GM 68239

RECOVERY OF LITHIUM CARBONATE FROM THE JAMES BAY SPODUMENE DEPOSIT

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RECOVERY OF LITHIUM CARBONATE FROM THE JAMES BAY SPODUMENE DEPOSIT

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EXECUTIVE SUMMARY

Lithium One, Inc. is developing the James Bay high-grade spodumene pegmatite deposit, located in northwestern Quebec, Canada. As part of this development program, Lithium One contracted Hazen Research, Inc. to complete a laboratory program to provide a preliminary flowsheet for producing lithium carbonate (Li_2CO_3) from the James Bay ore.

The flowsheet resulting from the Hazen program includes the following steps.

- The ore is crushed and ground to 80% passing 300 μm with 1.35 kg/t sodium hydroxide (NaOH) and deslimed with hydrocyclones. The deslimed ore is separated into coarse and fine fractions by wet-screening at 150 μm. Lithium recovery in the deslimed ore was 87–93%.

- A lithium concentrate is produced by flotation. Both the fine and coarse fractions are floated separately using Sylfat FA-1 (tall oil fatty acid) collector at 0.75 kg/t with one rougher stage and two stages of cleaning. Separate flotation circuits for the coarse and fine fractions were shown to provide a higher lithium grade and recovery. The laboratory flotation experiments achieved 80% lithium recovery in the combined concentrate. Improvements in lithium recovery from flotation are likely with further refinements of the flotation operation.

- The concentrates from both circuits are combined and calcined at 1,050°C for 1 h to convert the naturally occurring α-spodumene in the concentrate to acid-soluble β-spodumene.

- Acid is added at 478 kg/t to the calcine and the mixture baked at 250°C for 1 h to decompose the β-spodumene into lithium sulfate (Li_2SO_4), which is subsequently dissolved in the leaching step.

- A water leach is performed at 60°C for 30 min. Lime (CaO) is added to the leach to maintain the pH at 6.0–6.5. In this pH range, iron and aluminum solubilities are suppressed. Approximately 92% of the lithium in the calcined ore reported to the leach liquor.

- After solid–liquid separation, magnesium is removed from the leach solution by adding hydrated lime to precipitate magnesium hydroxide ($Mg(OH)_2$). The liquor from the subsequent solid–liquid separation step is concentrated by evaporation, and calcium is removed from the liquor by adding sodium carbonate (Na_2CO_3) to precipitate calcium carbonate ($CaCO_3$). Lithium losses in these purification steps were quite small at 0.3%.

- The resulting liquor is heated to 90–93°C, and lithium carbonate is precipitated from solution using sodium carbonate. Because lithium carbonate is partially soluble, only about 66% of the lithium is recovered as product. In a commercial operation, the resulting mother liquor, containing the unrecovered lithium, is recycled back to the lithium recovery step to reclaim most of this lithium as product. A mother liquor bleed is necessary to control impurity buildup.

The chemical analysis of the lithium carbonate product generated during this program is given in Table 1. If Lithium One's marketing data indicate that reduced impurity levels are required for the lithium carbonate product, additional purification steps will need to be considered.

Table 1. Analysis of Lithium Carbonate Product

Element	Assay, %
Li	17.1
CO_3^{2-}	72.1
Na	0.650
S	0.43
Mg	0.002
K	0.008
Ca	0.117
Fe	0.006
Al	0.020
Mn	0.001

Before proceeding to piloting this process on a continuous basis, a second phase of laboratory work is recommended. The goals of this Phase 2 laboratory program are to improve lithium recovery in several of the processing steps and to provide additional data necessary for generating a process mass balance and engineering data for designing a continuous pilot plant.

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INTRODUCTION AND SUMMARY

Lithium One, Inc. is developing the James Bay high-grade spodumene pegmatite deposit, located in northwestern Quebec, Canada. The pegmatites outcrop as irregular surface dikes, interlayered with biotite schist and greenstone inclusions. Spodumene is the most prevalent lithium-bearing mineral. Other minerals identified by Lithium One include typical pegmatite minerals, perthitic feldspar, quartz, and muscovite, along with minor or trace amounts of apatite, beryl, iron oxides, ilmenite, and serpentine.

At the request of Lithium One, Hazen Research, Inc. initiated a laboratory program to investigate the recovery of lithium as lithium carbonate (Li_2CO_3) from ore samples from the James Bay pegmatites. Commercial processes for recovering lithium from spodumene deposits similar to the James Bay pegmatites involve a roast–acid leach, such as depicted in the flowsheet shown in Figure 1. For this process, the ore is upgraded by physical beneficiation using flotation and perhaps other technologies. The beneficiated ore is then calcined at 1,050–1,100°C to convert the naturally occurring α-spodumene to β-spodumene, which is soluble in sulfuric acid (H_2SO_4). An acid bake is performed at 250°C to attack the β-spodumene, rendering the lithium soluble in the subsequent water leach step. Iron and aluminum solubilities are minimized in the leach step by controlling the pH at 6–6.5 with lime. Along with lithium, some impurities, such as magnesium and calcium, are also solubilized. A series of purification steps removes calcium, magnesium, and other impurities from solution. Sodium carbonate (Na_2CO_3) is then added to precipitate lithium carbonate. Depending on the desired grade of the lithium carbonate product, further purification steps may be required.

On January 11, 2010, approximately 2,700 kg of minus 6.3-mm assay rejects and sawed core halves from 11 dikes were received. Selected samples were characterized by:

- Chemical analysis
- X-ray fluorescence (XRF) whole-rock analysis
- Specific gravity
- Optical microscopy
- X-ray diffraction (XRD) analysis
- Comminution tests, including semiautogeneous (SAG) mill comminution (SMC), Bond ball mill work index (BW_i), and Bond abrasion work index (A_i) testing

Composites were generated from these samples and used for the process development work completed during this project. Additionally, a box of 48 analytical pulps for quality control lithium assay work was received on March 23, 2010.

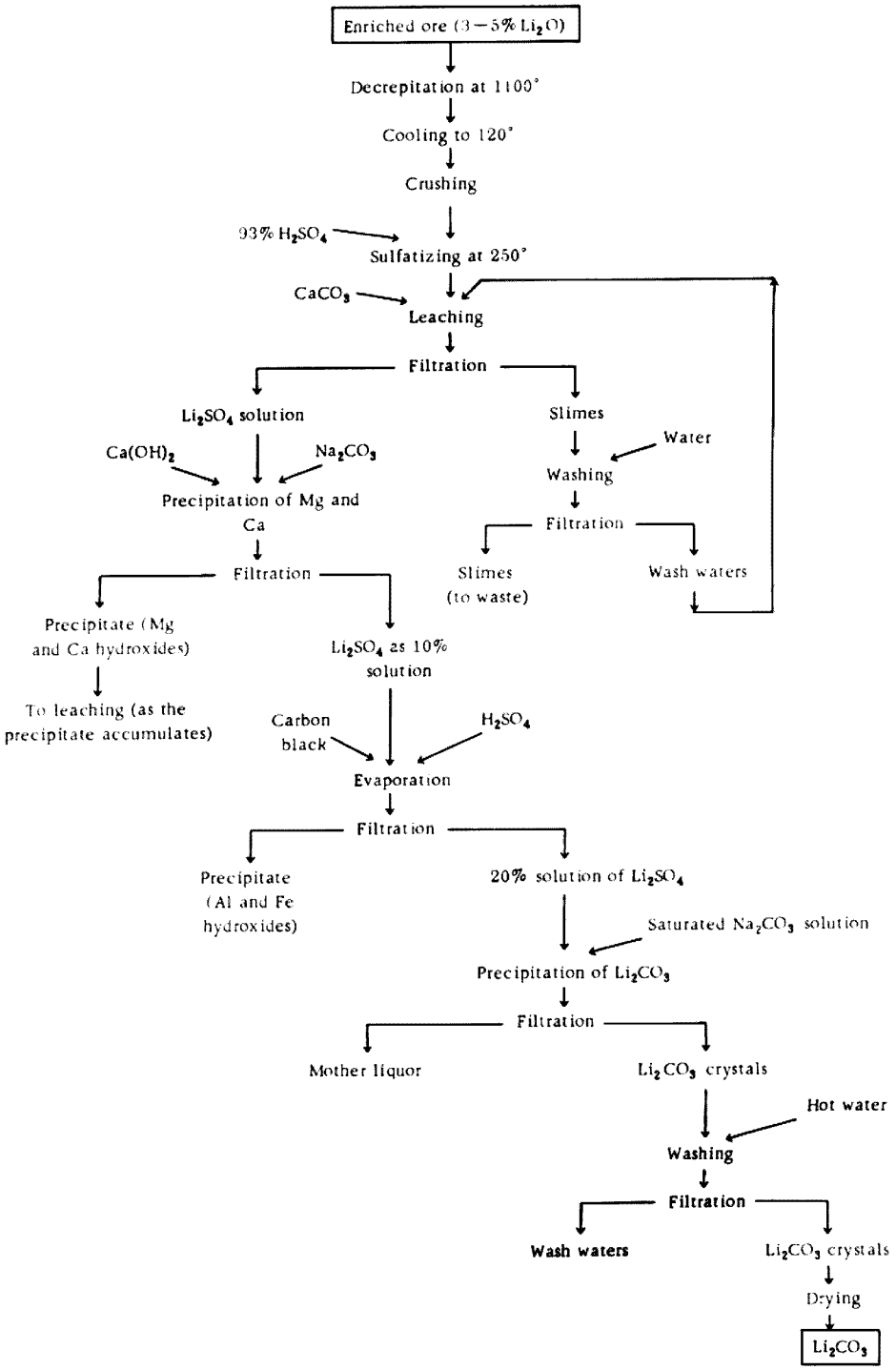


Figure 1. Flowsheet for Processing Spodumene by the Roast–Acid Leach Method[1]

[1]Reprinted courtesy of Zelikman, A. N., Krein, O. E., and Samsonov, G. V. 1966. *Metallurgy of Rare Metals.* Moscow: Izdatel'stvo Metallurgiya.

A series of batch flotation experiments to generate a spodumene concentrate was performed to investigate grind size and reagent additions. The flotation feed was ground to 80% passing 300 μm with 1.35 kg/t sodium hydroxide (NaOH) and deslimed using a hydrocyclone; the cyclone underflow was wet-screened at 150 μm. Both the coarse and fine fractions were subjected to flotation using Sylfat FA-1 (tall oil fatty acid) collector at a dose of 0.75 kg/t with two stages of cleaning. For the coarse fraction float, the second cleaner concentrate contained an estimated 80–85% spodumene and the rougher tails contained less than 1% spodumene. For the fine fraction float, the second cleaner concentrate contained an estimated 90% spodumene and the rougher tails contained less than 1% spodumene, indicating most of the fine spodumene was recovered by the split flotation procedure. Combining the coarse and fine concentrates recovered 80% of the lithium in the flotation feed. A bulk flotation concentrate was generated for the calcining experiments.

Batch rotary kiln experiments were performed with the spodumene-rich flotation concentrate to convert the naturally occurring α-spodumene to β-spodumene, which is amenable to leaching with sulfuric acid. Temperatures of 950, 1,000, and 1,050°C at a 60-min roasting time were investigated. Based on XRD and differential thermal analysis (DTA), 1,050°C at 60 min converted nearly all of the α-spodumene to β-spodumene. Calcining for 30 min at 1,050°C was insufficient for complete conversion. The calcines from these roasting experiments were blended for use in the leaching, purification, and lithium carbonate recovery experiments.

The amount of sulfuric acid used and intimate mixing of the acid with the calcine during the acid bake were found to be important variables for solubilizing lithium from the calcine during subsequent leaching. Six acid bake and leach experiments were performed to investigate these variables. These experiments showed that 478 kg/t acid, intimately mixed with the calcine before the acid bake, was required to obtain a higher lithium extraction. Using calcine without any further grinding achieved 91% lithium extraction from the calcine, while grinding the calcine to 80% passing 28 μm achieved extraction of 92%, indicating that fine grinding may not be necessary.

The lithium extractions from these six experiments plateaued at 92% from the calcine, suggesting that one or more refractory lithium minerals were preventing higher extraction. A mineralogical examination of the leach residues identified the presence of a lithium–aluminum silicate not named in the XRD database. However, it is not known if this lithium species is present in the original ore or is formed during the calcining or acid bake step. One thought was that this species formed during the leach, with lithium coprecipitating with aluminum when lime was used to maintain the pH at 6.0–6.5. A leaching experiment was completed to determine if lithium extractions were improved when the leach was conducted in an acid environment without the addition of lime. However, the lithium extraction was unchanged, indicating that coprecipitation with aluminum during the leach was not responsible for limiting the lithium extraction at 92%.

The leach liquor was purified in a series of steps to remove magnesium, calcium, and other impurities. Magnesium was removed by increasing the pH to 11.5 at room temperature with lime to precipitate magnesium hydroxide ($Mg(OH)_2$). Calcium leached from the ore and solubilized in the magnesium

removal step was then removed with the addition of a measured amount of sodium carbonate to precipitate calcium carbonate ($CaCO_3$). Only a very small amount of lithium was lost in these steps. Lithium was precipitated from the purified liquor as lithium carbonate by the addition of more sodium carbonate. The resulting slurry was centrifuged, and the solids were rinsed with a small amount of hot deionized (DI) water and dried. Approximately 66% of the lithium in the leach liquor was recovered as lithium carbonate product, because lithium carbonate is partially soluble in the liquor after precipitation. In a commercial operation, additional lithium is recovered by recycling this liquor upstream in the process.

The final lithium carbonate product was assayed with the results shown in Table 2. Of note are the sodium, sulfur, and calcium impurity levels. It is expected that additional washing of the lithium carbonate would reduce the sodium and sulfur in the final product. However, this will solubilize some of the lithium and will require that the wash liquors be recycled upstream in the process to recover this lost lithium. Additional calcium could be removed from the final product using more sodium carbonate in the calcium removal step or by using a two-step lithium carbonate recovery, in which the calcium would mostly report to the first-stage lithium carbonate. This less pure lithium carbonate could be reprocessed in the plant to achieve higher overall lithium recovery.

