CANADIAN SILICA RESOURCES:
A STUDY OF THE PROCESSING OF ONTARIO SILICA FOR GLASS AND FOUNDRY SAND

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SUMMARY

In cooperation with the Ontario Ministry of Natural Resources, selected deposits of Potsdam sandstone in the Ottawa-Kinna rea were core drilled and the core evaluated by CANMET as potential glass and foundry sand. Core analyses ranged from 84.6 to 98.5% SiO₂, 0.09 to 0.65% Fe₂O₃, and 0.07 to 2.10% Al₂O₃. The core were largely composed of white to grey, relatively tough sandstone. Impurities included iron as pyrite, and as iron oxides and staining on the quartz grains; carbonates as calcite; and aluminium oxide as feldspar. Beneficiation included the reduction to grain size, about 590 μm, by jaw, roll and cone crushers, and grinding in a muller-mixer. Removal of fines was achieved by attrition scrubbing and washing. Further removal of impurities was achieved by reverse flotation of pyrite, iron oxide, feldspar, and carbonate minerals employing both anionic and cationic collectors. Flotation resulted in a substantial improvement in the quality of most samples, the quartz concentrates analysing 98.5 to 99.7% SiO₂, 0.022 to 0.156% Fe₂O₃, and 0.07 to 0.62% Al₂O₃. Further reduction of the iron content by wet magnetic separation and acid leaching also was investigated. On the basis of the samples submitted, sandstones from six of the seven deposits studied have potential as sources of glass sand. Foundry sand tests were inconclusive because of the small quantity of sample available for study.
INTRODUCTION

Canada continues to import significant quantities of high-purity sand, chiefly from the United States, for use in glass manufacture and by the foundry industry. Imports in 1981, for example, were 1.14 Mt valued at $18.5 million. There are a number of potential sources of high-purity silica sand in Canada. This fact and the need for further study was highlighted at the Federal-Provincial Industrial Minerals Seminar that was held in 1980 in Ottawa. Discussions within CANMET following that meeting resulted in a decision to initiate a laboratory study and appraisal of the technical feasibility of producing silica sand from a few of the more attractive deposits for domestic consumption and export. This 3-year study would be made in cooperation with the provincial governments and was to concentrate initially on Ontario, the chief consuming province and, later, on selected resources in several of the remaining provinces. The paper outlines work undertaken on Ontario sandstones during 1982/83. The Ontario Ministry of Natural Resources co-operated fully in this work; Dr. Mike A. Klugman of OMNR was of invaluable assistance in this regard.

SOURCE AND DESCRIPTION OF SAMPLES

An initial review was made of two reports of Ontario sandstone, open file reports 5265 and 5305 of the Ontario Ministry of Natural Resources. On the basis of this review, silica deposits in five areas were selected for detailed study. These and those highlighted in the noted reports were generally included in the area bounded by Ottawa, Arnprior, Kingston and Brockville as shown in Figure 1.

The selection of these particular deposits was based on the assumption that the study should include a range of purity of sandstones to determine the effectiveness of developed methods of processing and beneficiation to glass sand purity, e.g., 99.6% SiO₂, 0.03% Fe₂O₃, 0.15% Al₂O₃ with low lime and magnesia. Drill core samples from the five noted areas were provided by the Ontario Ministry of Natural Resources. Deposits in two other areas, both under investigation by potential producers, were later included. Sample locations, identified as A to G, are shown in Figure 1.
PRELIMINARY STUDY OF SANDSTONE REDUCTION

In processing sandstone to sand for glass and foundry applications it is important that the parent sandstone is reduced to its natural grain size to free impurities for subsequent removal and, as well, that this be done without excessive fracturing of grains and resulting production of unwanted fines. Whole, unfractured grains are preferred in foundry moulding whereas fines are detrimental in both applications.

Preliminary reduction is easily achieved by jaw, cone, and roll crushers. Final reduction, to grain size, may be accomplished using one of several types of crushers/grinders, e.g., ring-roll, roller, ball or rod mills. Although normally performed dry, reduction of sandstone to grain size also may be accomplished by wet processing. Preliminary crushing/grinding studies were made using Potsdam sandstone samples from the Bells Corners area, west of Ottawa. Initial reduction, to 4.75 mm, was by a Blake jaw crusher with a 75 x 150 mm opening and, to 1.68 mm, by a 250 mm diameter Mine and Smelter cone crusher. Tertiary crushing, to 590 μm, was by a 125 x 200 mm double-roll, Sturtevant crusher. Further reduction to grain size was studied both dry and wet (98% solids) in a 450 mm diameter Simpson muller-mixer. An examination of the crushed products showed a good reduction of the sandstone to grain size with a minimum percentage of agglomerate and fractured grains.

SAMPLE PREPARATION AND PROCESSING

The seven core samples were carefully examined to determine grain fineness, friability, impurities such as pyrite, iron staining, lime, etc. and to select an appropriate interval of core for study. The results are noted in Table 1.

Crushing/grinding

Following examination of the core, the selected samples were separately processed through the jaw, cone and roll crushers, sized by Sweco screen at 125 μm, and the plus 125 μm product wet-ground at 95% solids for a 30-min. period in the Simpson muller-mixer. This scrubbing/grinding action reduced most of the compound grains to grain size without creating excessive fines.
Previous CANMET studies of the processing of sandstone-derived sand demonstrated attrition scrubbing to be an ideal secondary processing step. The intensive scrubbing aids breakdown of clay-bonded compound grains and the removal of coatings and surface staining on the individual quartz particles.

The muller-mixer product was scrubbed in a 10 kg capacity, Wemco attrition machine at 80% solids over a 20-min. period. The scrubbed product was water washed for 5 min. and the solids settled during an additional 10 min. period. The slimes were then decanted at 105 μm and the entire procedure repeated. Three washing-settling-decanting cycles were required to free and remove the slimes. Sieve analyses of the washed attrition products are noted in Table 2.

Mineralogy

Representative portions of each sample were mounted in petrographic thin sections for examination and, as well, pulverized for analyses by X-ray diffraction using a Guinier camera. Results are recorded in Table 3.

Flotation

Removal of small quantities of liberated impurities by flotation is an efficient method of benefication. Because silica is the major constituent it is logical to float the impurities by reverse flotation leaving a pure quartz concentrate.

The procedure adopted for removal of impurities was based on the known response of the minerals to flotation. In this respect pyrite is best floated using xanthates, graphite using kerosene, and calcite using oleic acid. Flotation practice has previously demonstrated that iron oxides and feldspar are best floated by first using petroleum sulphonate collectors to float iron oxides followed by cationic flotation using amines to float feldspar.

Previous cationic flotation studies on feldspar have always been conducted using HF as an activator for feldspar. Since HF is extremely hazardous it was decided to try to float feldspar using standard amine collectors without HF. In addition, the collector Duomeen TDO (n-tallow-1, 3-diaminopropane dioleate) was used, since it is selective for feldspar.
The above the following general procedure was adopted:

**Stage 1.** Graphite flotation using Kerosene at pH 9.

**Stage 2.** Sulphide flotation using Ethyl Xanthate at pH 6.

**Stage 3.** Calcite flotation using an Oleic Acid/Kerosene mixture at pH 7.

**Stage 4.** Iron Oxide flotation using Aerofloat 840 (natural petroleum sulphonate) at pH 3.5-5.0, employing high pulp density conditioning (≈50% solids).

