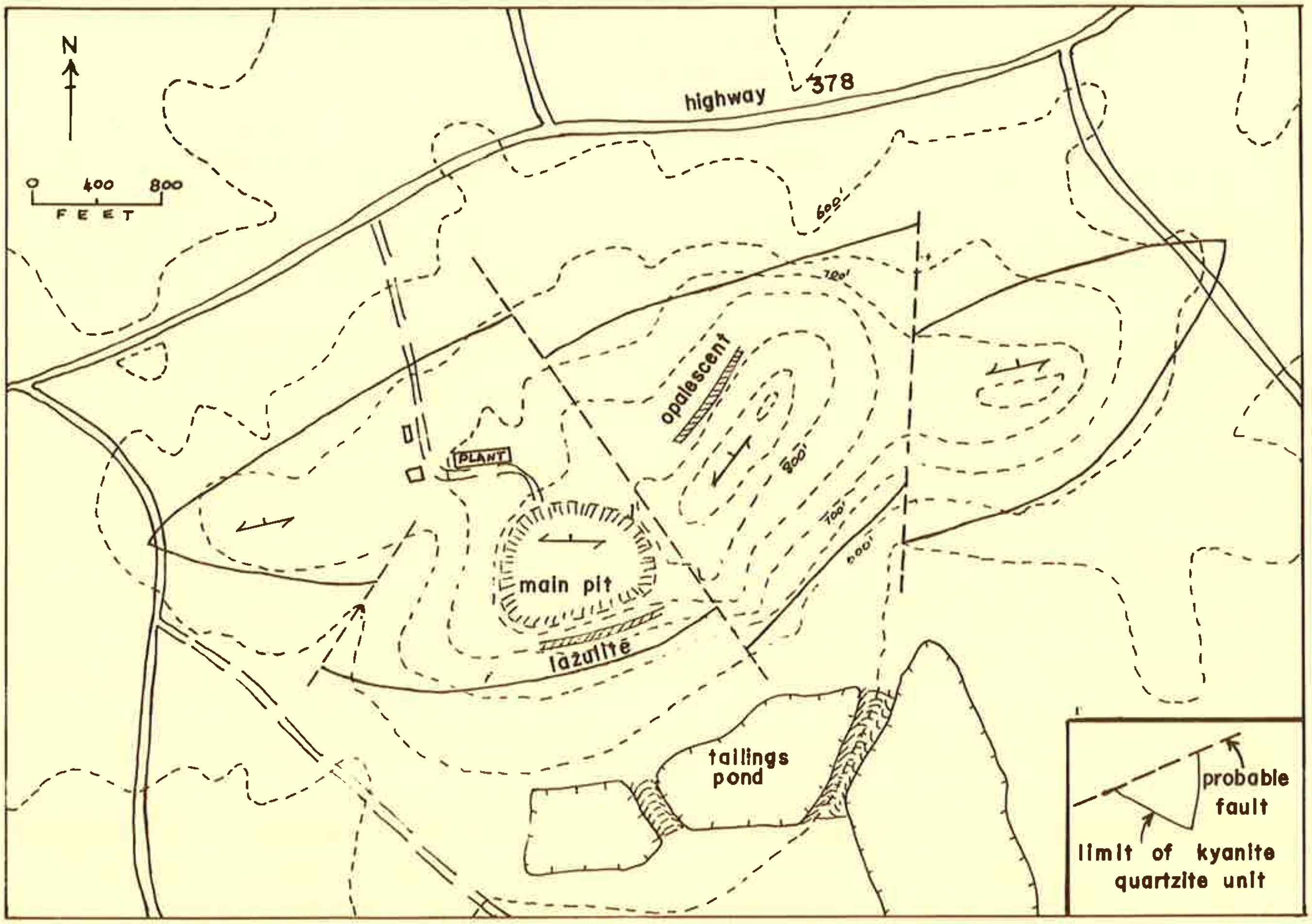

FIGURE 6 - Generalized geologic map of Graves Mountain. Opalescent refers to quartz porphyroclasts in a kyanite-poor pyrite quartzite band and lazulite refers to a lazulite-bearing kyanite pyrite rutile quartzite. These two units bound the main ore zone.

Many other deposits exist but most of these are eclipsed by the enormity of these first three deposits. Because of the restrictions imposed by the application of kyanite concentrates in refractories, the kyanite quartzites of the southeastern states are by far the most suitable deposits.

Kyanite quartzite occurs in a discontinuous belt from south central Virginia to northeast Georgia (Figure 2), a distance of almost 700 miles. No less than 14 deposits have been described by Espenshade and Potter (1960) in this belt. Ironically, the best deposits occur at the extremities of the belt. Kyanite Mining Corporation of Dillwyn, Virginia, operates five plants at their two mines located at Willis Mountain and Baker Mountain. Kyanite Mining are currently building a sixth plant at East Ridge, immediately east of Willis Mountain.

At the southern extremity of the belt, C-E Minerals, a subsidiary of Combustion Engineering, Inc., operates a shipping plant at Little River, and a processing plant at its Graves Mountain mine. The geology of the Graves Mountain deposit is fairly typical of the belt and a short description of this area illustrates the geology of this class of deposits.

Genesis of Graves Mountain

The origin of kyanite quartzite of the Graves Mountain type is generally thought to be metamorphism of volcanic rocks. A different origin is now suggested.

Hurst (1959) described the deposition of tuffaceous sediments and intercalated gravels along a zone passing through Graves Mountain. Espenshade and Potter (1960) describe the replacement of an igneous rock by aluminum and silicon, and Hartley (1976) described the metamorphism of vitric tuffs.