Table 2. Analysis of Lithium Carbonate Product

Element	Assay, %
Li	17.1
CO_3^{2-}	72.1
Na	0.650
S	0.43
Mg	0.002
K	0.008
Ca	0.117
Fe	0.006
Al	0.020
Mn	0.001

Based on the work completed during this program, the preliminary process flowsheet, shown in Figure 2, was developed. For this flowsheet, the ore is crushed and ground to 80% passing 300 µm with 1.35 kg/t NaOH and deslimed using a hydrocyclone; the cyclone underflow is then wet-screened at 150 µm. Both fractions are floated separately using Sylfat FA-1 (tall oil fatty acid) collector at 0.75 kg/t in one rougher stage and two cleaning stages. The concentrates from both circuits are combined and calcined at 1,050°C for 1 h to convert the naturally occurring α-spodumene in the concentrate to acid-soluble β-spodumene. Acid is added to the calcine and baked at 250°C for 1 h to decompose the β-spodumene into lithium sulfate, which is subsequently dissolved in the leaching step. Lime is added to the leach to maintain the pH at 6.0–6.5, which suppresses iron and aluminum solubilities. The leach is performed at 60°C for 30

min. After solid–liquid separation, magnesium is removed from solution by adding hydrated lime to precipitate magnesium hydroxide. The liquor from the subsequent solid–liquid separation step is concentrated by evaporation, and the calcium is removed by adding sodium carbonate to precipitate calcium carbonate. The resulting liquor is heated to 90–93°C, and lithium carbonate is precipitated from solution. Because lithium carbonate is partially soluble, only 66% of the lithium is recovered as product from the purified liquor. The resulting mother liquor, containing significant lithium, is recycled back to the lithium recovery step to reclaim additional lithium. A mother liquor bleed is necessary to control impurity buildup.

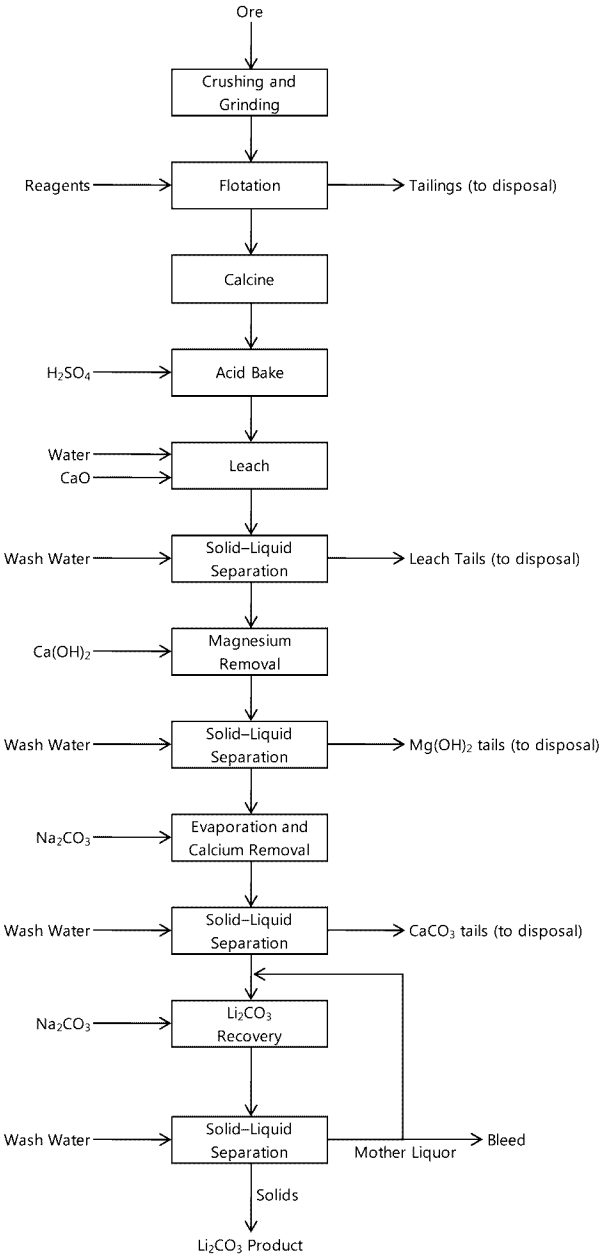


Figure 2. Preliminary Flowsheet for the Recovery of Li_2CO_3 from James Bay Spodumene

Based on the results of the laboratory program, lithium recoveries from each step are presented in Table 3. This table suggests several areas for additional laboratory work to improve lithium recoveries in the process. Better recoveries in the flotation circuit may be possible with changes in the operating conditions. Investigating methods for extracting lithium from the lithium–aluminum silicate in the leach residues would improve recovery in this step. Recycling the lithium-bearing mother liquor back to the process needs to be evaluated for improving lithium recovery in the process.

Table 3. Lithium Recoveries based on Laboratory Experiments

Process Step	Li Recovery in Each Step, %
Desliming	87–93
Flotation	80
Acid Bake and Leaching	92
Purification	99.7
Li_2CO_3 Product Recovery	66[a]

[a]This lithium carbonate recovery is from a single-pass precipitation. In a commercial operation, the lithium-bearing liquor from this step is recycled back to the process to recover most of the lithium.

Other potential improvements to the flowsheet include evaluating the feasibility of optical sorting to generate a spodumene-rich concentrate for flotation and investigating additional purification steps if reduced impurities are desired in the lithium carbonate product. The results from this current program and the additional laboratory work would be used to design and operate a pilot plant to demonstrate the process on a continuous basis and to provide engineering data to scale the process to commercial size.

SAMPLE CHARACTERIZATION AND PREPARATION

On January 11, 2010, approximately 2,700 kg of minus 6.3-mm assay rejects and sawed core halves from 11 dikes were received. A summary of these samples is provided in Table 4. Detailed sample descriptions are provided in Appendix A.

Table 4. Sample Identification

HRI	Sample Type	Client ID (Dike No.)	Sample Weight, kg
52346-1	Assay Rejects	7.2	112.4
52346-2		7.6	137.8
52346-3		8.3	107.3
52346-4		8.7	138.2
52346-5		9.2	91.4
52346-6		10.4	99.1
52346-7		11.2/11.4	138.4
52346-8		12.2	113.3
52346-9		13.2	86.8
52346-10		14.2	127.1
52346-11		15.1	130.1
Total			**1,281.9**
52346-1	Split Core	7.2	120.1
52346-2		7.6	141.9
52346-3		8.3	122.4
52346-4		8.7	155.1
52346-5		9.2	106.5
52346-6		10.4	108.0
52346-7		11.2/11.4	163.8
52346-8		12.2	126.3
52346-9		13.2	120.8
52346-10		14.2	143.8
52346-11		15.1	141.1
Total			**1,449.8**

Upon receipt, all samples were inventoried for each of the 11 dike samples.

ASSAY REJECT SAMPLES

The assay reject samples were composited by dike, then stage-crushed to minus 1.7 mm. During the screening and stage-crushing, some of the coarse mica was screened out and saved as a plus 1.7-mm fraction to facilitate the crushing of the remainder of the oversize material. The plus 1.7-mm fraction, which is about 1% of the total weight, was assayed for lithium.

Three assay pulps were prepared for each composite sample. One of the pulps was assayed for lithium, phosphorus, rubidium, and beryllium. Another pulp was subjected to whole-rock analysis by wavelength dispersive XRF, and the third pulp was subjected to XRD. Head assay and XRF results are shown in Tables 5 and 6, respectively. The XRD results are included in the mineralogy section of the report. The initial lithium analytical procedure used by Hazen was shown to not solubilize all of the lithium in the sample. Hence the dissolution procedure was modified and the samples reanalyzed. Both analyses are listed in Table 5.

On March 23, 2010, Hazen received a box of 48 analytical pulps for quality control lithium assay work. These samples, identified as 2851–2898, were previously analyzed at COREM. Assay results are shown in Table 7.

Initially, 1-kg splits of minus 1.7-mm material from Dike 11.2/11.4 (HRI 52346-7) were used for the flotation work because it was one of the larger samples. Later, four composites were made following a compositing scheme prepared by the client. It was agreed to prepare equal-weight composites of 30 kg from each individual dike. This preserved the majority of the assay reject material as-is, in the event different compositing ratios might be required in future work. Weighted composite preparation was not warranted as the weights of the individual dike samples do not necessarily reflect actual dike dimensions. The computed total composite weights are shown in Table 8. Composite head assay results are shown in Table 9. A master composite sample was made by combining 5 kg from each of the four composites. All the remaining flotation work used composites prepared from the assay reject samples.

Table 5. Head Assays

Sample ID	Client ID Dike	Weight		Assay, %						
		g	%	Li	Li (1st repeat)	Li (2nd repeat)	P	Rb	Rb (1st repeat)	Be
52346-1										
Plus 1.7 mm	7.2	848	0.8	1.80						
Minus 1.7 mm		111,582	99.2	0.66	0.81	0.87	0.076	0.061	0.068	0.023
Total		**112,430**	**100.0**							
52346-2										
Plus 1.7 mm	7.6	1,433	1.0	1.96						
Minus 1.7 mm		136,367	99.0	0.59	0.74	0.82	0.114	0.077	0.075	0.018
Total		**137,800**	**100.0**							
52346-3										
Plus 1.7 mm	8.3	1,959	1.8	1.98						
Minus 1.7 mm		105,331	98.2	0.51	0.69	0.78	0.122	0.080	0.081	0.021
Total		**107,290**	**100.0**							
52346-4										
Plus 1.7 mm	8.7	1,082	0.8	1.84						
Minus 1.7 mm		137,128	99.2	0.65	0.79	0.84	0.112	0.084	0.085	0.016
Total		**138,210**	**100.0**							
52346-5										
Plus 1.7 mm	9.2	1,644	1.8	1.72						
Minus 1.7 mm		89,736	98.2	0.45	0.68	0.74	0.123	0.087	0.085	0.016
Total		**91,380**	**100.0**							
52346-6										
Plus 1.7 mm	10.4	1,644	1.7	1.57						
Minus 1.7 mm		97,496	98.3	0.52	0.70	0.74	0.110	0.088	0.085	0.018
Total		**99,140**	**100.0**							
52346-7										
Plus 1.7 mm	11.2	1,573	1.1	1.75						
Minus 1.7 mm		136,837	98.9	0.56	0.79	0.83	0.164	0.079	0.072	0.013
Total		**138,410**	**100.0**							
52346-8										
Plus 1.7 mm	12.2	666	0.6	1.21						
Minus 1.7 mm		112,584	99.4	0.54	0.79	0.84	0.107	0.098	0.090	0.012
Total		**113,250**	**100.0**							
52346-9										
Plus 1.7 mm	13.2	2,082	2.4	1.83						
Minus 1.7 mm		84,718	97.6	0.53	0.71	0.80	0.119	0.089	0.084	0.012
Total		**86,800**	**100.0**							
52346-10										
Plus 1.7 mm	14.2	1,708	1.3	1.97						
Minus 1.7 mm		125,342	98.7	0.56	0.72	0.78	0.140	0.089	0.095	0.012
Total		**127,050**	**100.0**							
52346-11										
Plus 1.7 mm	15.1	2,307	1.8	0.79						
Minus 1.7 mm		127,813	98.2	0.51	0.71	0.81	0.257	0.084	0.091	0.021
Total		**130,120**	**100.0**							

Table 6. Results of XRF Analysis

Constituent	52346-1	52346-2	52346-3	52346-4	52346-5	52346-6	52346-7	52346-8	52346-9	52346-10	52346-11
	Dike 7.2	Dike 7.6	Dike 8.3	Dike 8.7	Dike 9.2	Dike 10.4	Dike 11.2	Dike 12.2	Dike 13.2	Dike 14.2	Dike 15.1
Analyte, wt%											
Na_2O	3.94	4.19	4.67	4.22	4.29	4.85	4.48	4.65	4.65	4.72	4.49
MgO	0.07	<0.05	<0.05	<0.05	0.15	0.06	<0.05	<0.05	0.09	0.25	<0.05
Al_2O_3	15.7	15.6	15.7	15.7	15.8	16.2	16	15.9	15.4	15.7	15.5
SiO_2	77.2	77.1	76.8	77	76.5	75.8	76.6	76.9	77.1	76.5	76.5
P_2O_5	0.28	0.46	0.46	0.42	0.45	0.45	0.57	0.41	0.45	0.46	0.83
S	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Cl	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
K_2O	2.70	2.71	2.47	2.71	3.00	2.78	2.56	2.34	2.42	2.25	2.37
CaO	0.31	0.32	0.28	0.24	0.29	0.28	0.27	0.22	0.27	0.37	0.25
TiO_2	0.01	<0.01	<0.01	0.01	0.01	<0.01	0.01	<0.01	0.02	0.03	0.01
MnO_2	0.07	0.09	0.07	0.07	0.07	0.08	0.08	0.07	0.09	0.10	0.09
Fe_2O_3	0.47	0.46	0.35	0.40	0.42	0.37	0.41	0.29	0.47	0.60	0.37
BaO	0.02	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Element, ppm											
V	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10
Cr	<10	<10	<10	<10	<10	<10	<10	<10	<10	23	<10
Ni	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10
Cu	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10
Zn	26	62	32	35	31	26	29	73	35	45	52
As	91	90	30	<20	33	<20	<20	<20	77	50	33
Sn	<50	<50	<50	<50	<50	<50	<50	<50	<50	<50	<50
Pb	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10
Mo	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10
Sr	113	79	59	79	74	63	71	37	55	52	42
U	<20	<20	<20	<20	<20	<20	<20	<20	<20	<20	<20
Th	<20	<20	<20	<20	<20	<20	<20	<20	<20	<20	<20
Nb	74	45	58	48	49	73	44	78	55	63	70
Zr	<10	<10	<10	<10	<10	<10	<10	14	10	11	<10
Rb	495	590	622	634	703	699	649	755	691	706	673
Y	<10	<10	<10	<10	<10	<10	<10	10	<10	<10	<10

Table 7. Lithium Assays of COREM Samples Received 3/23/10

Client ID	Li Assay, %
2851	0.74
2852	0.89
2853	0.90
2854	0.001
2855	0.68
2856	0.53
2857	0.66
2858	1.14
2859	0.36
2860	0.001
2861	0.56
2862	0.56
2863	0.85
2864	0.42
2865	1.32
2866	0.10
2867	0.72
2868	1.17
2869	0.93
2870	0.001
2871	0.85
2872	0.07
2873	0.68
2874	0.68
2875	0.79
2876	1.81
2877	0.41
2878	0.34
2879	0.54
2880	0.002
2881	0.63
2882	0.64
2883	1.17
2884	0.66
2885	0.86
2886	2.25
2887	0.43
2888	0.70
2889	0.92
2890	0.002
2891	0.97
2892	0.99
2893	0.66
2894	0.42
2895	1.07
2896	0.43
2897	0.36
2898	1.40

Table 8. Composite Weights

Composite	HRI	Client ID, Dike	Weight, kg		
			Current	Composite Sample	Remaining
1	52346-1	7.2	107	30	77
	52346-2	7.6	132	30	102
	52346-3	8.3	101	30	71
	52346-4	8.7	132	30	102
	Total		**472**	**120**	**352**
2	52346-5	9.2	88	30	58
	52346-6	10.4	95	30	65
	52346-7	11.2	108	30	78
	Total		**290**	**90**	**200**
3	52346-8	12.2	109	30	79
	52346-9	13.2	82	30	52
	52346-10	14.2	122	30	92
	Total		**313**	**90**	**223**
4	52346-11	15.1	123	60	63
	Total		**123**	**60**	**63**

Table 9. Head Assays of Composites

Sample	Weight, kg	Li Assay, %
Composite 1	120	0.78
Composite 2	90	0.68
Composite 3	90	0.71
Composite 4	60	0.74

SAWED CORE SAMPLES

Fourteen core interval samples were selected by the client for specific gravity determination so that Lithium One could correlate the measured specific gravity with the calculated average grade of the samples. Three pieces of core from each of the selected intervals were hand-picked based on their spodumene content (low, medium, and high). Each piece was documented by photograph and sent to Advanced Terra Testing, Inc. (ATT) for specific gravity determination according to ASTM Procedure C97. A summary of the specific gravity determinations is shown in Table 10. Photographs of the selected core pieces and tests results from ATT are shown in Appendix B.