**Stage 5.** Feldspar flotation using
   - Armac T (tallow amine acetate)
   - Armac C (coco amine acetate)
   - Duomeen TDO (n-tallow -1, 3-diaminopropane dioleate),
   employing high pulp density conditioning (≈50% solids)

The outlined procedure was used only as a guide. In some tests, some of the stages were omitted as, for example, stage 3 when using sample E as it contained very little calcite, and stage 1 only was used for sample G, since it was the only sample that contained graphite.

After calcite flotation, the remaining quartz concentrate was filtered and washed twice to remove residual flotation reagent before repulping and conditioning at 50% solids. After conditioning the pulp density was diluted to ≈30% solids and flotation conducted without further conditioning.

Following completion of the final flotation stage all products were filtered and dried. Since only very small quantities of mineral were sometimes floated, it was decided to analyse only the quartz concentrate. The resulting concentrate analysis when compared to the flotation feed analysis provided a good indication of the degree of success obtained in removing the impurities.

The results, Table 4, show the analysis figures for the best test obtained from each sample; the average analysis figures, computed from all flotation tests on the same sample, also are shown. Reagent consumption data for the best tests are shown in Table 5.
With the exception of sample C_T, the Fe\textsubscript{2}O\textsubscript{3} levels were reduced to below 0.1% Fe\textsubscript{2}O\textsubscript{3}. For sample C_T, flotation was only able to reduce the level from 0.28% to 0.11% Fe\textsubscript{2}O\textsubscript{3}. The flotation concentrate of sample C_T was treated in a high-intensity Sala wet magnetic separator to determine whether a further reduction of the iron was feasible. Results are noted in Table 4.

**Acid leaching**

To determine whether the iron oxide content of the quartz concentrates could be further reduced, several selected concentrate samples were acid leached. Results are noted in Table 4.

**Foundry sand testing**

Samples of the scrubbed attrition products were submitted to the sand laboratory for evaluation as foundry sand. The small size of samples permitted only a very limited study. An initial size analysis of each sample was made, the fractions examined, and the AFS numbers determined. Moulding tests were performed using 2 kg of sand and the required amounts of bentonite and cereal. Moisture content, moulding characteristics, permeability, and green compressive and tensile strengths were determined. Results are reported in Table 7.

**RESULTS AND EVALUATION**

**Crushing, grinding, scrubbing**

Examination of products from each stage of crushing and of the mesh fractions from the Wemco attrition scrubber indicate that the various stages of size reduction satisfactorily reduced this sandstone to grain size without excessive fracturing of grains and production of unwanted waste fines. Recovery of \(-590 + 105 \mu m\) sand from the attrition scrubber averaged 86.5% for the samples studied, and ranged from 75% for sample F to 95% for sample E. Most samples had a fairly uniform grain size with the bulk of the material being in the \(-590 + 150 \mu m\) range; sample A was more finely divided than the others. The products generally had relatively
percentages of compound and fractured grains; however, grain angularity or sharpness showed a substantial increase in the finer, -105 µm sizes. Sample F contained a substantial portion of sharply angular grains due to numerous inclusions of pockets or intersections of vein quartz in the drill core.

Most sands were white to light-grey in colour, however, sample E was light brown due to a thin brown coating on the individual quartz grains. Impurities mostly consisted of pyrite, which was both free and firmly attached to quartz grains, iron staining, scattered dark mineral grains, and carbonate. Carbonate content varied quite widely from sample to sample. Sample G was distinctly different to the others in that it contained a significant portion of graphite as small specks adhering to the individual quartz particles.

**Flotation**

In general flotation successfully reduced iron oxide and feldspar to levels acceptable for the glass industry as the results in Table 4 show. Visual examination of flotation products also showed that pyrite and calcite responded well to flotation.

With the exception of sample CT where the Fe₂O₃ level could not be reduced below 0.107% Fe₂O₃, the remaining samples produced quartz concentrates analysing 0.024-0.043% Fe₂O₃, which is within range of the 0.03% Fe₂O₃ specification for glass manufacture. With regard to feldspar reduction, the Al₂O₃ level was reduced for all samples, except samples CT and D, to between 0.067-0.098% Al₂O₃ which is below the 0.15% Al₂O₃ glass sand specification.

The selectivity of the collectors employed in iron oxide and feldspar flotation is best illustrated by referring to Table 6, where the collector(s) employed to achieve the best test results are tabulated. This summary shows that Duomeen TDO was selective for floating feldspar in four samples. Whether Duomeen TDO is selective for feldspar flotation cannot accurately be assessed due to the small quantities of minerals involved, there is, however, a trend which suggests that it is.

**Magnetic separation**

Wet magnetic separation in the Sala high-intensity separator resulted in a significant reduction of the iron in sample CT, from 0.133 to 0.064%
However, this value is still in excess of the 0.03% Fe₂O₃ typically specified for glass sand.

Acid leaching

Acid leach test results, noted in Table 4, show significant reductions of the iron content of the flotation concentrates. The iron content of sample C₁, for example, was reduced from 0.140 to 0.061%, and of D, from 0.055 to 0.028%, by leaching. This latter value is acceptable for glass sand.

Foundry evaluation

Although some of the processed sands may have potential as foundry sand, the small size of samples tested did not permit a full-scale evaluation.

CONCLUSION

On the basis of samples submitted, sandstones from six of the seven areas represented by these samples have potential as sources of glass sand, namely A, B, C₁, E, F, and G. Unfortunately C₂, representing the bottom 12.8 m of a 25 m drill hole is overlain by C₁, representing the top 12.2 m. C₁ did not quite meet glass sand specifications following beneficiation. Further research with larger and more representative samples is recommended to strengthen and confirm the results of this preliminary study. Further research to more fully determine the potential of these sandstones as foundry sand is also recommended.
Fig. 1 - Location of seven sandstone sample sites
### Table 1 - Drill core description

<table>
<thead>
<tr>
<th>Sample</th>
<th>Type</th>
<th>Description</th>
</tr>
</thead>
</table>
| A      | Whole core | 0 to 10.4 m  
fine grain, white to brown, friable to tough, impure sandstone 9.1 to 10.4 m; 1.8 to 9.1 m intersection chosen for laboratory study. |
| B      | Whole core | 0 to 8.8 m  
fine grain, white, friable sandstone; core loss about .6 m between 3.1 to 4.3 m; minor iron staining upper 3.1 m, white sandstone 3.1 to 7.6 m; hard limy sandstone with calcite veins 7.6 to 8.8 m; 0 to 7.6 m intersection chosen for laboratory study. |
| C      | Split core | 0 to 12.2 m  
0 to 0.9 m missing, possibly overburden; fine grain, white to grey, hard sandstone with minor iron staining and thin shaley seams or partings throughout; minor carbonate 9.1 to 12.2 m, 0.9 to 12.2 m intersection chosen for laboratory study. |
| D      | Whole core | 12.2 to 25 m  
fine grain, white to grey sandstone; minor iron staining, calcite, shale partings and quartz inclusions; 12.2 to 25.0 m intersection chosen for laboratory study. |
| E      | Whole core | 0 to 6.9 m  
fine grain, white to grey and brown, tough sandstone; some iron staining, shale partings and calcareous sections; 0 to 6.9 m intersection chosen for laboratory study. |
|        |            | 0 to 9.1 m  
fine grain, pink, relatively friable sandstone; minor inclusions of quartz with some iron staining 2.4 to 7.6 m; impure sandstone 7.6 to 8.5 m; Precambrian 8.5 to 9.1 m; 0 to 7.6 m intersection chosen for laboratory study. |
Table 1 - Drill core description (Cont'd)