The evidence for a pre-metamorphic igneous origin is based on the occurrence of the rounded opalescent quartz grains in the kyanite-poor pyrite quartzite which occurs at the northwest margin of the main ore zone (Figure 6). The opalescent quartz is supposedly bipyramidal and hence igneous in origin. The grains are rounded and this is explained by magmatic resorption. It requires a fair amount of imagination to observe bipyramidal quartz. The present writer has never observed a quartz grain which is convincingly bipyramidal.

If this opalescent-quartz bearing quartzite were located elsewhere, such as the Blue Ridge of Virginia, it would be called a metaconglomerate which is much more plausible. It is even possible for a clay-bearing sand to contain large fragments of bipyramidal quartz, weathered from an igneous rock. Metamorphism of this sedimentary facies might produce the rock unit observed at the margin of the main Graves Mountain ore zone.

If we strip away the complications imposed on the rock series by secondary hydrothermal activity and weathering, we find that the mineralogy of the Graves Mountain-type kyanite quartzites is essentially the same on a world-wide basis: kyanite, quartz, pyrite, rutile and lazulite. Occasionally substitute minerals, based on local variations of pressure and temperature, are also found, such as: andalusite for kyanite, goyazite for lazulte, and occasionally ilmenite. This mineral assemblage tends to be intercalcalated with a schistose assemblage characterized by a potassium-bearing mica (sericite), quartz, pyrite and generally some kyanite in small quantities. Whether or not kyanite occurs is based on the relative chemical mass balance of alkalis and alumina in this facies.

The rather distinct mineralogy must be significant, especially that of the quartzite. It is proposed that the kyanite and quartz crystallized from a sandy kaolinite sedimentary facies under metamorphic conditions. The occurrence of pyrite reflect the depositional environment, and the occurrence of rutile and lazulite represents a heavy mineral suite or possibly a phosphatic horizon

The late-Cretaceous, early-Tertiary Kaolin deposits of Georgia, Alabama and the Carolinas form a subcontinuous arcuate belt and are known to be deposited in coastal or lagoonal environments. The kaolinite was weathered from the exposed crystalline Paleozoic piedmont rocks and transported towards the ocean where it was deposited in a relatively low energy environment. Such environments are generally sandy, may be characterized by dark-colored pyrite-bearing clays in the estuarine marshes and lagoons and normally contain a small heavy mineral suite of ilmenite ($FeTiO_3$), rutile or anatase (TiO_2), and apatite, ($Ca_5(PO4)_3(OH)$, togehter with kyanite and sillimanite. Occasionally phosphatic sand horizons (associated with the near shore environment) also occur.

From an industrial viewpoint the occurrence of anatase and altered pyrite in the Georgia kaolin deposits and their passage laterally or vertically vertically into either a sandy facies or a pyritic sericite clay facies is well known.

Metamorphism of this normal sedimentary facies will produce the exact mineralogy observed in the Graves Mountain-type of kyanite quartzites (Table 4). Further, it will also produce the interbedding with sericite pyrite schist.

If this hypothesis is correct then the occurrence of kyanite quartzites in the near linear belt extending from Graves Mountain, Georgia, through the Carolinas into Virginia may well represent a paleo-shoreline of lower Paleozoic age. It would also follow that the associated volcanic rocks which are prevalent in Georgia and parts of the Carolinas were deposited near-shore and possibly under shallow water conditions. This is somewhat similar to the China-Korea mainland and its association with the continental shelf of the Sea of Japan and the Japanese volcanic archepelago at present.

lead to descriptions of at least 8 distinct, important rock types. Detailed analysis shows however that they are only varients of the two basic rock types. The apparent difference results from later weathering and/or hydrothermal alteration.

The two original pre-alteration rocks of Graves Mountain were characterized by the omnipresence of pyrite (FeS_2) and generally small quantities of rutile (TiO_2). Weathering and/or hydrothermal alteration differentially leached pyrite which then generated staining iron-rich solutions. The staining solutions preferentially coat kyanite and cause serious metallurgical problems. The change in rock type is illustrated below:

blue kyanite pyrite quartzite	red kyanite quartzite + iron oxides
white sericite pyrite schist	red sericite schist + iron oxides

This four-fold rock division is further complicated by a zone of kyanite-pyrite-rutile-quartzite containing small amounts of lazulite, $(MgFe)Al_2(PO_4)_2(OH)_2$. This latter occurs along the southeast margin of the bodies (Figure 6). On the northwest margin of the main ore body a zone of kyanite-poor pyrite quartzite contains bluish colored, rounded, opalescent quartz porphyroclasts (?) up to 10mm in diameter.

This series of rocks is further complicated by an intersecting network of hydrothermal massive quartz veins, resulting in localized pyrophyllite (Hurst, 1959) and locally recrystallized (?) kyanite. It can be observed that within at least 3 feet of the massive quartz veins, the kyanite in the kyanite-quartzite increases markedly in size. If this recrystallization process were operative it can also explain the selective occurrence of the large museum size (100mm diameter) back rutile crystals at the margin of the narrow (12-24") massive quartzite veins which are themselves thoroughly steeped in hydrous iron oxides. Rutile normally occurs as small (0.5mm diameter) red crystals. It is possible that the hydrothermal solutions dissolved pyrite (FeS_2) from the quartzites. This could generate an iron-staining solution of sulfuric acid which would dissolve the smaller rutile crystals. The titanium later precipitated from solution as large, black iron-bearing rutile crystals at the margins of the quartz vein. This would explain the juxtaposition of intense iron staining and large rutile crystals, a fact borne out by direct field observations.

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