Table 10. Summary of Specific Gravity Determinations

HRI	Dike	Interval Received	Weight, g	Calculated Grade,[a] % Li	sg	Average sg
52461-1	7.2	753025–753027	1,161.8	2.07	2.75 2.76 2.78	2.76
		753085–753087	1,025.2	1.79	2.85 2.83 2.80	2.83
		753407–753409	1,107.9	1.63	2.70 2.75 2.79	2.75
		753366–753369	1,183.9	2.24	2.70 2.72 2.85	2.76
52461-2	7.6	753534–753536	967.3	1.95	2.83 2.87 2.73	2.81
52461-3	8.3	279562–279565	1,210.4	1.55	2.86 2.70 2.74	2.77
		105–107	977.1	1.44	2.68 2.70 2.63	2.67
52461-4	8.7	658–659	965.6	2.19	2.71 2.79 2.79	2.76
		234–237	1,090.4	1.33	2.76 2.68 2.79	2.74
52461-5	9.2	354–357	998.9	1.25	2.69 2.67 2.60	2.65
		399–402	975.0	1.00	2.66 2.74 2.70	2.70
52461-6	10.4	1545–1546	853.8	2.46	2.63 3.02 2.76	2.80
52461-7	11.2 (11.4 included)	1610–1613	1,161.9	1.15	2.78 2.75 2.64	2.72
52461-9	13.2	2226–2228	964.2	1.83	2.74 2.72 2.79	2.75

[a]Calculated grade values provided by Lithium One

The split core intervals were composited by dike for comminution work. Portions from each dike were saved for possible ore sorting work and verification flotation work. A flow diagram for the sample preparation of the split core samples is shown in Figure 3. The computed total composite weights are shown in Table 11. Selected splits were used for the comminution work shown in Figure 3. The remaining composites were held for future work.

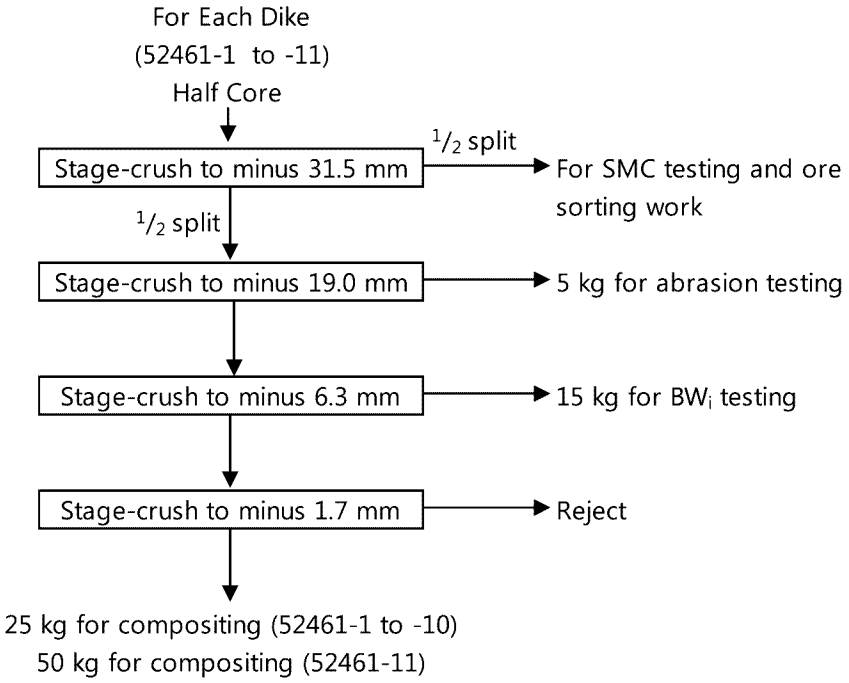


Figure 3. Flow Diagram for Core Sample Preparation

Table 11. Summary of Composite Weights for the Core Samples

Composite	HRI	Client ID, Dike	Weight, kg		
			Current	Recommended Composite Sample	Remaining
1	52461-1	7.2	120.1	25	95.1
	52461-2	7.6	141.9	25	116.9
	52461-3	8.3	122.4	25	97.4
	52461-4	8.7	155.1	25	130.1
	Total		**539.5**	**100**	**439.5**
2	52461-5	9.2	106.5	25	81.5
	52461-6	10.4	108.0	25	83.0
	52461-7	11.2	163.8	25	138.8
	Total		**378.3**	**75**	**303.3**
3	52461-8	12.2	126.3	25	101.3
	52461-9	13.2	120.8	25	95.8
	52461-10	14.2	143.8	25	118.8
	Total		**390.9**	**75**	**315.9**
4	52461-11	15.1	141.1	50	91.1
	Total		**141.1**	**50**	**91.1**

GRINDING AND DESLIMING PROCEDURES AND RESULTS

Selected 1-kg test charges of minus 1.7-mm assay rejects were ground in a laboratory 9- by 10-in. mild steel rod mill at 50% solids for 5 min to achieve a target grind of 80% passing 300 µm. The rod charge distribution is shown in Table 12. After grinding, the sample was deslimed using a FL Smidth–Krebs gMax 1U-3125 hydrocyclone with a 4-mm apex. The hydrocyclone underflow was subjected to a second desliming using the same hydrocyclone to ensure complete removal of slimes prior to flotation.

Table 12. Sepor 9- by 10-in. Rod Mill Charge

Rod Diameter, mm	No. of Rods	Weight, g
31.5	1	2,057.0
16.0	2	1,247.3
19.0	6	2,941.0
14.3	9	3,209.4
12.5	9	1,887.6
9.5	5	940.0
6.3	5	659.0
Total	**37**	**12,941.3**

A particle size distribution (PSD) was determined by wet-screening the hydrocyclone underflow at 38 µm and dry-screening the plus 38-µm material at 850, 600, 425, 300, 212, 150, 75, 45, and 38 µm. The hydrocyclone overflow was wet-screened at 10 µm. A flowsheet of the PSD procedure is shown in Figure 4.

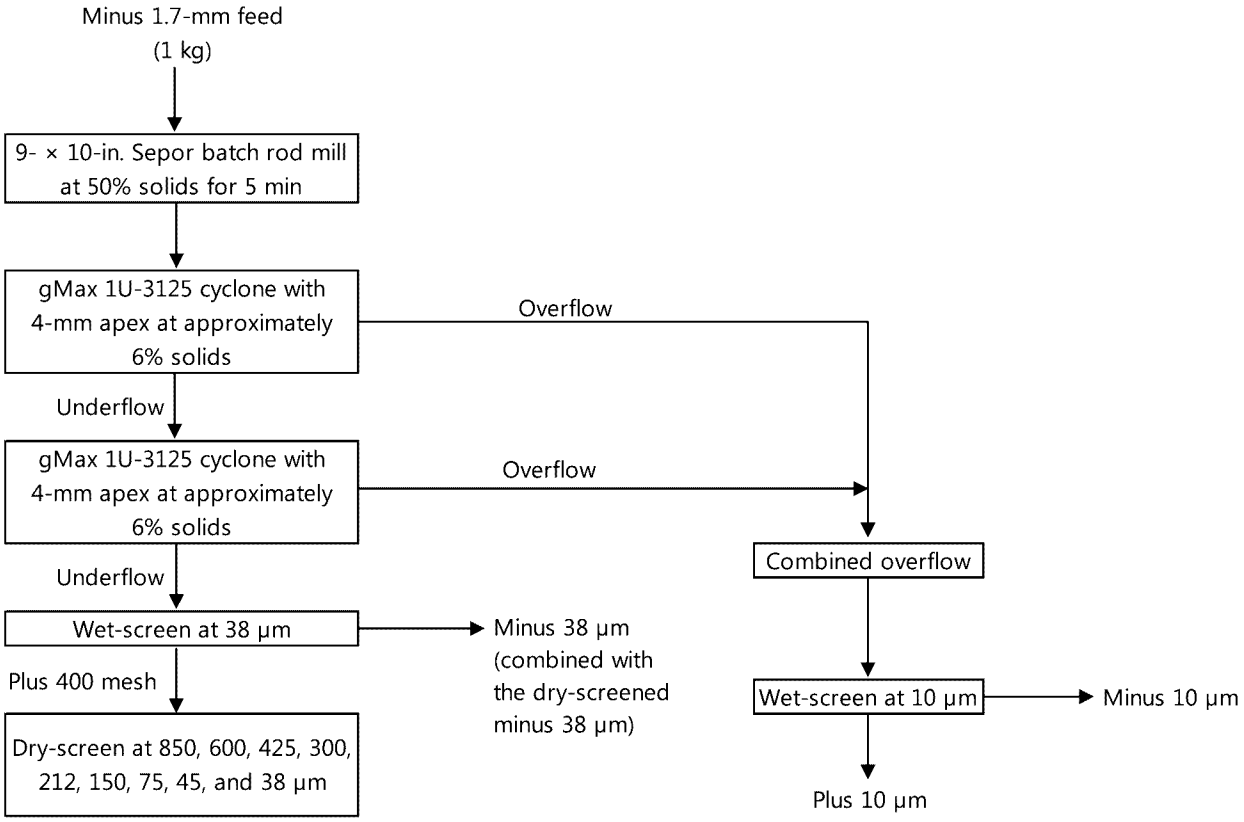
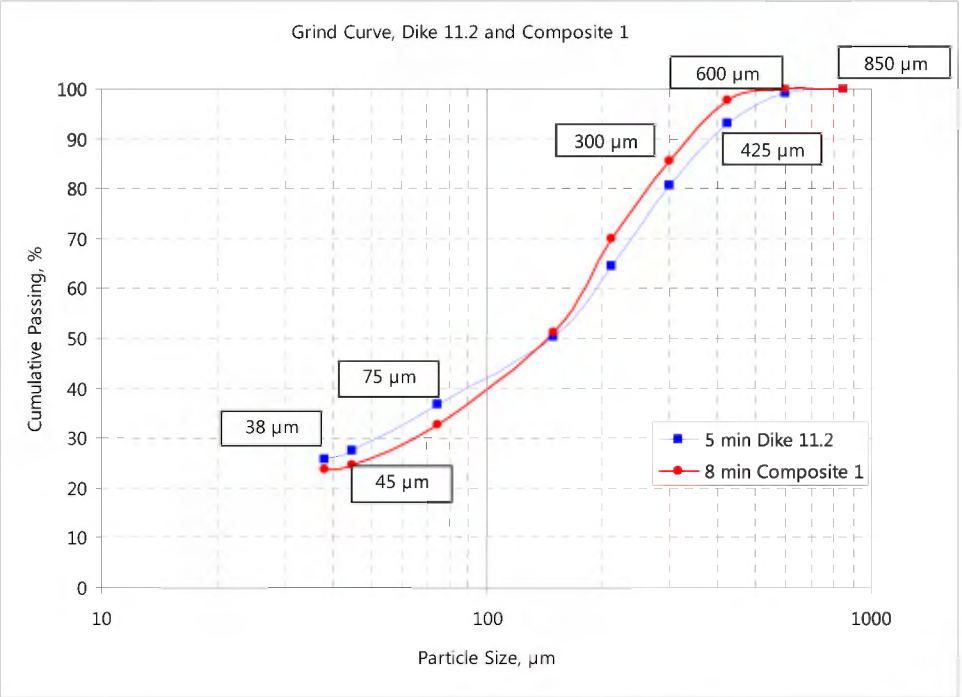


Figure 4. Flowsheet of the PSD Procedure

After two flotation experiments, the grind time was increased to 8 min because of plugging of the hydrocyclone and sanding out in the pumps and lines from coarse mica and spodumene. Grind curves and the PSD for the 5- and 8-min grinds are shown in Figure 5.

The amounts passing 300 µm for the 5- and 8-min grinds were 81 and 86%, respectively.



Product	Dike 11.2[a]		Composite 1[a]	
	5-min Grind		8-min Grind	
	Weight			
	g	%	g	%
Plus 38 µm dry-screened				
Plus 850 µm	0.8	0.1	0.0	0.0
Plus 600 µm	8.4	0.9	1.0	0.1
Plus 425 µm	55.5	6.0	19.7	2.1
Plus 300 µm	115.2	12.5	115.3	12.3
Plus 212 µm	147.3	16.0	145.2	15.5
Plus 150 µm	129.1	14.0	176.6	18.8
Plus 75 µm	126.6	13.8	173.9	18.5
Plus 45 µm	85.4	9.3	74.9	8.0
Plus 38 µm	14.4	1.6	9.7	1.0
Minus 38 µm	63.2	6.9	97.6	10.4
Total	745.9	81.1	813.9	86.8
Cyclone overflow wet-screened				
Plus 10 µm	80.4	8.7	34.4	3.7
Minus 10 µm	94.0	10.2	89.6	9.6
Total	174.3	18.9	124.0	13.2
Calculated feed	920.2	100.0	937.9	100.0

[a]Assay reject samples

Figure 5. Grind Curves and Weight Distributions for Cyclone Underflow and Overflow Products for 5- and 8-min Grinds

MINERALOGY

In the course of the process development studies, XRD analyses were completed on representative head splits of the 11 dike samples to determine whether significant mineralogical differences exist between samples that could affect lithium extraction. A sample (minus 1.7-mm split) from Dike 11.2, which is one of the larger dikes and was also used for the initial flotation work, was examined in somewhat greater detail. This included a heavy-liquid separation to produce a spodumene concentrate in order to establish the lithium content of essentially pure spodumene as a baseline measure of the maximum attainable lithium content for guidance during beneficiation testing. Prior to proceeding with sample preparation and metallurgical testing, specimens of core (received as cut drill core halves) were wetted down and inspected, which revealed highly variable amounts of mostly rather coarse-grained spodumene (mostly greenish colored) intergrown with feldspar (mostly whitish, occasionally dark gray to almost black due to fine black pigment), quartz (smokey colored), and minor micaceous minerals (brownish sheath-like aggregates). Small amounts of lithiophilite and ferri-sicklerite, along with tourmaline and blue apatite, were also found.

Several intervals were photographed, and selected typical specimens were submitted to ATT for specific gravity measurements. Some of the photographs in Appendix B, exhibiting a range in textures and variation in spodumene abundance, were sent to Terra Vision for their evaluation of optical sorting potential. Based on these photographs, Terra Vision prepared a proposal to test the feasibility of optical sorting in their laboratory. This may be pursued in the future.