<table>
<thead>
<tr>
<th>Sample</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>F</td>
<td>Whole core  - 0 to 8.8 m fine grain, impure, grey to mottled brown sandstone; substantial percentage of quartz inclusions and veins; some lost core; 0 to 8.8 m intersection chosen for laboratory study.</td>
</tr>
<tr>
<td>G*</td>
<td>Broken core - 16 boxes of core from 7 drill holes, 5 holes to 15.2 m with 2 samples per hole, 0 to 7.6 m and 7.6 to 15.2 m, 2 holes to 22.9 m with 3 samples per hole, 0 to 7.6 m, 7.6 to 15.2 m, and 15.2 to 22.9 m</td>
</tr>
</tbody>
</table>

*Note: Because all 7 samples were from one deposit and similar, material from the intersections chosen for study were combined to make one large sample for processing.*
Table 2 - Sieve analysis, washed attrition products

Sieve analysis wt %

<table>
<thead>
<tr>
<th>Size (μm)</th>
<th>A</th>
<th>B</th>
<th>C_T</th>
<th>C_B</th>
<th>D</th>
<th>E</th>
<th>F</th>
<th>G</th>
</tr>
</thead>
<tbody>
<tr>
<td>+420</td>
<td>6.6</td>
<td>6.0</td>
<td>12.3</td>
<td>13.5</td>
<td>14.6</td>
<td>21.0</td>
<td>16.2</td>
<td>14.1</td>
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<tr>
<td>-420, +297</td>
<td>20.3</td>
<td>36.6</td>
<td>30.3</td>
<td>36.5</td>
<td>29.7</td>
<td>46.9</td>
<td>33.1</td>
<td>27.0</td>
</tr>
<tr>
<td>-297, +210</td>
<td>16.0</td>
<td>27.9</td>
<td>25.0</td>
<td>25.2</td>
<td>18.1</td>
<td>18.9</td>
<td>20.0</td>
<td>18.4</td>
</tr>
<tr>
<td>-210, +149</td>
<td>30.7</td>
<td>20.6</td>
<td>19.9</td>
<td>17.4</td>
<td>22.9</td>
<td>8.5</td>
<td>18.9</td>
<td>19.8</td>
</tr>
<tr>
<td>-149, +105</td>
<td>22.7</td>
<td>8.1</td>
<td>10.3</td>
<td>6.6</td>
<td>11.0</td>
<td>3.9</td>
<td>10.0</td>
<td>14.7</td>
</tr>
<tr>
<td>-105</td>
<td>3.7</td>
<td>0.8</td>
<td>2.2</td>
<td>0.8</td>
<td>3.7</td>
<td>0.8</td>
<td>1.5</td>
<td>6.0</td>
</tr>
<tr>
<td>Total</td>
<td>100.0</td>
<td>100.0</td>
<td>100.0</td>
<td>100.0</td>
<td>100.0</td>
<td>100.0</td>
<td>100.0</td>
<td>100.0</td>
</tr>
</tbody>
</table>
Table 3 - Mineralogical examination, attrition products

<table>
<thead>
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<th>Sample No.</th>
<th>Comment</th>
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</thead>
<tbody>
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<td>A</td>
<td>quartz with traces of calcite and amphibole; most quartz grains are clear and rounded to subrounded with few overgrowths; no inclusions were observed in the quartz nor coatings on the grains; a few opaque, free grains were observed.</td>
</tr>
<tr>
<td>B</td>
<td>quartz with minor calcite and possibly amphibole; quartz grains are rounded to subangular with overgrowths on many grains; most grains are liberated and inclusions in grains is rare; contaminants, principally carbonate, are smaller than the quartz grains.</td>
</tr>
<tr>
<td>C_t</td>
<td>quartz with trace amounts of K-feldspar and calcite, indications of amphibole and dolomite; quartz grains are rounded to sub-angular; feldspar grains, without twinning or alteration, are not discernable; most of the carbonate grains are liberated and smaller than the quartz grains; some unidentified, opaque grains were observed, mostly attached to quartz grains.</td>
</tr>
<tr>
<td>C_b</td>
<td>quartz with trace amounts of carbonate, probably both calcite and dolomite, and amphibole; quartz grains are rounded to sub-angular and many have overgrowths; combined grains and included grains in quartz are rare; most contaminants are in small, free grains; rare opaque grains are usually attached to quartz.</td>
</tr>
<tr>
<td>D</td>
<td>quartz with trace amounts of K-feldspar and carbonate, both calcite and dolomite; quartz grains are sub-rounded; a few large aggregates of quartz and carbonate were observed; inclusions were rare; no feldspar was observed.</td>
</tr>
<tr>
<td>E</td>
<td>quartz with trace amounts of carbonate, probably both calcite and dolomite, and amphibole; quartz occurs in rounded grains with few inclusions; some grains have thin partial coatings of fine material; most of the contaminants occur as fine, dust-like particles; a few larger, opaque grains and grains of amphibole were observed; inclusions in quartz were rare.</td>
</tr>
<tr>
<td>Sample No.</td>
<td>Comment</td>
</tr>
<tr>
<td>-----------</td>
<td>---------</td>
</tr>
<tr>
<td>F</td>
<td>not yet available</td>
</tr>
</tbody>
</table>

- quartz with trace amounts of feldspar and amphibole; quartz grains were mostly clear, rounded to sub-rounded, with some overgrowths; inclusions were rare; grains with partial coatings of clay and ferruginous material were more common than in other samples; rare aggregate of quartz and carbonate were present.
Table 4 - Summary of typical results