XRD Analysis of Dike Samples

The results of the XRD analyses of the different dike samples show the same major mineral composition for all the samples consisting of major quartz and albite with subordinate amounts of microcline, minor spodumene, and minor muscovite.

Although the XRD analyses show only relatively small peaks for spodumene (32° 2-theta is larger than the major peak at about 30.5° 2-theta, presumably due to preferred orientation), the macroscopic core examination showed a number of intervals where spodumene is a major component. The individual XRD patterns are compiled in Appendix C.

Mineralogical Examination of Dike 11.2 (Minus 1.7 mm)

The mineralogical investigation of the Dike 11.2 sample consisted of gravity separation with heavy liquid, microscopic examination, XRD analysis, and chemical assays of the minus 1.7-mm head sample and the gravity separation products.

For the gravity separation, a split of the minus 1.7-mm head sample was wet-screened at 25 μm. The plus 25-μm fraction was separated with heavy liquid using bromoform at a specific gravity of 2.84 to concentrate the spodumene.

The sink, float, and unseparated minus 25-µm slimes were assayed for lithium. Table 13 gives the weight distribution and assay results, which show a lithium content of 3.66% in the gravity concentrate. This is equal to approximately 7.9% Li_2O, which is the upper limit for lithium content in published analyses.[2] The float product assayed 0.11% Li, which may be mostly due to lithium in the muscovite.

Table 13. Weight Distribution and Assay Results from Gravity Separation

Product	Weight		Li Assay,	Li Distribution,
	g	%	%	%
1.7-mm by 25-µm Sink	93.9	19.5	3.66	82.7
1.7-mm by 25-µm Float	316.4	65.8	0.11	8.0
Minus 25-µm Slimes	70.5	14.7	0.55	9.3
Calculated Head	480.7	100.0	0.86	100.0
Assayed Head			0.84	

Optical examination of the minus 1.7-mm head showed chiefly spodumene, quartz, feldspar, and mica. The spodumene is frequently intergrown with the quartz and the feldspar.

In the description of the gravity-separation products and the captions of the photomicrographs, both quartz and feldspar are referred to as gangue. The examination of the gravity-separation products showed that the majority of the sink (concentrate) consists of discrete, coarse spodumene and minor amounts of mica and gangue. Sometimes the spodumene occurs intergrown with the gangue. The float (tails) consists mainly of coarse gangue and mica, with minor to trace amounts of spodumene, which is sometimes intergrown with the gangue. The minus 25-µm slimes are composed of abundant gangue and liberated spodumene particles. Figures 6–13 show some photomicrographs.

The results of the XRD analysis of the plus 25-µm heavy-liquid separation products and the minus 25-µm slimes are shown in Table 14. Figures 14–17 show the XRD patterns.

[2]Anthony, J. W., Bideaux, R. A., Bladh, K. W, and Nichols, M. C. 1995. Silica, Silicates Part 2. Volume II of *Handbook of mineralogy*, 747. Tucson, AZ: Mineral Data Publishing.

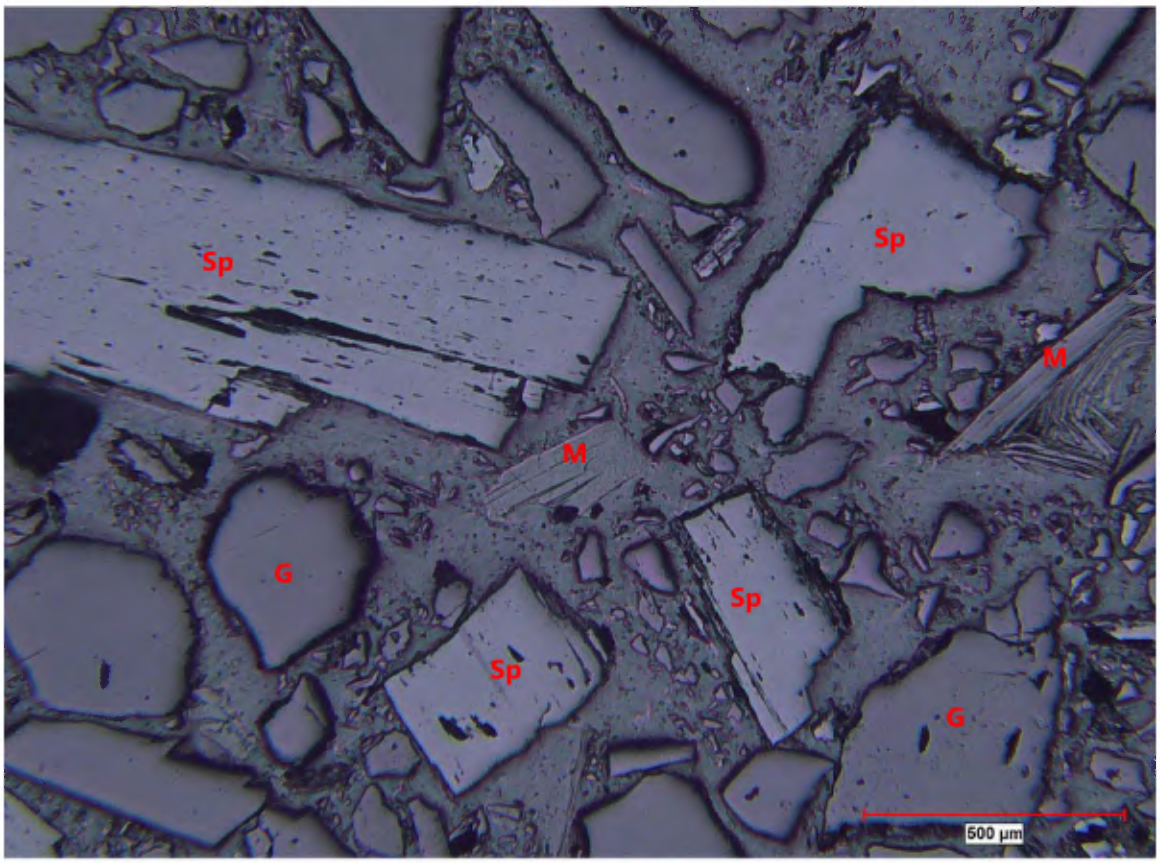


Figure 6. Dike 11.2, Minus 1.7-mm Head, Example 1

Photomicrograph showing discrete spodumene (Sp), gangue (G), and mica (M).

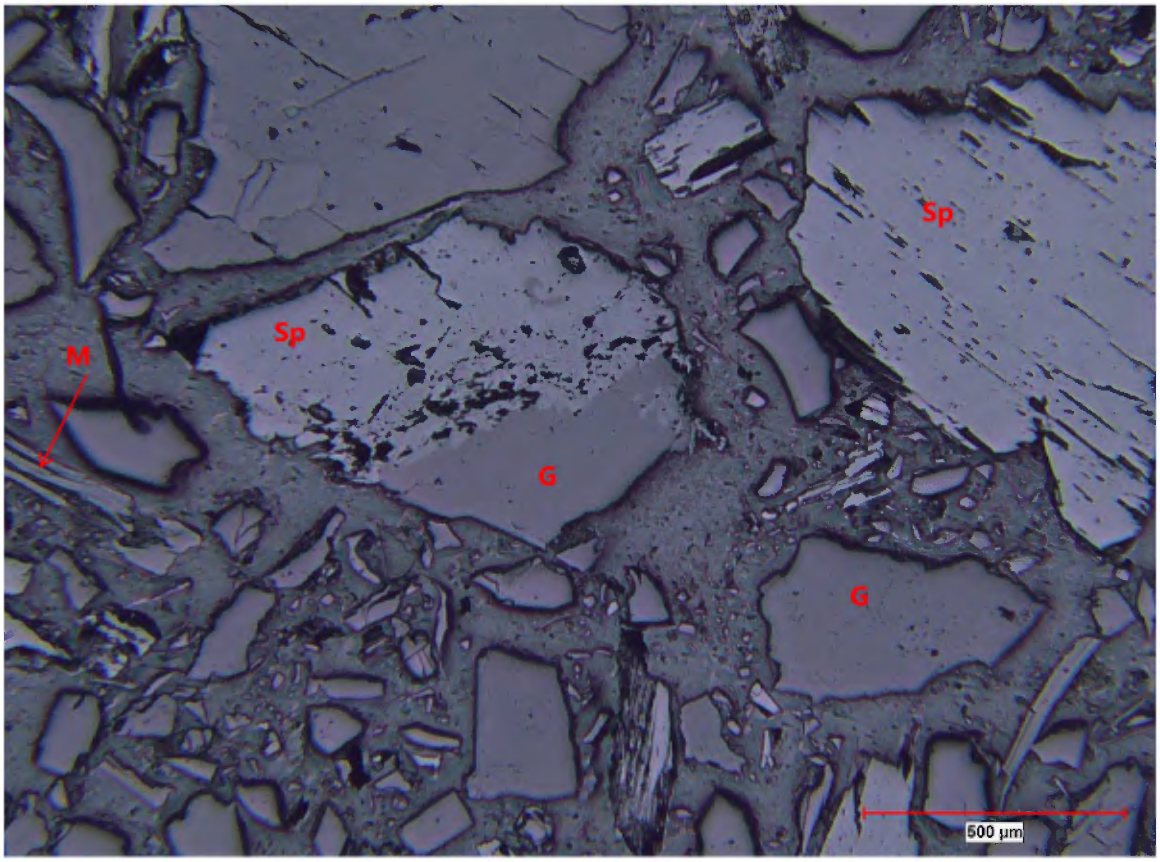


Figure 7. Dike 11.2, Minus 1.7-mm Head, Example 2

Photomicrograph showing an intergrowth of spodumene (Sp) and gangue (G). Discrete spodumene, gangue, and mica (M) are also shown.

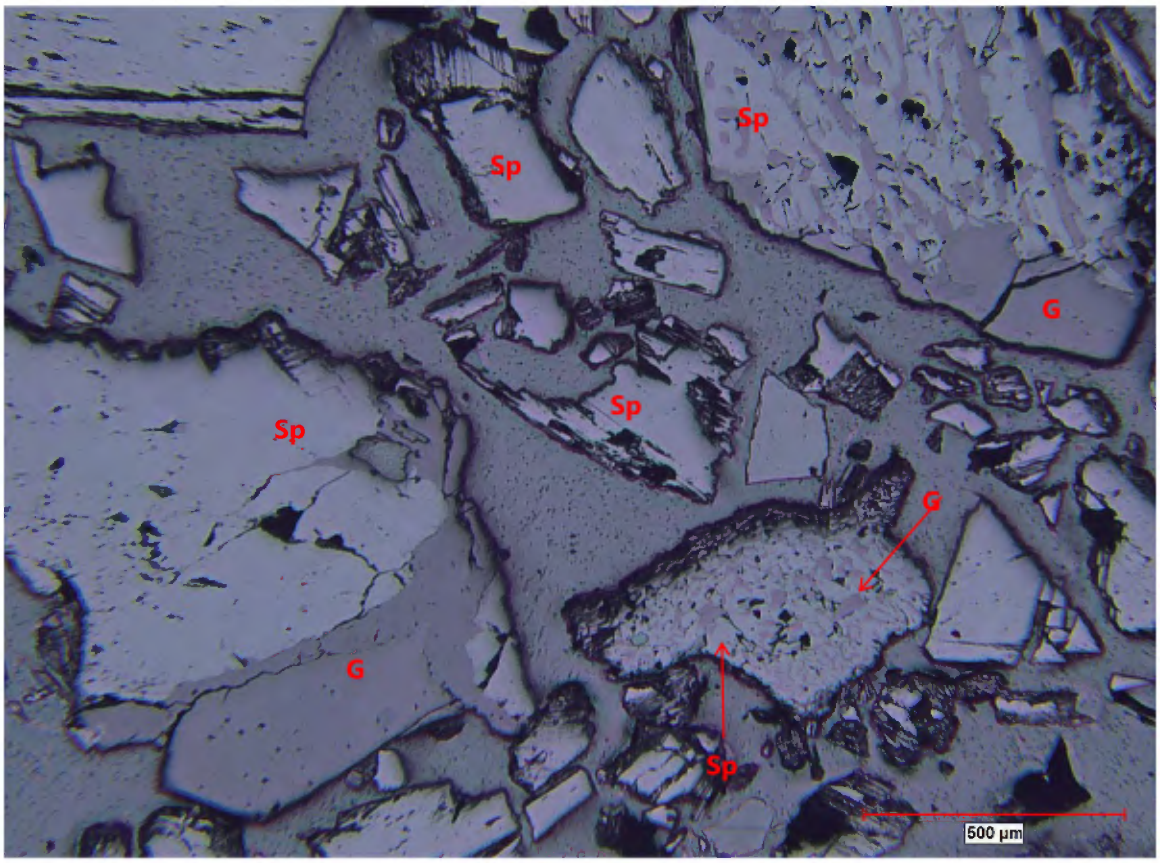


Figure 8. Dike 11.2, Minus 1.7-mm by 25-µm Sink Product, Example 1

Photomicrograph showing discrete spodumene (Sp) and spodumene associated with gangue (G) forming simple and complex intergrowths.

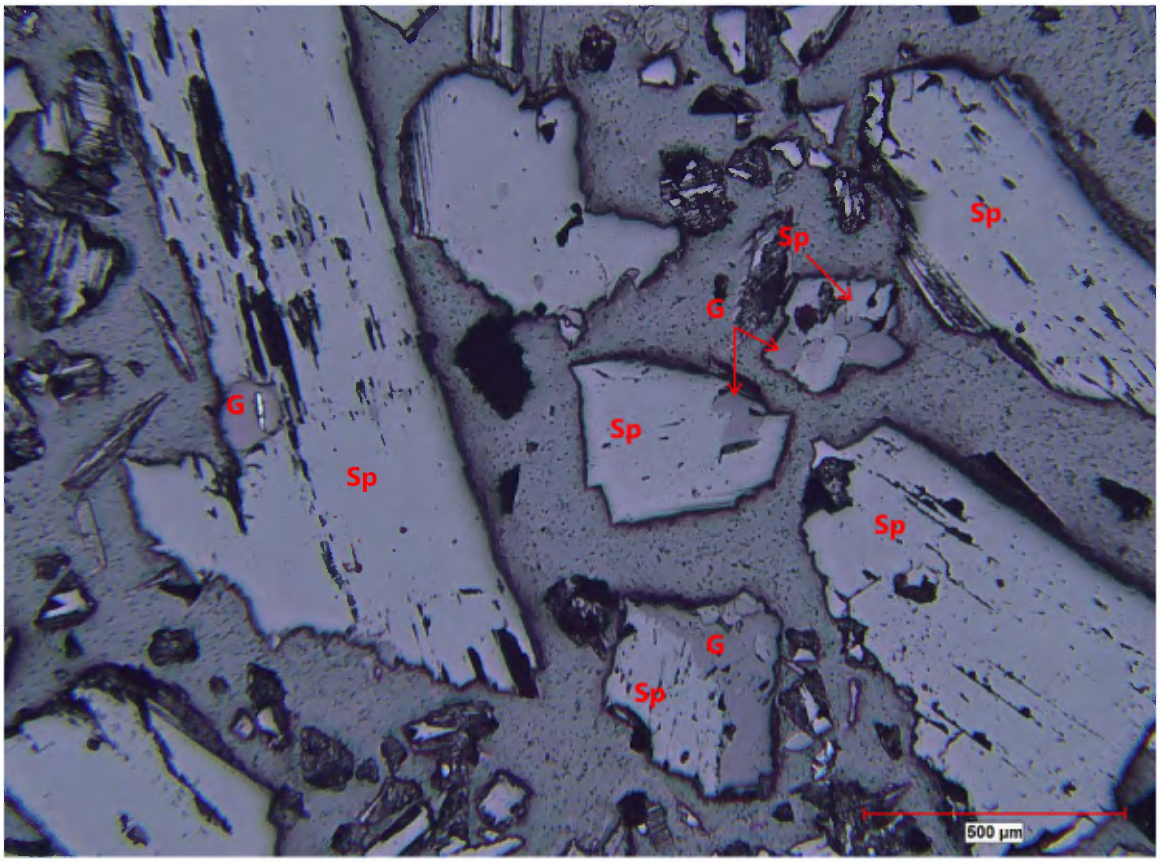


Figure 9. Dike 11.2, Minus 1.7-mm by 25-µm Sink Product, Example 2

Photomicrograph showing discrete spodumene (Sp) and spodumene intergrown with gangue (G).

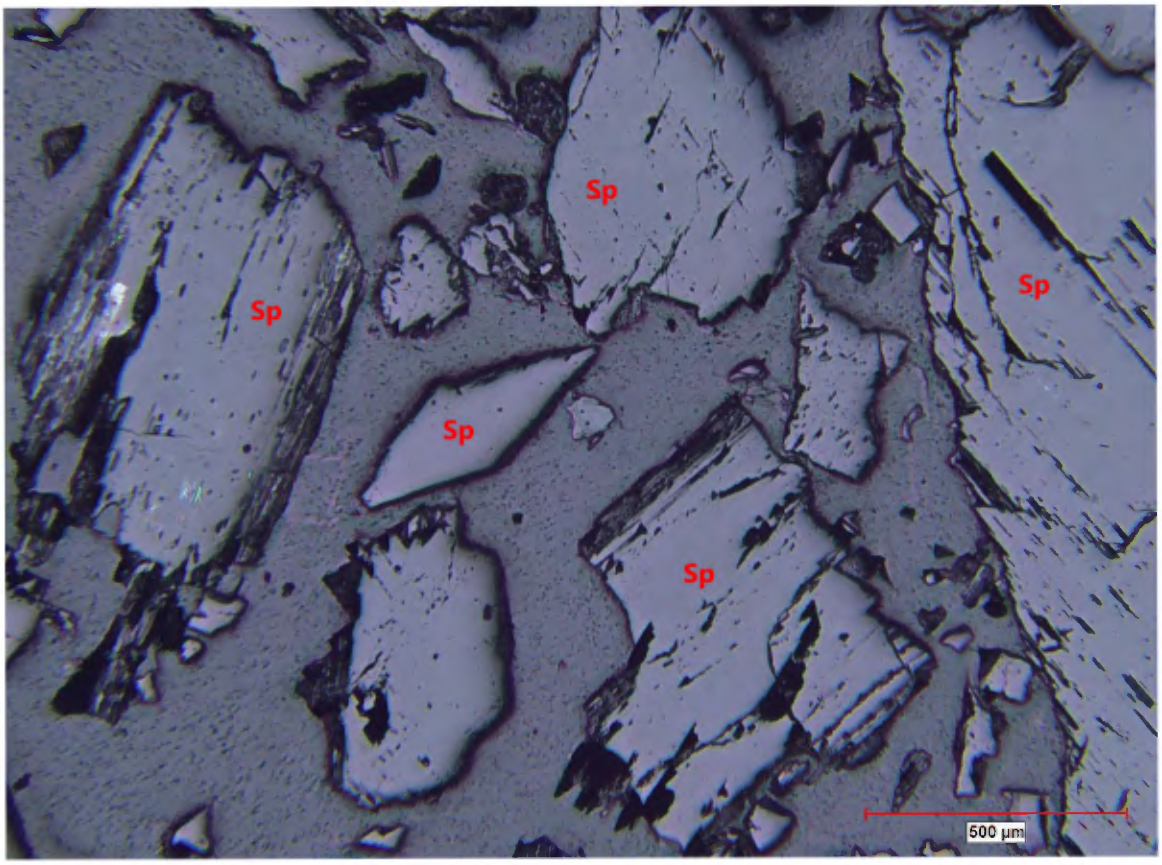


Figure 10. Dike 11.2, Minus 1.7-mm by 25-µm Sink Product, Example 3

Photomicrograph showing coarse, discrete spodumene (Sp).

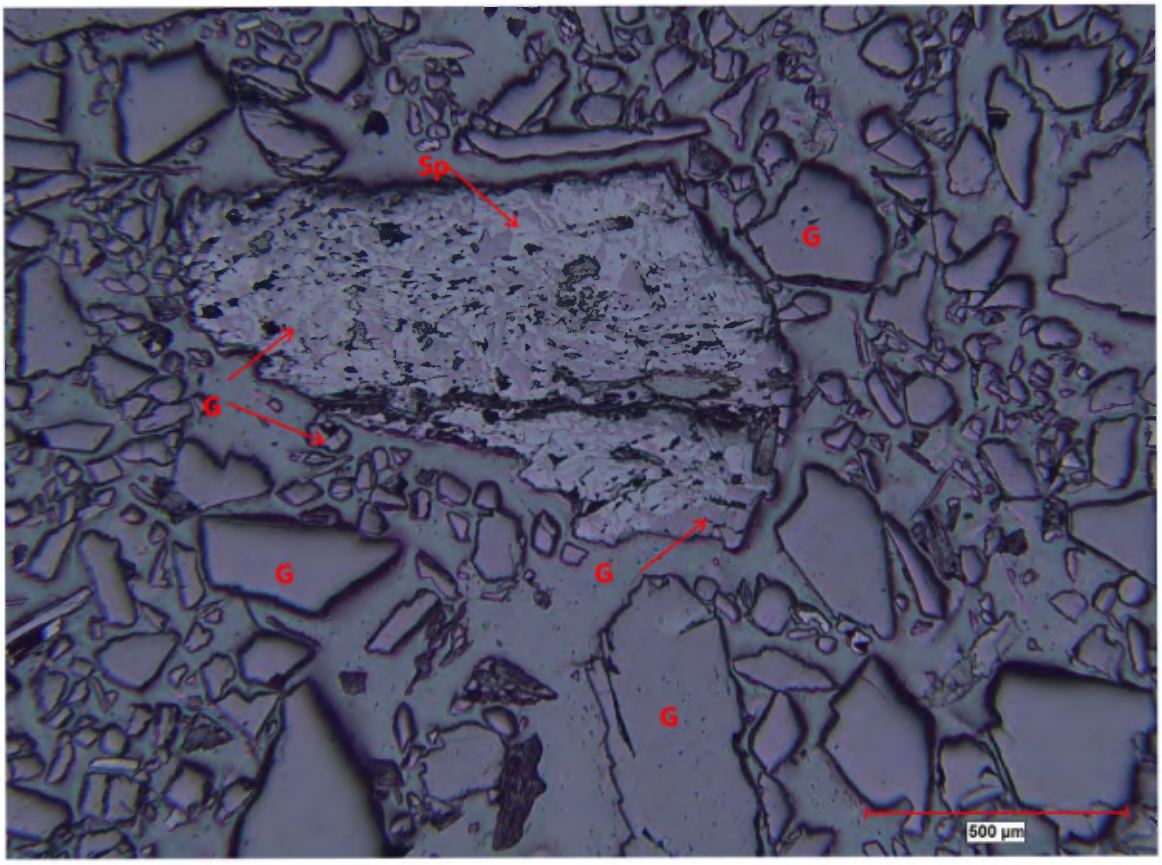


Figure 11. Dike 11.2, Minus 1.7-mm by 25-µm Float Product, Example 1

Photomicrograph showing discrete gangue (G) and a complicated intergrowth of gangue and spodumene (Sp).

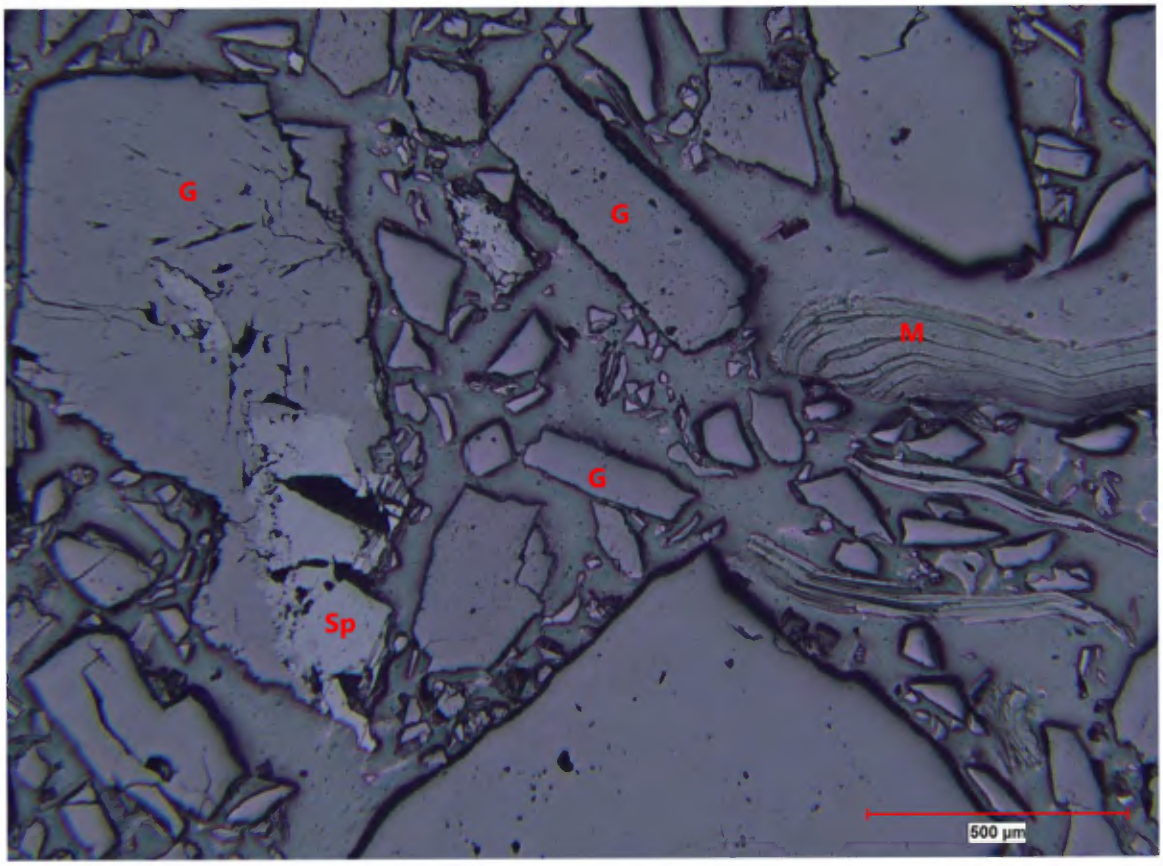


Figure 12. Dike 11.2, Minus 1.7-mm by 25-µm Float Product, Example 2

Photomicrograph showing discrete gangue (G) and mica (M) and an intergrowth of gangue and spodumene (Sp).

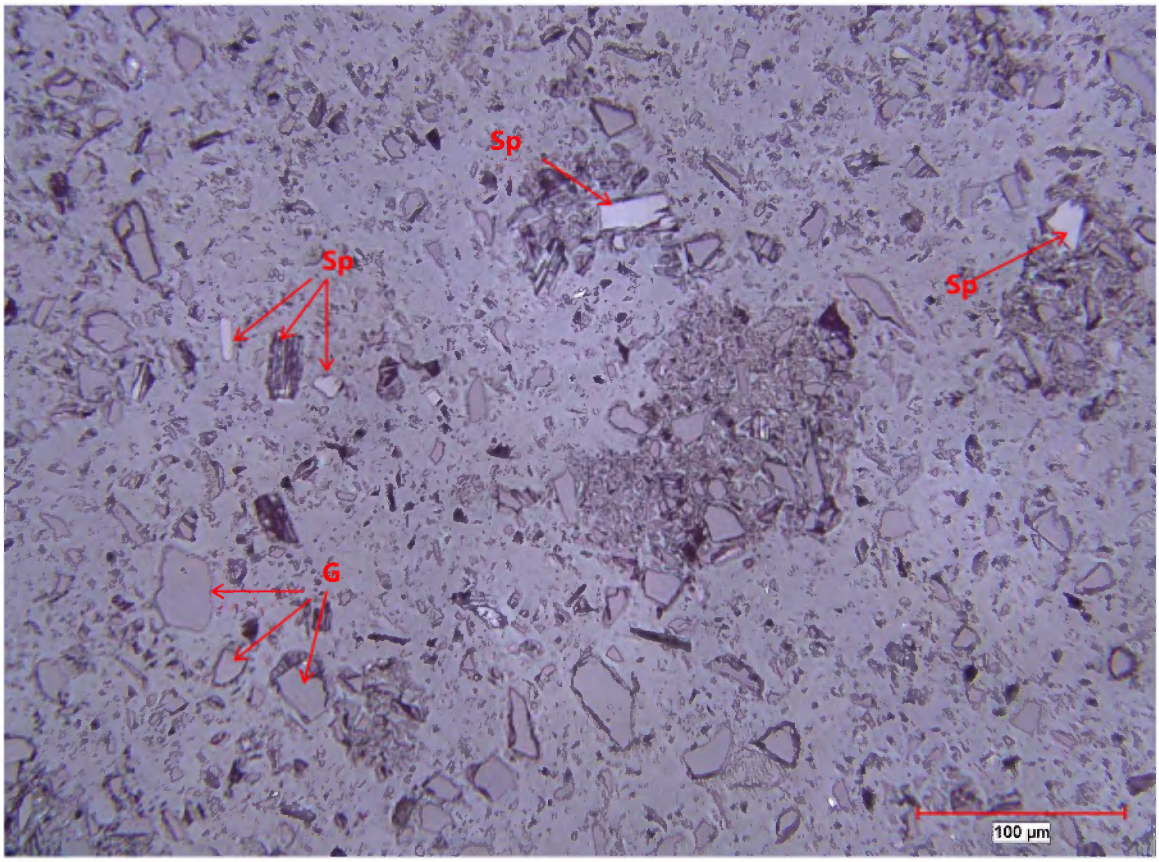


Figure 13. Dike 11.2, Minus 25-µm Slimes

Photomicrograph showing discrete spodumene (Sp) and gangue (G).