<table>
<thead>
<tr>
<th>Product*</th>
<th>Recovery</th>
<th>Chemical analysis %</th>
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<tbody>
<tr>
<td></td>
<td>wt %</td>
<td>SiO₂</td>
</tr>
<tr>
<td>Sample A</td>
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<td></td>
</tr>
<tr>
<td>Head</td>
<td>100.0</td>
<td>96.86</td>
</tr>
<tr>
<td>Flotation feed</td>
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<td>99.80</td>
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<tr>
<td>Flotation conc</td>
<td></td>
<td></td>
</tr>
<tr>
<td>– average</td>
<td>99.47</td>
<td>0.024</td>
</tr>
<tr>
<td>– best</td>
<td>99.50</td>
<td>0.024</td>
</tr>
<tr>
<td>Leached float conc</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Sample B</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Head</td>
<td>100.0</td>
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<td>87.1</td>
<td>94.40</td>
</tr>
<tr>
<td>Flotation conc</td>
<td></td>
<td></td>
</tr>
<tr>
<td>– average</td>
<td>98.33</td>
<td>0.034</td>
</tr>
<tr>
<td>– best</td>
<td>99.59</td>
<td>0.029</td>
</tr>
<tr>
<td>Leached float conc</td>
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<td>0.039</td>
</tr>
<tr>
<td>Sample Cₐ</td>
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<td></td>
</tr>
<tr>
<td>Head</td>
<td>100.0</td>
<td>95.01</td>
</tr>
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<td>Flotation feed</td>
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<td>97.50</td>
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<td>Flotation conc</td>
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<tr>
<td>– average</td>
<td>98.65</td>
<td>0.133</td>
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<td>– best</td>
<td>98.82</td>
<td>0.107</td>
</tr>
<tr>
<td>Leached float conc</td>
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<td>0.061</td>
</tr>
<tr>
<td>Sample Cₐ</td>
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<tr>
<td>Flotation conc</td>
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</tr>
<tr>
<td>– average</td>
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<td>0.037</td>
</tr>
<tr>
<td>– best</td>
<td>99.54</td>
<td>0.030</td>
</tr>
<tr>
<td>Leached float conc</td>
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<td>0.025</td>
</tr>
<tr>
<td>Sample D</td>
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<tr>
<td>Head</td>
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<td>93.70</td>
</tr>
<tr>
<td>Flotation feed</td>
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<td>96.99</td>
</tr>
<tr>
<td>Flotation conc</td>
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<td></td>
</tr>
<tr>
<td>– average</td>
<td>98.87</td>
<td>0.053</td>
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<td>– best</td>
<td>99.05</td>
<td>0.048</td>
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<tr>
<td>Leached float conc</td>
<td></td>
<td>0.028</td>
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Table 4 - Summary of typical results (Cont'd)

<table>
<thead>
<tr>
<th>Product*</th>
<th>Recovery wt %</th>
<th>Chemical analysis %</th>
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<tbody>
<tr>
<td></td>
<td>SiO₂</td>
<td>Fe₂O₃</td>
</tr>
<tr>
<td>Sample E</td>
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<td></td>
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<tr>
<td>Head</td>
<td>100.0</td>
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<td>99.80</td>
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<td>Flotation conc</td>
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<td></td>
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<tr>
<td>- average</td>
<td>99.63</td>
<td>0.044</td>
</tr>
<tr>
<td>- best</td>
<td>99.65</td>
<td>0.034</td>
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<tr>
<td>Leached float conc</td>
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<td></td>
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<tr>
<td>Sample F</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Head</td>
<td>100.0</td>
<td>95.94</td>
</tr>
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<td>Flotation feed</td>
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<td>Flotation conc</td>
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<tr>
<td>- average</td>
<td>99.48</td>
<td>0.046</td>
</tr>
<tr>
<td>- best</td>
<td>99.37</td>
<td>0.031</td>
</tr>
<tr>
<td>Leached float conc</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Sample G</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Head</td>
<td>100.0</td>
<td>88.50</td>
</tr>
<tr>
<td>Flotation feed</td>
<td>83.8</td>
<td>93.90</td>
</tr>
<tr>
<td>Flotation conc</td>
<td></td>
<td></td>
</tr>
<tr>
<td>- average</td>
<td>98.80</td>
<td>0.083</td>
</tr>
<tr>
<td>- best</td>
<td>99.11</td>
<td>0.043</td>
</tr>
<tr>
<td>Leached float conc</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

*Flotation feed is equivalent to washed attrition product
Table 5 - Flotation reagent consumption (kg/t)

<table>
<thead>
<tr>
<th>Sample</th>
<th>Test no.</th>
<th>Sulphide float</th>
<th>Calcite float</th>
<th>pH</th>
<th>Iron oxide &amp; feldspar flotation</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Eth. Xan</td>
<td>Oleic Acid</td>
<td></td>
<td>Armac † Aero. 840 Duo. TDO Kerosene</td>
</tr>
<tr>
<td>A</td>
<td>30</td>
<td>0.07</td>
<td>0.27</td>
<td>2.5-3.0</td>
<td>0.37 - - -</td>
</tr>
<tr>
<td>B</td>
<td>45</td>
<td>0.07</td>
<td>0.49</td>
<td>2.5-2.8</td>
<td>0.25 - - -</td>
</tr>
<tr>
<td>C&lt;sub&gt;T&lt;/sub&gt;*</td>
<td>52</td>
<td>0.08</td>
<td>0.40</td>
<td>5.0</td>
<td>- 0.28 - -</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>2.1-2.5</td>
<td>0.28</td>
</tr>
<tr>
<td>C&lt;sub&gt;B&lt;/sub&gt;</td>
<td>49</td>
<td>0.10</td>
<td>0.61</td>
<td>2.1-2.4</td>
<td>- - 0.34 -</td>
</tr>
<tr>
<td>D*</td>
<td>54</td>
<td>0.09</td>
<td>0.83</td>
<td>3.5-4.0</td>
<td>- 0.31 - -</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>2.1-2.5</td>
<td>0.31</td>
</tr>
<tr>
<td>E*</td>
<td>56</td>
<td>-</td>
<td>-</td>
<td>4.0</td>
<td>0.25 - 0.24</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>2.5-2.8</td>
<td>0.25 0.16</td>
</tr>
<tr>
<td>F</td>
<td>38</td>
<td>0.09</td>
<td>0.38</td>
<td>2.5-2.8</td>
<td>- - 0.45 -</td>
</tr>
<tr>
<td>G**</td>
<td>60</td>
<td>0.05</td>
<td>0.78</td>
<td>2.3-2.5</td>
<td>0.45 - - -</td>
</tr>
</tbody>
</table>

* The petroleum sulphonate flotation stage was conducted before the feldspar flotation stage.

** The sulphide flotation stage comprised a combined sulphide & graphite float.
Table 6 - Collector(s) employed in achieving optimum results

<table>
<thead>
<tr>
<th>Sample</th>
<th>Test no.</th>
<th>Collector(s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>30</td>
<td>Armac T</td>
</tr>
<tr>
<td>B</td>
<td>45</td>
<td>Armac T</td>
</tr>
<tr>
<td>Cₜ</td>
<td>52</td>
<td>Aerofloat 840 followed by Duomeen TDO</td>
</tr>
<tr>
<td>Cₜ</td>
<td>49</td>
<td>Duomeen TDO</td>
</tr>
<tr>
<td>D</td>
<td>54</td>
<td>Aerofloat 840 followed by Duomeen TDO</td>
</tr>
<tr>
<td>E</td>
<td>56</td>
<td>Aerofloat 840 followed by Armac T</td>
</tr>
<tr>
<td>F</td>
<td>38</td>
<td>Duomeen TDO</td>
</tr>
<tr>
<td>G</td>
<td>60</td>
<td>Armac T</td>
</tr>
</tbody>
</table>
Table 7 - Foundary test results

<table>
<thead>
<tr>
<th>Sample</th>
<th>Moist</th>
<th>Mould</th>
<th>Perm</th>
<th>Gr. comp.</th>
<th>Gr. tensile</th>
<th>AFS No.</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>3.4</td>
<td>74</td>
<td>109</td>
<td>6.3</td>
<td>10.3</td>
<td>65.1</td>
</tr>
<tr>
<td>B</td>
<td>2.8</td>
<td>75</td>
<td>175</td>
<td>6.9</td>
<td>12.9</td>
<td>51.4</td>
</tr>
<tr>
<td>C</td>
<td>not enough sand</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>C_B</td>
<td>3.6</td>
<td>74</td>
<td>238</td>
<td>7.6</td>
<td>12.0</td>
<td>45.9</td>
</tr>
<tr>
<td>D</td>
<td>3.3</td>
<td>73</td>
<td>155</td>
<td>6.1</td>
<td>11.7</td>
<td>50.3</td>
</tr>
<tr>
<td>E</td>
<td>3.3</td>
<td>76</td>
<td>297</td>
<td>5.7</td>
<td>10.7</td>
<td>39.9</td>
</tr>
<tr>
<td>F</td>
<td>3.4</td>
<td>75</td>
<td>167</td>
<td>6.6</td>
<td>12.7</td>
<td>54.0</td>
</tr>
<tr>
<td>G</td>
<td>3.3</td>
<td>73</td>
<td>146</td>
<td>7.3</td>
<td>12.6</td>
<td>55.1</td>
</tr>
</tbody>
</table>
REFERENCES

1. Information Systems Division, Mineral Policy Sector, Energy, Mines and Resources, Ottawa, Canada.


ACKNOWLEDGEMENTS

The authors acknowledge, with thanks, the cooperation of Dr. M.A. Klugman of the Ontario Ministry of Natural Resources in providing open file reports for study and drill core samples for laboratory investigation, and of the following CANMET staff: G.A. Brown for assistance in the processing and investigation of the sandstone samples; M. Raicevic for the Sala magnetic separation tests; R.M. Buchanan for mineralogy; B.H. Lucas for the acid leach study; and E.I. Szabo and R. Gauthier for the foundry evaluation.
Summary of Property and Development Details For
ELGIN SILICA LIMITED
1983 April 15 - E.B.R.

A. PROPERTY
We first visited the Fred Lake property in 1962. At that time it was under lease by Canadian Silica Ltd. We did some diamond drilling on the Onera property nearby. In 1964 we leased 300 acres of the Lake Farm. In 1968 we did some diamond drilling and test quarrying. Material has been hauled to Cambridge ever since. In 1978 we purchased 25 acres south of the road from Lake. In 1981 we optioned about 220 acres, including 45 acres south of the road and about 175 acres north of the road. We obtained a quarry permit covering the entire 245 acres. We also negotiated lease-purchase options with Cooper, Coon and Ostrom. Ostrom not complete. Exploration permit from Natural Resources on further parcel.

Lake Farm - Bastard Township, part Lots 26 & 27, Con. VII
± 175 acres north of Hartsgravel Road

Cooper Farm - part Lots 26 & 27, Con. VI
± 200 acres

Coon Farm - Part Lot 27, Con. VII
± 75 acres

Ostrom - Part Lot 28, Con. VII
± 200 acres (not completed)

Ministry of Natural Resources - Part Lot 28 - 29 - 30, Con. X
± 200 acres. Exploration permit only.

850 possible acres

All development work to date has been confined to the Lake Farm. It is known however from Rochester Pittsburgh work in 1958 that all properties consist of extensive silica deposits.

B. DIAMOND DRILLING
5 holes were drilled in 1968 1P, 2P, 1A, 2A, 3A
7 holes were drilled in 1973 1 - 7
10 holes were drilled in 1981 10 - 19

Holes were located to cover extreme limits of Lake Farm north of Hartsgravel Road (± 175 acres). Deep section around hole 13 was explored in detail.

Diamond drilling by Eastern Ontario Diamond Drilling Cores inspected by
Kriens See Section E
G. Robert Guillet Section D
Klugman & Bourque Section C
C. Michael Klugman (Mines Co-ordinator for Eastern Ontario)
   Marika Bourque (Stratigrapher) File XVII-25.1

Old Quarry was a river bed. Main property was a lake. Examined early drillings
(holes No. 1, 6 & 7). Klugman very positive. Plant requires ½ million tons per
year for 10 years.

Klugman returned Aug 4th to inspect new cores (to No. 13). Excited about #13
friable core. Suggests investigation of hydraulic mining and transportation.

September '81 - Klugman invites us to a meeting in Toronto with Allison of DREE
They propose that we excavate and they finance a ramp to 50'. They back out Nov.
1st but we carry on and excavate ramp to about 40'. Build road to area of hole 13.

D. G. Robert Guillet - Geologist - (File 25.1)

Inspected property and cores about August 20th, 1981. Very impressed with cores
particularly lower zone below 30 ft. thickness from 28 - 77 ft. (later tested as
type G). Recommended further drilling and logging, beneficiation testing, and
acquiring more property. All these were done.

Further report (November 1982) estimates property clear of buildings at 160 acres
Estimates lower beds (G) reserves of 12 million tons. Upper beds (0-40) types
A & B, 24 million tons.

E. IMD Laboratories - Jack Kriens, President (File 25.3.2)

Inspected property July 31st, 1981. Extensive experience with Indusmin in silica
chemistry processing. Planning on new lab on Lesmill, in Toronto. Suggested con-
sulting Guillet on geology and mineralogy. Inspected drill cores 1-13. Suggested
a deposit of 15 million tons desirable.

Report 9006 August 12th, 1981 - some cores excellent. Recommended hole No. 13
be examined and tested. This was carried out by Lakefield Research (Section F).

August 28th - visit to laboratory - some samples under microscope, some round,
some sub-angular. Magnetic separation cost 15-20$/ton + capital. Flotation
cost and drying $5-6 per ton.

November - work on drill core #17 commenced. Divided into sections.

<table>
<thead>
<tr>
<th>Section</th>
<th>Grade</th>
</tr>
</thead>
<tbody>
<tr>
<td>0 - 18'</td>
<td>0.37 Fe₂O₅</td>
</tr>
<tr>
<td>18 - 30'</td>
<td>1.0 Fe₂O₅</td>
</tr>
<tr>
<td>30 - 50'</td>
<td>0.43 Fe₂O₅</td>
</tr>
<tr>
<td>50 - 70'</td>
<td>0.13 Fe₂O₅</td>
</tr>
</tbody>
</table>

0 - 18 attrition scrub needed - mostly -40 mesh, not round
enough and too many agglomerates for blasting sand.

18 - 30 slightly more rounded. More 'shale' coating.

30 - 50 somewhat more rounded - bits of 'shale' and pyrite.

50 - 70 somewhat more rounded - cleaner - some pyrite
20 - 40 mesh, some round grains - 50% agglomerates.

Magnetic separation not recommended.
E. (Continued...)  