Table 14. XRD Results

Sample	Mineral Constituents			
	Major	Subordinate	Minor	Trace
Minus 1.7-mm Head	Quartz, albite		Spodumene, microcline	Muscovite
1.7-mm by 25-µm Sink	Spodumene		Quartz, albite[a](?)	
1.7-mm by 25-µm Float	Quartz	Albite	Microcline	Spodumene, muscovite
Minus 25-µm Slimes	Quartz	Albite	Spodumene, microcline	Muscovite

[a]The main albite line at 27.89° 2θ is masked by the 35% intensity peak of the spodumene at 27.94° 2θ.

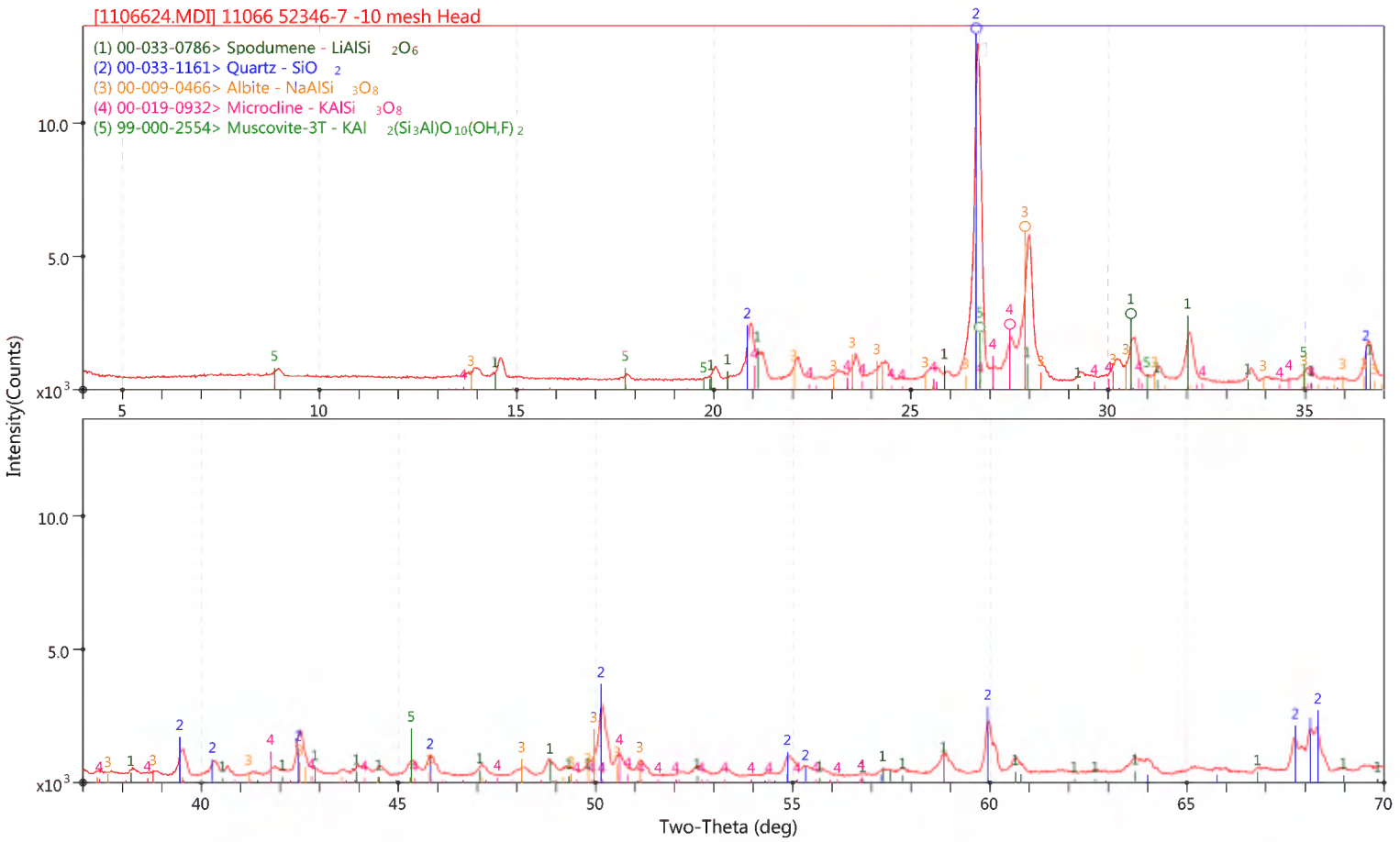


Figure 14. XRD Pattern, Dike 11.2, Minus 1.7-mm Head

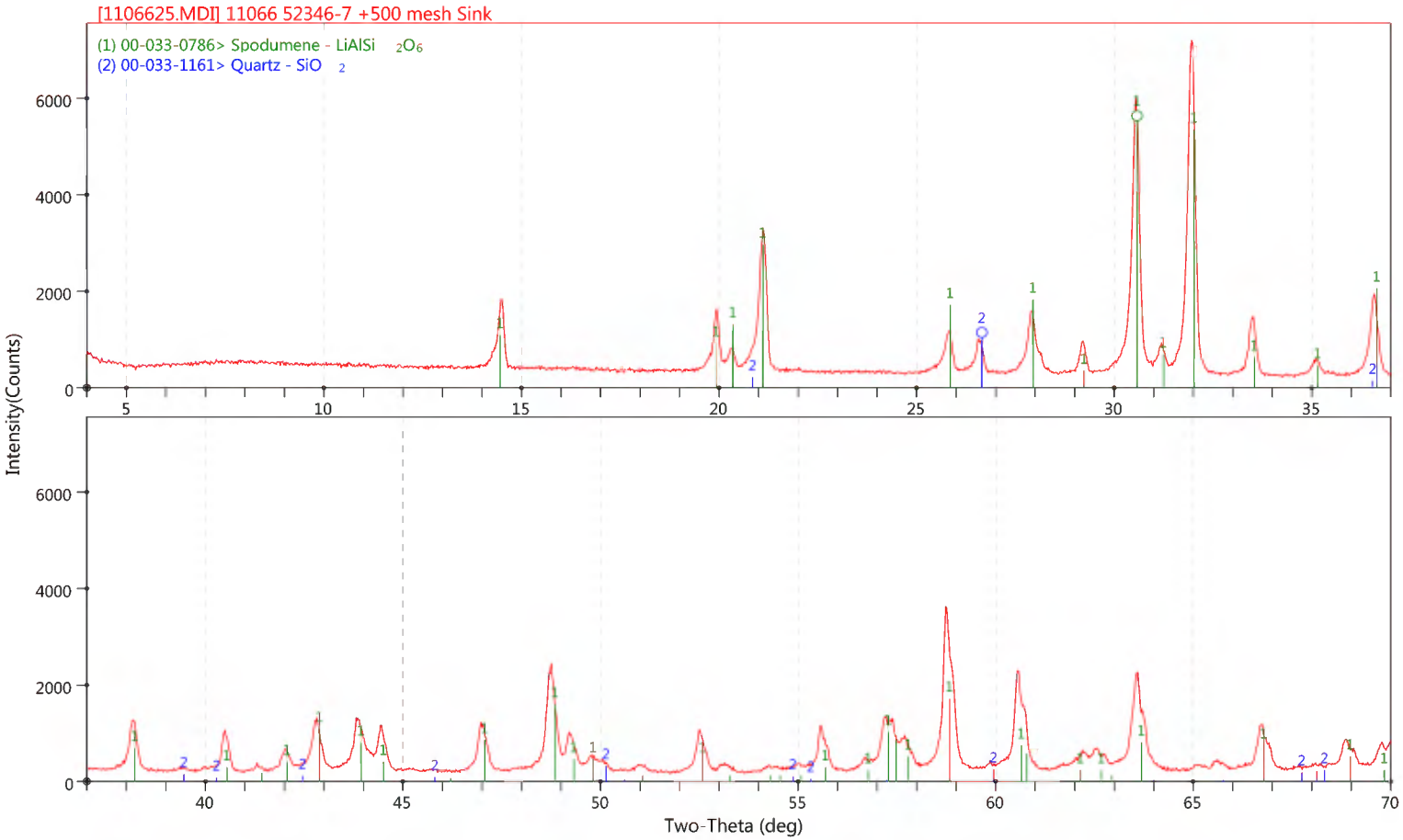


Figure 15. XRD Pattern, Dike 11.2, 1.7-mm by 25-µm Sink

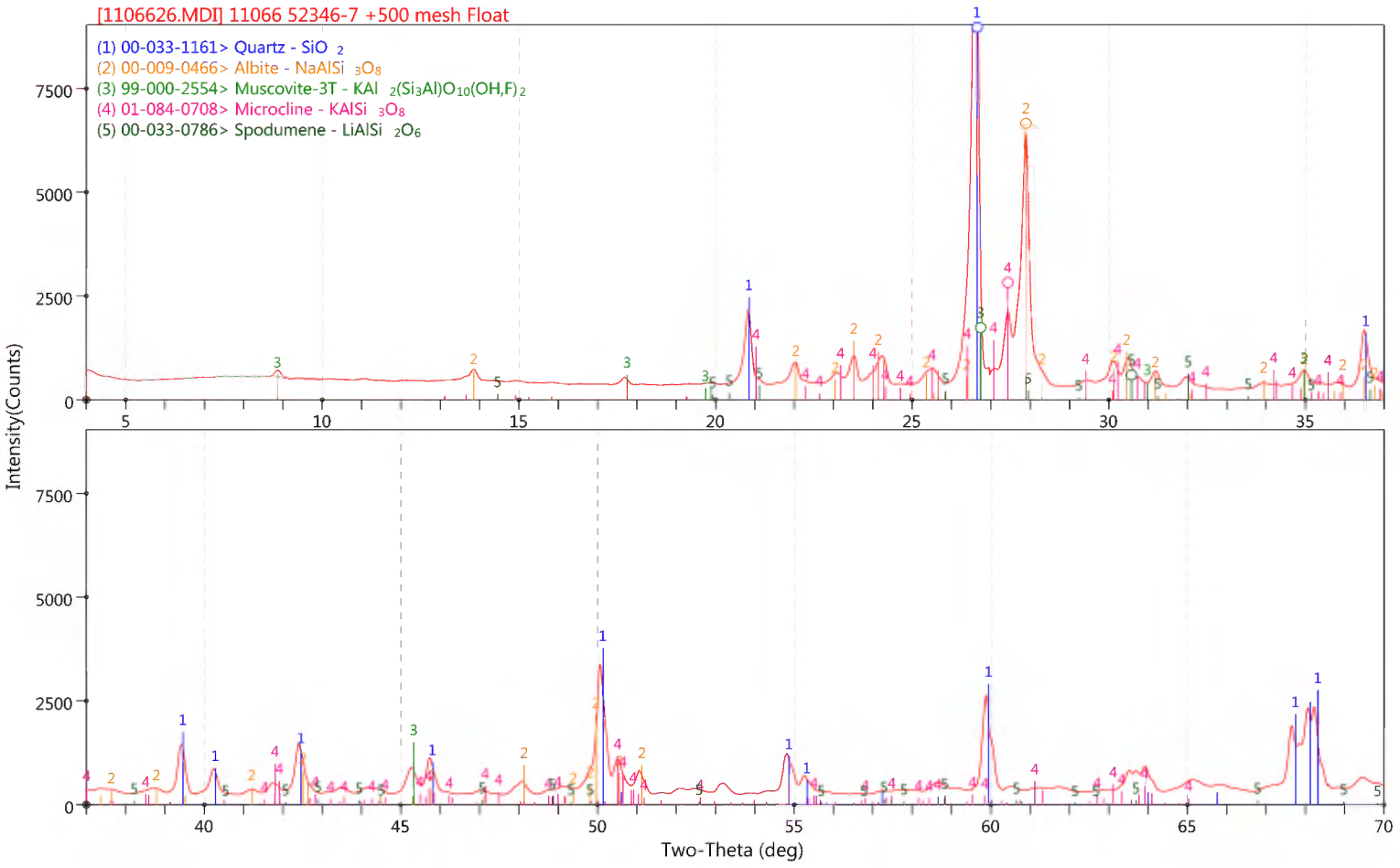


Figure 16. XRD Pattern, Dike 11.2, 1.7-mm by 25-µm Float

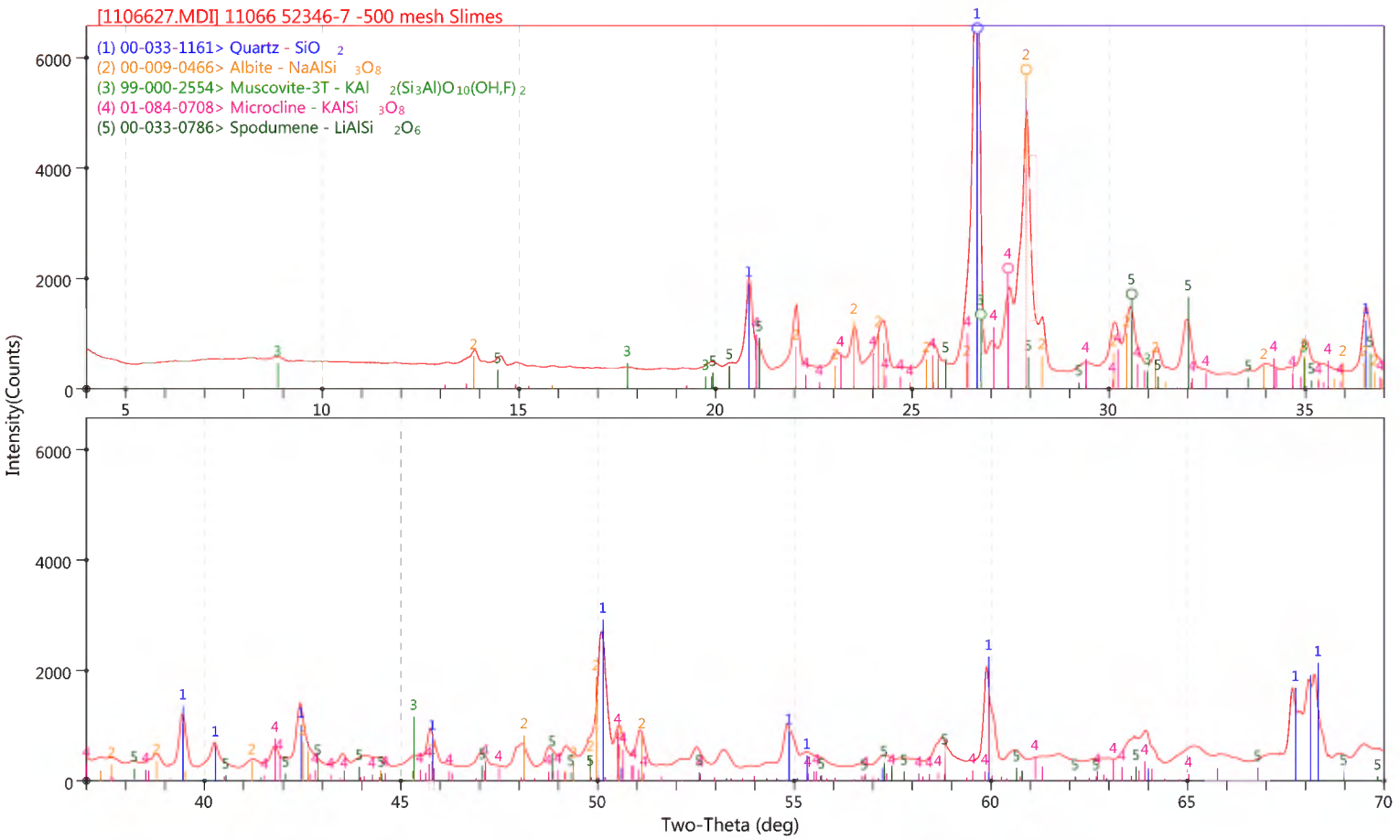


Figure 17. XRD Pattern, Dike 11.2, Minus 25-µm Slimes

COMMINUTION TESTS

Selected sawed core dike samples were subjected to SMC, BW_i, and A_i testing. Table 15 lists the samples tested and includes the Hazen number assigned to each sample for internal identification and future reference. This table also lists the measured BW_i and A_i values for each of these samples. The SMC parameters for these samples, determined by SMC Testing Pty Ltd., are presented in Table 16. The previously issued comminution report, including the data for each of these tests, is included in Appendix D.

Table 15. BW_i and A_i Test Results

HRI	Client ID	BW_i, kWh/t	A_i, g
52461-1	Dike 7.2	18.0	0.4659
52461-2	Dike 7.6	17.5	0.4244
52461-3	Dike 8.3	17.6	0.4443
52461-4	Dike 8.7	17.5	0.3936
52461-5	Dike 9.2	17.6	0.4151
52461-6	Dike 10.4	17.6	0.4224
52461-7	Dike 11.2	18.2	0.4380
52461-8	Dike 12.2	17.0	0.4174
52461-9	Dike 13.2	16.9	0.4136
52461-10	Dike 14.2	17.4	0.4280
52461-11	Dike 15.1	17.9	0.3750

Table 16. Summary of SMC Breakage Evaluations

Parameter	Value										
	52461-1	52461-2	52461-3	52461-4	52461-5	52461-6	52461-7	52461-8	52461-9	52461-10	52461-11
sg (by weighing in water and air)	2.53	2.69	2.71	2.76	2.76	2.69	2.74	2.73	2.72	2.71	2.74
SMCT Parameters											
A	59.8	65.2	64.6	64.3	66.8	63.0	60.5	62.7	63.5	63.4	63.0
b	1.31	1.10	1.11	127	1.00	1.06	1.40	1.13	1.21	1.27	1.23
A × b	78.3	71.7	71.7	81.7	66.8	66.8	84.7	70.9	76.8	80.5	77.5
SMC Test DW_i, kWh/m^3	3.23	3.77	3.79	3.38	4.14	4.03	3.22	3.84	3.54	3.39	3.54
DW_i, %	19	24	25	20	29	27	19	25	22	20	22
M_{ia}, kWh/t	11.6	12.3	12.3	11.0	12.9	13.0	10.7	12.3	11.6	11.2	11.5
M_{ih}, kWh/t	7.4	8.1	8.1	7.1	8.7	8.7	6.8	8.2	7.5	7.2	7.5
M_{ic}, kWh/t	3.8	4.2	4.2	3.7	4.5	4.5	3.5	4.2	3.9	3.7	3.9
t_a	0.80	0.69	0.68	0.77	0.63	0.64	0.80	0.67	0.73	0.76	0.73

A = maximum breakage; b = relation between energy and impact breakage; A × b = overall AG–SAG hardness; DW_i = drop-weight index; M_{ia} = work index for grinding coarser particles (>750 µm) in tumbling mills; M_{ih} = work index for grinding in high-pressure grinding rolls; M_{ic} = work index for size reduction in conventional crushers; and t_a = low-energy abrasion component of breakage

The mineralogical examination of the James Bay ore showed that the majority of the spodumene exists as relatively large, discrete crystals, suggesting that a concentrate could be generated by flotation. Flotation has been successfully used for other spodumene deposits, and methods developed for these deposits were adapted for this ore. A series of batch flotation experiments was completed to determine the proper grind size, reagent requirements, and other operating parameters.

FLOTATION EQUIPMENT AND PROCEDURES

All batch flotation experiments were carried out in a Denver subaeration flotation machine using cells with 1- and 2.2-L capacities. The impeller speeds were 1,200 rpm for the 1-L cell and 1,500 for the 2.2-L cell.

The pulp density for the rougher stage was estimated at 30–35% solids. The pulp densities for the first and second cleaner stages were between 14 and 27% and 12 and 23%, respectively. The pulp density for the third cleaner stage used in Test 30230-112 was 14%. A summary of the pulp densities is shown in Table 17.

Table 17. Flotation Pulp Densities

Test	Solids, %			
	Rougher[a]	Cleaner 1	Cleaner 2	Cleaner 3
3230-102	30–35	22.0	22.9	na
3230-103	30–35	22.8	17.7	na
3232-107	30–35	18.2	18.2	na
3230-108	30–35	24.1	20.7	na
3230-109	30–35	21.5	16.9	na
3230-110	30–35	22.7	19.2	na
3230-111	30–35	20.9	16.9	na
3230-112	30–35	19.6	15.0	14
3230-113	30–35	27.1	22.2	na
3230-114	30–35	13.6	11.5	na
3230-115	35	27.0	22.0	na
3230-116	35	14.0	12.0	Na

na = not applicable
[a]estimated

After desliming by hydrocycloning, the feed was conditioned in the flotation cell for 10 min at a pulp density of 50–55% solids. Flotation retention times ranged from 4 to 5 min for the rougher stages and 3 to

4 min for the cleaner stages. Sodium hydroxide (200 g/L) was used during grinding to clean the spodumene surfaces. The pH ranged from 7.5 to 8.2 for the rougher stage.

The reagent schedule and dosages for spodumene flotation and mica flotation are shown in Table 18. A tall oil fatty acid (Sylfat FA-1) from Arizona Chemical Company was used as a spodumene collector for the majority of the work. A tall oil fatty acid (Flotinor 1682) from Clariant Corporation was used as a spodumene collector for Test 3230-102. An amine (Aeromine 3000C) from Cytec Industries and Fuel Oil #5 from Texaco Corporation were used as mica collectors for Test 3230-112. A polyglycol frother (Oreprep F-549) from Cytec Industries was the only frother used.

Table 18. Reagent Schedule and Dosage

Test	Function							
	pH Control		Spodumene Collector		Mica Collector		Frother	
	Reagent	Dosage, kg/t	Reagent	Dosage, kg/t	Reagent	Dosage, kg/t	Reagent	Dosage, kg/t
3230-102	NaOH	0.249	Flotinor 1682	0.754	na	na	na	na
3230-103	NaOH	0.249	Sylfat FA-1	0.754	na	na	Oreprep F-549	0.04
3232-107	NaOH	0.249	Sylfat FA-1	0.754	na	na	Oreprep F-549	0.04
3230-108	NaOH	0.249	Sylfat FA-1	0.754	na	na	Oreprep F-549	0.04
3230-109	NaOH	0.249	Sylfat FA-1	0.754	na	na	Oreprep F-549	0.04
3230-110	NaOH	0.249	Sylfat FA-1	0.754	na	na	Oreprep F-549	0.04
3230-111	NaOH	1.356	Sylfat FA-1	0.754	na	na	na	na
3230-112	NaOH (spodumene) H_2SO_4 (mica)	1.356 0.20	Sylfat FA-1	0.754	Aeromine 3000C Fuel Oil #5	0.35 2.55	na	na
3230-113	NaOH	1.356	Sylfat FA-1	0.754	na	na	na	na
3230-114	NaOH	1.356	Sylfat FA-1	0.754	na	na	na	na
3230-115	NaOH	1.356	Sylfat FA-1	0.754	na	na	na	na
3230-116	NaOH	1.356	Sylfat FA-1	0.754	na	na	na	na

na = not added

FLOTATION RESULTS

A series of ten small-scale flotation tests and two bulk flotation tests was performed on 1- and 2-kg and 16-kg splits, respectively. The objective of the work was to investigate direct spodumene flotation and produce a spodumene concentrate of reasonable grade for subsequent roasting and leaching work. Individual flotation data sheets are shown in Appendix E. A summary of test results, including lithium assays and distributions, estimated spodumene content, and percent solids (pulp density) is presented in Table 19.

The lithium recovery in the cleaner concentrates ranged from 62% (Test 3230-103) to 80% (Tests 3230-113 and 3230-114 combined). The spodumene content in the cleaner concentrate (Tests 3230-111, -112, and -114) is estimated at 90%, which is based on optical and XRD methods. The amount of lithium in the slimes ranged from 7 to 13% in 11–20% of the total weight.

It must be emphasized that the lithium assays of the rougher and cleaner concentrates do not accurately reflect the lithium distributions and recoveries of lithium from spodumene. This is because both fine spodumene and lithium-bearing mica report to the tails. Therefore, the actual recoveries of spodumene are somewhat higher than indicated by the lithium assays.

Beginning with Test 3230-111, the NaOH dosage was increased to 1.35 kg/t from the previous 0.25 kg/t. A technical literature review indicated cleaning is important for selectivity, and most of the NaOH dosages stated in the literature are in the range of 1.35–2.71 kg/t. The estimated spodumene grade in the cleaner concentrate improved to 90% (Test 3230-111) from an estimated 60% spodumene in the cleaner concentrate in Test 3230-110.

For Flotation Test 3230-112, a third cleaner stage was added because a microscopic examination of the cleaner tails from previous flotation work indicated good quartz, feldspar, and mica rejection. The spodumene grade improved slightly but probably not enough to justify the additional cleaner stage.

After the direct spodumene flotation, a mica float was performed on the combined first cleaner tailings and the spodumene rougher tails. The combined tails were deslimed, then conditioned with H_2SO_4 to adjust the pH to 3, followed by conditioning at about 60% solids using an amine collector and a fuel oil. There was no selectivity during the mica float, possibly due to an overdosing of collector. No other mica flotation was performed.

Because of the abundance of fine spodumene in the cleaner tails, two 1-kg batches of Composite 1 were ground with 1.35 kg/t NaOH and deslimed using a hydrocyclone; the cyclone underflow was wet-screened at 150 μm. Both the coarse and fine fractions were subjected to flotation with two stages of cleaning. For the coarse fraction float (3230-113), the second cleaner concentrate contained an estimated 80–85% spodumene and the rougher tails contained less than 1% spodumene. For the fine fraction float (3230-114), the second cleaner concentrate contained an estimated 90% spodumene and the rougher tails contained less than 1% spodumene, indicating the fine spodumene was recovered by the split flotation procedure. Microscopic examination of the flotation products from both tests indicate very high-grade

concentrates and improved spodumene recoveries. A summary of results for both tests is shown in Table 20.

Tests 3230-115 and 3230-116 were bulk flotations following the coarse–fine split flotation procedure used for Tests 3230-113 and 3230-114. A summary of the concentrate assay results is shown in Table 21. About 2.7 kg of spodumene concentrate were produced from 16 kg of feed for roasting and leaching work. The combined spodumene concentrate assayed 2.8% Li.

Table 19. Summary of Flotation Tests

Test	Feed	Solids, %			Cleaner Concentrate Weight, %	Assay, %				Distribution, %	
						Concentrate		Rougher Tails		Concentrate	Rougher Tails
		Rougher	Cleaner 1	Cleaner 2		Li	Estimated Spodumene	Li	Estimated Spodumene	Li	Li
3230-102	Dike 11.2	30–35	22.0	22.9	54.8	1.1	na	0.45	na	76.9	7.1
3230-103	Dike 11.2	30–35	22.8	17.7	22.6	2.3	na	0.31	na	61.5	16.4
3232-107	Comp 1	30–35	18.2	18.2	23.8	2.8	75	0.18	5	73.8	9.8
3230-108	Comp 2	30–35	24.1	20.7	26.5	2.4	70	0.15	5	78.9	8.4
3230-109	Comp 3	30–35	21.5	16.9	25.2	2.5	70	0.15	5	76.7	9.5
3230-110	Comp 4	30–35	22.7	19.2	25.0	2.7	60	0.19	7	77.2	11.6
3230-111	Comp 1	30–35	20.9	16.9	22.4	3.0	90	0.13	2	76.7	7.7
3230-112	Comp 1	30–35	19.6	15.0[a]	19.5	3.2	90	na	na	73.1	na
3230-113	Comp 1, underflow plus 150 µm	30–35	27.1	22.2	15.6	3.2	80–85	0.10	<1	58.1[b]	3.4
3230-114	Comp 1, underflow minus 150 µm	30–35	13.6	11.5	7.6	2.5	90	0.05	<1	22.1[b]	1.1
3230-115	Master comp, underflow plus 150 µm	35	27.0	22.0	11.4	3.1	na	0.17	na	43.8[c]	6.7
3230-116	Master comp, underflow minus 150 µm	35	14.0	12.0	7.8	2.9	na	0.06	na	28.0[c]	1.7

na = not applicable

[a]Three cleaner stages were used; the third cleaner was 14.0% solids. All other tests used two cleaner stages.

[b]Coarse–fine flotation, combined lithium recovery was 80%

[c]Coarse–fine flotation, combined lithium recovery was 71.8%

Table 20. Summary of Results for Tests 3230-113 and 3230-114

Product	Weight		Li		
	g	%	Analysis, %	Weight, g	Distribution, %
Cyclone underflow plus 150 µm					
Cleaner 2 concentrate	302.5	15.6	3.2	9.6	58.0
Cleaner 2 tails	35.8	1.8	0.78	0.3	1.7
Cleaner 1 tails	83.0	4.3	0.36	0.3	1.8
Rougher tails	573.0	29.6	0.10	0.6	3.4
Total	**994.3**	**51.3**	**1.08**	**10.8**	**64.9**
Cyclone underflow minus 150 µm					
Cleaner 2 concentrate	146.7	7.6	2.5	3.7	22.0
Cleaner 2 tails	17.5	0.9	0.61	0.1	0.6
Cleaner 1 tails	41.1	2.1	0.26	0.1	0.6
Rougher tails	395.3	20.4	0.05	0.2	1.1
Total	**600.6**	**31.0**	**0.67**	**4.0**	**24.4**
Cyclone overflow	343.55	17.7	0.51	1.8	10.7
Calculated feed	1938.5	100.0	0.85	16.6	100.0
Assayed feed			0.78		

Table 21. Concentrate Assay Summary for Tests 3230-115 and 3230-116

Product	Weight		Li (Original Assays)			Li (Repeat Assays)		
	g	%	Analysis, %	Weight, g	Distribution, %	Analysis, %	Weight, g	Distribution, %
3230-115 Cleaner 2 concentrate (sand)	1,768	59.4	3.1	55.0	61.0	3.0	53.0	64.6
3230-116 Cleaner 2 concentrate (slimes)	1,210	40.6	2.9	35.2	39.0	2.4	29.0	35.4
Calculated feed	2,978	100.0	3.03	90.2	100.0	2.76	82.1	100.0
Assayed feed			2.85			2.82		

Leaching and recovery of lithium from spodumene ore requires that the naturally occurring α-spodumene be converted to β-spodumene by thermal treatment. A series of seven batch calcining experiments were conducted using a 4-in.-diameter quartz kiln to investigate the temperature and time necessary to achieve nearly complete conversion of α-spodumene to β-spodumene. Two additional calcining runs were performed to generate sufficient material for the leaching and lithium recovery experiments.