December 1981 - Review of Lakefield report (Section G). Hopes he can avoid extensive scrubbing, especially acid scrubbing.  
Magnetic separation attempted -  
Dry solids leach attempted -  
30-50' level Fe₂O₃ reduced to 0.06 with fatty acid + xanthate.  
50-70' level Fe₂O₃ reduced to 0.03 with fatty acid + xanthate.  

December - Ramp material - 30-40' type B - pebble milling by us reduces Fe₂O₃ from 0.65 to 0.19.  
Flotation to 0.09 (2 stage fatty acid + xanthate).  
Similar material rod milled by us reduced to 0.07.  

April 1982 4 types of silica tested rod mill & flotation  
<table>
<thead>
<tr>
<th></th>
<th>Fe₂O₃</th>
<th>Fe₂O₃</th>
<th>Fe₂O₃</th>
</tr>
</thead>
<tbody>
<tr>
<td>A ramp 0 - 30', unmilled</td>
<td>0.42</td>
<td>0.23</td>
<td>0.09</td>
</tr>
<tr>
<td>B ramp 30 - 40',</td>
<td>0.43</td>
<td>0.22</td>
<td>0.06</td>
</tr>
<tr>
<td>C white core from Holes 15 &amp; 16</td>
<td>0.13</td>
<td>0.12</td>
<td>0.02</td>
</tr>
<tr>
<td>D Cambridge stockpile</td>
<td>0.67</td>
<td>0.27</td>
<td>0.08</td>
</tr>
<tr>
<td>G Guillet 'good' core</td>
<td></td>
<td></td>
<td>0.05</td>
</tr>
</tbody>
</table>

All samples contained iron particles about which silica crystals agglomerated. Iron from mill.  

September 1982 - G silica repeated with magnetic separation. No improvement - 0.05% Fe₂O₃.  
B material contained 'shale' which on analysis by UMEX laboratories was determined to be carbonate fluorapatite (calcium - phosphorous - fluorine) but in extent less than 0.01%.  

F. CANMET (Ottawa) Tests on Diamond Drill Core #18. 1981 & 1982 (File XVI-25.3)  
Core was tested top beds 0-12m and bottom beds 12-25m.  
Samples were handled separately through crushing, grinding, scrubbing, 4 stage flotation, acid leach, and for top sample magnetic separation.  

Top  - Fe₂O₃ was reduced from 0.48 % by flotation to 0.13 % and further to 0.06 % by either acid leach or by magnetic separation. Both were not tried.  
Bottom - Fe₂O₃ was reduced from 0.16 by flotation to 0.04 % and further to 0.025 % by acid leach.  

G. Lakefield Research file XVI-25.3.1  
Testing of core No. 13 (Sept. - Nov. 1981)  
Core was tested in 2 sections, upper 0 - 30 ft., lower 30 - 117 ft.  
Ground materials were leached with combined acids (HCl, HNO₃, HF) to determine the base level of Fe₂O₃ content.  

0 - 30 ft. sample reduced from Fe₂O₃ 0.81 to 0.009%  
30 - 117 ft. sample reduced from Fe₂O₃ 0.27 to 0.003%  
This indicates that iron is not within the grains.
Beneficiation of lower beds (30' - 117') Fe₂O₃ 0.27%.

Flowsheet I
Grind to -35 mesh, scrub 10 min., 2 stage condition - flotation produced 84% at 0.06 Fe₂O₃ (P)
Acid scrub 30 minutes produced 83% at 0.035 Fe₂O₃
Magnetic separation of (P) produced 74% at 0.044 Fe₂O₃

Flowsheet II
Grind to -28 mesh, scrub 30 min., 2 stage condition - flotation produced 80% at 0.034 Fe₂O₃

Flowsheet III
Grind to -14 mesh, scrub 30 min., acid scrub 30 min. - 2 stage condition, flotation, produced 81% at 0.025 Fe₂O₃

Beneficiation of upper beds (0' - 30') Fe₂O₃ 0.81%

Flowsheet IV
Grind to -48 mesh, scrub 10 min., condition 2 stage float 1 stage, acid scrub 2 each 30 min. produced 70% yield at 0.05 Fe₂O₃

Flowsheet V
Grind to -28, scrub 30 min., condition 2 stage, float 3 stage produced 56% at 0.15 Fe₂O₃

Flowsheet VI
Grind to -14, scrub 30 min., acid scrub 30 min. condition 2 stage, float 1 stage, screen -35 for regrind 26%
produced 50% at 0.04 Fe₂O₃ (without regrind)

There is some concern that these processes have limited practical application since

1. Scrubbing requires expensive equipment, high power consumption, and produces excess fines.
2. Acid is expensive, messy, and hard to remove.
3. Multi stage conditioning and flotation are expensive.

H. Tests by U.S. Bureau of Mines (file X11-25.3)

(Dr. Geo. Savanik)

A one ton block was shipped to Iowa to investigate the possibilities of hydraulic mining and transportation. A pressure of 4200 psi was required and erosion was slow. The process was not considered practical.
EXAMINATION AND IDENTIFICATION
OF HEAVY MINERALS CONTAINED IN SILICATE MATERIAL
(P.O. 65424)

L. Patterson
J.-J. Lefebvre
Methodology:

A heavy liquid separation using tetrabromoethane (sp.gr. 2.95) was performed on the first sample. The resulting phases were then rinsed, dried and the heavy fraction was further separated using a Franz electromagnetic separator.

An x-ray spectrum was obtained for the non-magnetic fraction, however, due to a lack of material, it was not possible to x-ray the magnetic phase. For this reason, a second sample was obtained.

Once again, a heavy liquid separation was performed on the sample with the sinking phase being further separated, using the Franz. Next, the gray shale-like material was hand separated and then an x-ray spectrum was produced using the Debye-Scherrer camera film method. The two remaining phases (non-magnetic and magnetic) were then prepared as polished thin sections and later studied using an optical microscope. The results of the above procedures are described below.

First Sample

Sinking phase non-magnetic: The x-ray spectrum indicated the presence of Quartz, Calcite, K-feldspar and Dolomite or Ankerite.

Second Sample

Sinking phase non-magnetic: gray shale-like material: results of Debye-Scherrer spectrum indicates the presence of Fluorapatite and Quartz.

Sinking phase non-magnetic: optical studies on polished thin section indicates the presence of the following:

- abundant aggregates of pyrite and marcasite, anhedral crystals of pyrite and marcasite;
- also aggregates of finely crystallized rutile and abundant quartz with pyrite inclusions;
- abundant rounded grains of zircon (40%);
- minor carbonate incrustations.

Sinking phases - magnetic: results of petrographic study follows:

- minute inclusions of pyrite in quartz;
- anhedral oxydized pyrite locked with quartz;
- angular inclusions of pyrite and marcasite in quartz;
- goethite;
- aggregates of finely crystallized rutile and pyrite;
Second Sample (Cont'd.)

- minor ilmenite;
- one grain of chalcopyrite with quartz;
- minor dark brown-green biotite;
- anhedral grains of green amphibole;
- minor rounded zircon and tourmaline;
- numerous carbonate incrustations;
- rare quartz with opaque inclusions.

Weight distribution - second sample:

<table>
<thead>
<tr>
<th>Weight Category</th>
<th>Weight (gm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Floating phase</td>
<td>471.68</td>
</tr>
<tr>
<td>Sinking - non-magnetic</td>
<td>.01</td>
</tr>
<tr>
<td>Sinking - magnetic</td>
<td>.13</td>
</tr>
<tr>
<td>Sinking - gray-shale type material</td>
<td>&lt; .01</td>
</tr>
</tbody>
</table>
January 18, 1983

Mr. E.B. Ratcliffe,
Chairman, Angelstone Limited,
P.O. Box 3190,
Cambridge (Preston)
Ontario N3H 4S8

Dear Ed:

The Ontario silica study is now nearing completion and we are now getting the report into shape for publication. Results on your samples were quite interesting as noted below.

These samples were, I believe, taken from near Phillipsville. Could you please indicate the precise location on the attached map and, also, provide information re County, Lot, Concession and Township. Many thanks.

Regards,

Ron

R.K. Collings, Head,
Non-Metallic Minerals Section.

Typical Products - Chemical Analyses - Wt%:

<table>
<thead>
<tr>
<th>Sample</th>
<th>SiO₂</th>
<th>Fe₂O₃</th>
<th>Al₂O₃</th>
<th>LOI</th>
</tr>
</thead>
<tbody>
<tr>
<td>#2 - 0 to 40 ft</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Head Sample</td>
<td>95.0</td>
<td>0.48</td>
<td>1.00</td>
<td>0.94</td>
</tr>
<tr>
<td>Flotation Conc.</td>
<td>98.7</td>
<td>0.14</td>
<td>0.49</td>
<td>0.32</td>
</tr>
<tr>
<td></td>
<td></td>
<td>0.03</td>
<td>0.07</td>
<td></td>
</tr>
<tr>
<td>#4 - 40 to 82 ft</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Head Sample</td>
<td>98.1</td>
<td>0.16</td>
<td>0.14</td>
<td>0.38</td>
</tr>
<tr>
<td>Flotation Conc.</td>
<td>99.5</td>
<td>0.03</td>
<td>0.07</td>
<td>0.15</td>
</tr>
</tbody>
</table>

Note: Results from acid leach tests not yet available.

RKC/mv
c.c. P. Andrews

Canada
To: E.B.R.
C.J.
File IV-90

Discussion with Ron Collings - Ottawa
Re: His report on Eastern Ontario silica sands

It appears that our -40 +85 material is as good as any they found but they do have other material that comes as clean and is at the surface.

He did not think much could be done with the +40 foot material.

He did not know what floatation was necessary on the individual samples. This will be in the report.

The report will not be published for 3 or 4 months (at least) but as soon as a draft is ready we are to get one.
December 17, 1982

Mr. E.B. Ratcliffe,
Chairman Angelstone Ltd.
P.O. Box 3190,
Cambridge, Ontario.
N3H 4S8

Dear Ed,

Enclosed please find the flotation samples from the last test. Also, enclosed in a small plastic vial are the magnetic particles removed from the sample before flotation. Folded in the paper are additional magnetic particles removed by a hand magnet after flotation.

Product recovery from this test was 72.3% by weight.

The chemical analysis of the flotation products were as follows:

\[
\begin{align*}
\text{SiO}_2 & \quad 99.0 \\
\text{Al}_2\text{O}_3 & \quad 0.10 \\
\text{CaO} & \quad 0.09 \\
\text{MgO} & \quad 0.03 \\
\text{Na}_2\text{O} & \quad 0.06 \\
\text{K}_2\text{O} & \quad 0.04 \\
\text{Fe}_2\text{O}_3 & \quad 0.053 \\
\text{MnO} & \quad 0.00 \\
\text{TiO}_2 & \quad 0.03 \\
\text{P}_2\text{O}_5 & \quad 0.01 \\
\text{L.O.I.} & \quad 0.38 \\
\end{align*}
\]

- PPM

Zr 50
Sr 20
Cr 30

cont'd......
As you know we did the flotation twice, after obtaining the rather disappointing 0.12% Fe₂O₃ on the first test.

As agreed, we have not billed you for the flotation work, as I felt that I should have picked up earlier that your method of grinding caused contamination with abraded iron in some of the previous samples.

We have all of your samples stored in our laboratory. You may wish to have them returned to you. If so, let me know and I will ship them to you.

Please let me know if there is anything I may be able to do for you.

Wishing you a Merry Christmas and a good business year in 1983.

Yours sincerely,

I.M.D. LABORATORIES LTD.

J. Kriens

Encls.

/jp
January 13th, 1984

Bruce Gordon,  
O.M.E.P.,  
Room #4649,  
Queen's Park,  
TORONTO, Ontario,  
M7A 1W3

Dear Bruce;

Per our discussion Wednesday I am re-compiling the information which I submitted to you under date of August 17th, 1983. Apparently this information never reached your hands.

I think this is a good time to review the total project. Basically the development of a silica quarry and beneficiation facility on this property has been dropped by Arriscraft Corporation-Elgin Silica Limited. As the submitted reports indicate there is a substantial amount of high grade silica on the property, however the best material is all below the 30 foot level. There does not appear to be an effective way to extract this material unless the top 30 feet could be sold in a lower grade silica market.

It should be mentioned that there are some excellent high grade silica deposits in the mid-western U.S.A. and these are the source of the majority of the silica used in glass production in the northeastern portion of the continent. The production costs from these deposits is very low but transportation can be significant. For this Ontario property to be usable the beneficiation procedure had to be relatively cost competitive. This does not appear to be the case at this time.

Continued page two/...
The final factor to be considered was reliability of transportation. Material coming in from the U.S.A. did go through a period (3-4 years ago) where supply was a problem due to transportation problems. This increased the glass companies' interest in local sources. Over the last few years there have been no such problems and the glass companies are no longer showing much interest in new sources.

As you will notice we have very few formal reports. Most of the work on this project consisted of collecting samples and doing beneficiation studies. These studies were mainly done by I.M.D. Laboratories (Jack Kriens). Little time was spent on reports as information was conveyed over the phone and instructions and discussions over how testing should proceed were done in person.

I trust this information is sufficient.

Yours truly,
ARRISCRAFT CORPORATION

Randy White, C.A.
Vice-President
RW/smog

Encls.
July 28, 1983

Mr. E.B. Ratcliffe,  
Chairman of the Board,  
Arriscraft Corporation,  
Box 3190,  
Cambridge, Ontario.  
N3H 4S8

Dear Ed,

As per our discussion, I have reviewed the report by R.K. Collings and P.R.A. Andrews of Energy Mines & Resources. In addition, I reviewed the excellent summary prepared by Randy White.

From this collective data and the work I did for you, I have drawn the following conclusions which I suspect will not be surprising to you.

My conclusions are:

1) There is no question the lower level material is of a quality suitable for flint glass production.

2) The upper level material, as confirmed by independently conducted investigations by various sources, conclusively shows that the upper level material does not respond to economically viable processing.

The combination of flotation plus high intensity wet magnetic separation, as reported in the report by Collings & Andrews upgrades the upper level material to a silica grade that is considered suitable for the production of coloured container or fiberglass.

3) It is, in my opinion, significant that in all cases the various investigators have achieved similar results on both the upper and lower levels of material. In some instances, this has been achieved by extensive, multi-stage flotation or attrition scrubbing over long periods including an acid scrub.

cont'd....
4) Acid leaching with hydrofluoric, hydrochloric and nitric acids as conducted by Lakefield Research reduced the residual Fe$_2$O$_3$ levels to less than 0.01%. This indicates that all the contaminant minerals are present as distinct mineral grains as attached grains on silica particles or part of agglomerates as a binding material.

5) Leaching with large quantities of hot sulfuric acid over extended periods as conducted by Collings & Andrews resulted in a silica product with a high residual Fe$_2$O$_3$ content well above the level achieved by Lakefield Research. This indicates that the mineral contributing most of the residual iron content is not soluble in hot sulfuric acid. Most sulfide minerals are not soluble in sulfuric acid.

6) It is my considered opinion that most of the residual iron remaining in the lower level ore after flotation is present as grains of contaminants attached to silica grains or as part of the matrix material between agglomerated grains. The contaminating minerals are likely to be of small particle size.

7) Four stages of flotation, acid scrubbing, acid leaching with sulfuric acid is not a viable economic solution for the production of high quality silica. It is my contention that the solution must be found in the grinding process preparing the products for flotation. In my opinion more work on this aspect of the process is warranted.

It is unfortunate, but all indications up to this point are that the top level silica is difficult, if not impossible, to process by an economically viable process to a high quality silica product suitable for the production of flint glass. It is a difficult decision to proceed with additional work. It is equally difficult to abandon any further efforts.

To be frank, I believe there has been too much emphasis on material from the ramp and insufficient emphasis on the core from the various holes.

cont'd....
The summary by R. White indicates that most or all of the recent drilling has been on just one of the properties under your control. Possibly, the other properties offer better opportunities.

I trust this letter has provided you with a summary of my conclusions of the work conducted on your silica deposit(s).

Yours sincerely,

I.M.D. LABORATORIES LTD.

Jack Kriens
President

/jp
I.M.D. Laboratories Ltd.
260 Lesmill Rd.,
Toronto, Ontario
M3B 2T5

Attention: Mr. J. Kriens

Re: Work Performed by our Laboratory
(P.O. 65424)

Charges related to the study of silicate material, as per report LAB0 0083, dated April 22, 1982:

Three heavy liquid separations $150
Two Franz magnetic separations 45
One conventional x-ray spectrum 45
One Debye-Scherrer film spectrum 80
Preparation of two polished thin sections 45
Study of two polished thin sections 90

Total $455
April 13th, 1983

Jack Kriens,
I.M.D. Laboratories Ltd.,
Industrial Minerals Processing,
260 Lesmill Road,
DON MILLS, Ontario,
M3B 2T5

Dear Jack;

Enclosed is a copy of a draft report received from Ron Collings and some comments by Randy. After you peruse it we can talk about it.

Our material (shown as C1 and C2 is from a D. D. Hole in the same area as No. 17 which you tested. Superficially the results are in the same general area.

You will note that C1 achieved only 0.13 Fe2O3 by flotation but was subsequently reduced to 0.06 by either magnetic separation or acid leach. This raises the question of using more effective flotation to reach below 0.1 followed by both magnetic separation and acid leach.

The top material is what we have a lot of and we probably have to figure out what to do with that first.

Let's be in touch.

Yours truly,
ARRISCRRAFT CORPORATION

E.B. Ratcliffe,
Chairman of the Board
EBR/smcg

Encls.
(1) Description:

They tested silica from a number of eastern Ontario locations. There were 5 locations which they selected and 2 samples, one supplied by ourselves (C_T & C_B) (top and bottom) and one supplied by another potential supplier (© © © © © ©). From the drill core description it is clear that the other potential supplier's sample is sample #G.

(2) Beneficiation (C_T = Top 20 ft.; C_B = Bottom; Hole #13)

**Crushing/Grinding**

<table>
<thead>
<tr>
<th>Attraction Scrub</th>
</tr>
</thead>
<tbody>
<tr>
<td>Flotation</td>
</tr>
</tbody>
</table>

**C_T**

i) Sulphide - using xanthate  
ii) Calcite - using oleic acid  
iii) Iron Oxide - Afrofloat 840  
iv) Felspar - duo TDO

**C_B**

i) Sulphide - xanthate  
ii) Calcite - oleic acid  
iii) Felspar - duo TDO

(4 stage)  
(3 stage)

All other samples tested needed a three stage flotation.

The reagent consumption during floatation was generally at or above average for our material.

**Acid Leach**

This acid leach was done to determine just how low it would be possible to reduce the level of Fe_2O_3.

(3) Results:

With the exception of our C_T material (+ 30 ft) all samples were reduced to at or below 0.03% Fe_2O_3 after the acid leach. (our C_T went to 0.06 - probably unacceptable.) Also to 0.06 with magnetic separation.

If we rank the results by Fe_2O_3 content after floatation the results are as follows:
<table>
<thead>
<tr>
<th>Sample</th>
<th>Fe$_2$O$_3$ (Average)</th>
<th>Fe$_2$O$_3$ (Best)</th>
<th>Recovery %</th>
</tr>
</thead>
<tbody>
<tr>
<td># &quot;A&quot;</td>
<td>.024</td>
<td>.024</td>
<td>90.0</td>
</tr>
<tr>
<td># &quot;B&quot;</td>
<td>.034</td>
<td>.029</td>
<td>87.1</td>
</tr>
<tr>
<td># &quot;C_B&quot;</td>
<td>.037</td>
<td>.030</td>
<td>92.0</td>
</tr>
<tr>
<td># &quot;E&quot;</td>
<td>.044</td>
<td>.034</td>
<td>95.4</td>
</tr>
<tr>
<td># &quot;F&quot;</td>
<td>.046</td>
<td>.031</td>
<td>75.6</td>
</tr>
<tr>
<td># &quot;D&quot;</td>
<td>.053</td>
<td>.048</td>
<td>82.1</td>
</tr>
<tr>
<td># &quot;G&quot;</td>
<td>.083</td>
<td>.043</td>
<td>83.8</td>
</tr>
<tr>
<td># &quot;C_T&quot;</td>
<td>.133</td>
<td>.107</td>
<td>86.2</td>
</tr>
</tbody>
</table>

Conclusion:

Based upon these results there are sources as good (chemically) as our own property where the deposits start at or near the surface. Both samples "A" and "B" meet this criteria. They show a comparable yield and tend to use less reagents.

Sample "A" came from a location closer to Ottawa and sample "B" from a location close to our own.

Sample "G" comes from another potential supplier. This material could only be brought to an average iron of .083% with floatation. This is probably not acceptable.

In the introduction to this report they state that in 1981 there were 1.14 mt. of silica imported at a value of $18.5 million. (value = $16.23/ton). The source of this figure is apparently "Information Systems Division - Energy, Mines & Resources".

We have seen many different figures as to the price paid for silica sand and I suspect this figure does not include transportation. Perhaps this department might be able to tell us if it does and if not how much is paid for transportation.

* probably

George