BATCH KILN SYSTEM

A batch kiln processing system was used to conduct the thermal treatment of the spodumene to complete the alpha-to-beta conversion. The batch kiln consisted of a quartz tube that was expanded to a 4-in.-diameter (10-cm) section in the middle to form a containment chamber. The chamber was 14 in. long and had numerous small dimples depressed into the glass to aid in mixing the test charge as the kiln rotated. The inlet and outlet ends of the kiln were 1-in.-diameter (25.4-mm), concentrically attached tubes with ground-glass fittings that created a rotation joint and seal against ambient pressure. The kiln was positioned in a horizontal, split-shell furnace, which had a programmable temperature controller. Adjusting the flow of specific gases (air and N_2) with rotameters controlled the roasting atmosphere. Gas was metered from high-pressure cylinders, and a variable-speed gear motor engaged to a sprocket on the kiln controlled the rotational speed of the kiln.

Thermocouples positioned in the split-shell furnace (between the kiln and heating elements) and the test charge monitored the temperature of each roast. The process offgas was directed into a water bubbler to create a slight backpressure on the system. A slipstream of the process gas (upstream of the scrubber) was directed through a filter and into continuous emissions monitors (CEMs) to determine the composition of the process gas. The monitors analyzed the gas for O_2, CO_2, and CO. The specific gas analyzers and ranges are listed in Table 22. Temperature and gas composition measurements were recorded continuously with a data acquisition system and were monitored from a computer screen. Operational data were recorded manually on a round sheet for each test. Copies of the individual data sheets are provided in Appendix F.

Table 22. Gas Analyzer Specifications

Constituent	Instrument (Range)
O_2	Horiba MPA-510 (0–25%)
CO_2	Infrared (0–25%)
CO	Infrared (0–20%)

To initiate testing, the tare weight of the empty kiln was recorded and then a measured mass of the α-spodumene concentrate (Cleaner 2 concentrate) was added and the gross weight was recorded. The

loaded kiln was positioned within the furnace, connections were made to the end fittings, and the sprocket was connected to the gear motor. The kiln rotation and metered sweep gas were started. Power was applied to the furnace, and the system was heated to the desired temperature. When the roast was complete (by maintaining the desired retention time at temperature), the sweep gas was changed from air to N_2, the furnace heat was stopped, and the split-shell was opened to initiate cooling. After a reasonable period, the kiln was removed from the furnace and placed on a cooling rack to complete the cooling. When cooling was complete, the kiln and contents were weighed and the calcine mass was determined by difference between the final gross weight and the kiln tare weight. Calcine was recovered and saved in a screw-cap jar. Subsequently, the calcine was sampled and analyzed by XRD to determine the presence of any residual α-spodumene. Photographs of the batch kiln system are shown in Figures 18 and 19. A process flow diagram is shown in Figure 20.

Figure 18. Photograph of the Batch Kiln System

Figure 19. Photograph of Drive Mechanism

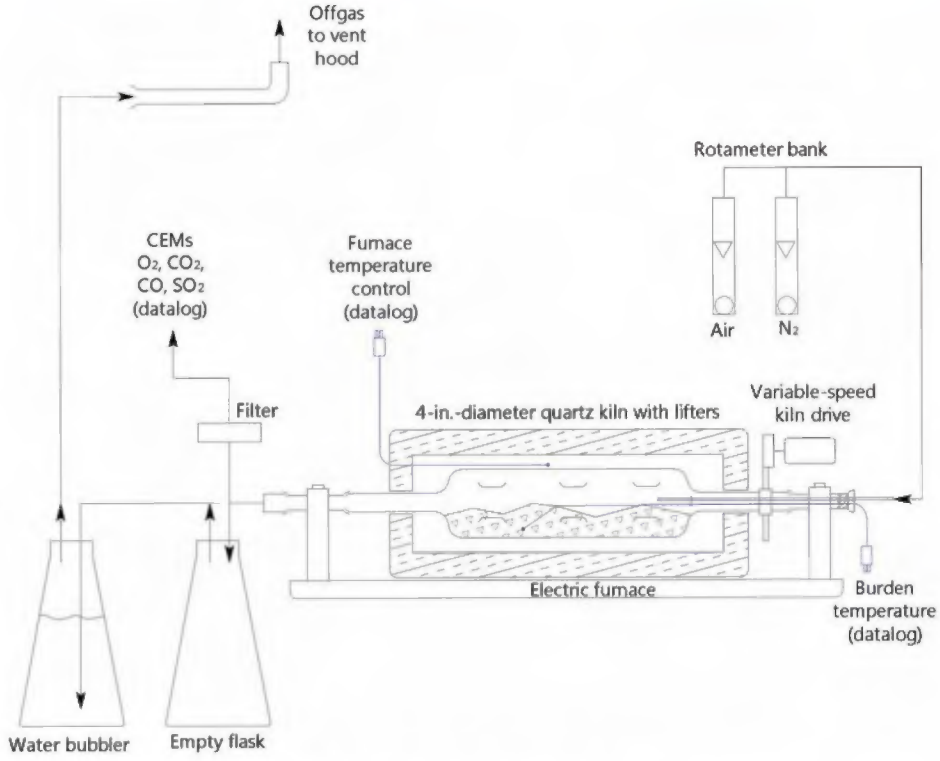


Figure 20. Batch Kiln Process Flow Diagram

CALCINING EXPERIMENTS

Flotation Concentrate 3230-116 was used for the calcining program. Initially, a DTA and a TGA were performed on the concentrate in air to investigate temperatures up to 1,100°C, when reactions proceed during calcining. A TGA shows the temperatures at which a sample changes weight, indicating the temperatures at which reactions take place. A DTA measures temperature differences between a sample and an inert alumina standard as the sample is heated at a constant rate, showing the temperatures at which exothermic and endothermic reactions occur.

The TGA thermogram for this concentrate, shown in Figure 21, did not provide much information because there was little weight loss. The DTA thermogram in Figure 22 indicated an endotherm between 900 and 1,100°C, which is in the range that α-spodumene is converted into β-spodumene. Because there is no weight change for this conversion, the lack of a weight change in the TGA at these temperatures supports the spodumene phase change as the cause of the endotherm seen in the DTA.

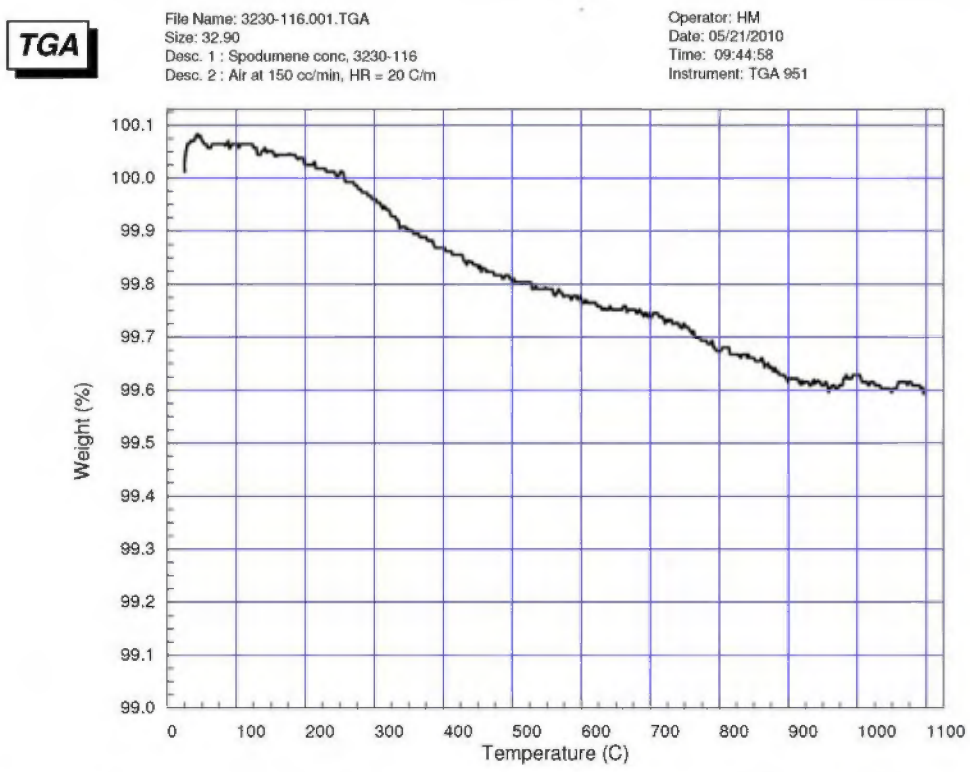


Figure 21. TGA Thermogram of the Spodumene Concentrate 3230-116

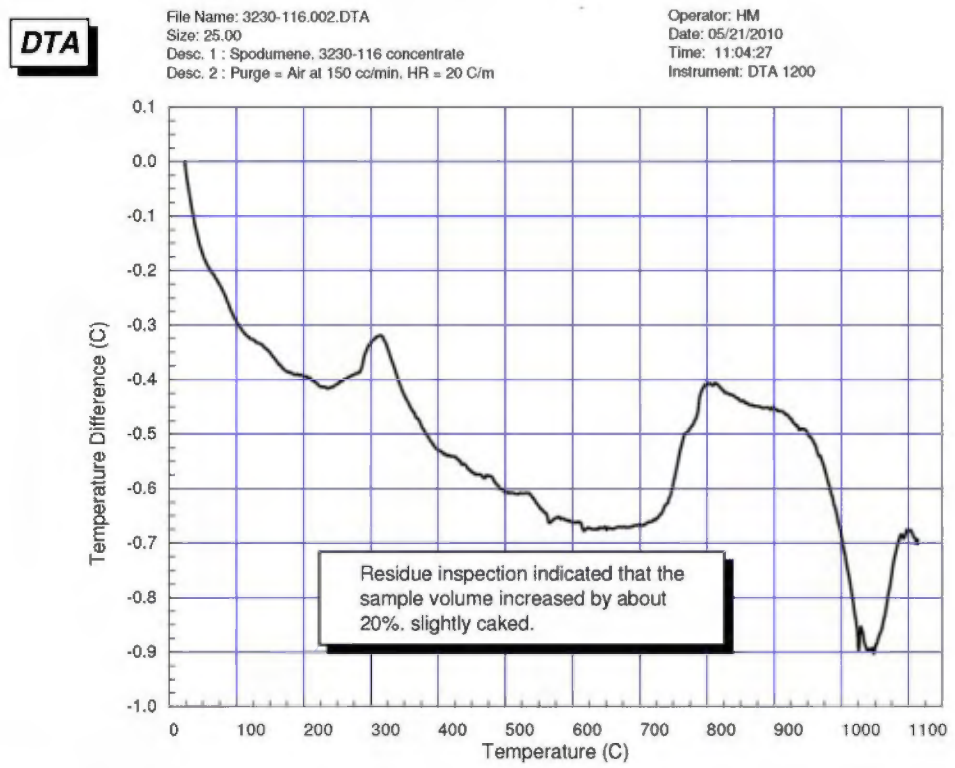


Figure 22. DTA Thermogram of the Spodumene Concentrate 3230-116

Based on the results of the DTA and TGA, four calcining experiments were conducted to investigate the effect of temperatures of 950, 1,000, and 1,050°C at roasting times of 60 min, with Experiment 4 repeating the Experiment 1 conditions. A photograph of the calcined product, shown in Figure 23, illustrates the exfoliation of the spodumene after calcining. The resulting calcines were analyzed by XRD to determine the amount of α-spodumene remaining in the sample. The data, summarized in Table 23, showed substantial α-spodumene remaining when calcining at 950°C. Less α-spodumene remained when calcining at 1,000°C, and only a trace remained at 1,050°C. Table 24 shows the mineral constituents from the XRD in the calcines.

Figure 23. Photograph of Calcined Product

Table 23. Summary of Batch Calcining Experiments

Experiment	Feed	Feed, g	Temperature, °C	Roasting Time, min	Calcine, g	α-spodumene[a]
1	Concentrate	303.4	1,000	60	301.5	minor
2	Concentrate	304.8	1,050	60	297.6	trace
3	Concentrate	302.8	950	60	301.3	major
4	Concentrate	302.7	1,000	60	301.1	na
5	Concentrate	299.9	1,050	30	298.3	minor
6	Concentrate	1,205.1	1,050	60	1,198.7	trace
7	Concentrate	496.5	1,050	60	495.4	none
8	Calcines 1, 2, and 5	791.7	1,050	60	791.3	trace
9	Calcines 1, 3, and 5	122.8	1,050	60	122.6	none

na = not analyzed

[a]The amount of α-spodumene was determined by XRD.

Table 24. XRD Results of Flotation Concentrate and Calcines

Sample	Temperature, °C	Mineral Constituents		
		Major	Minor	Trace
Feed Flotation Concentrate	--	α-spodumene	Quartz	Microcline
Experiment 1 Calcine	1,000	β-spodumene	α-spodumene Quartz	Microcline
Experiment 2 Calcine	1,050	β-spodumene	Quartz	α-spodumene Microcline
Experiment 3 Calcine	950	α-spodumene β-spodumene	Quartz Microcline	
Experiment 5 Calcine	1,050	β-spodumene	Quartz	α-spodumene Microcline
Experiment 6 Calcine	1,050	β-spodumene	Quartz Microcline	α-spodumene
Experiment 7 Calcine	1,050	β-spodumene	Quartz Microcline	
Experiment 8 Calcine	1,050	β-spodumene	Quartz	α-spodumene Microcline
Experiment 9 Calcine	1,050	β-spodumene	Quartz	Microcline
Final Calcine Composite	1,050	β-spodumene	Quartz	α-spodumene Microcline

Experiment 5 investigated reducing the calcining time to 30 min, but more α-spodumene remained in the calcine than at 60 min. Figures 24 and 25 show the patterns for the calcines from Experiments 2 and 5 overlaid for direct comparison. The remaining XRD patterns for these experiments are provided in Appendix G. From this series of experiments, 1,050°C was selected as the desired calcining temperature.

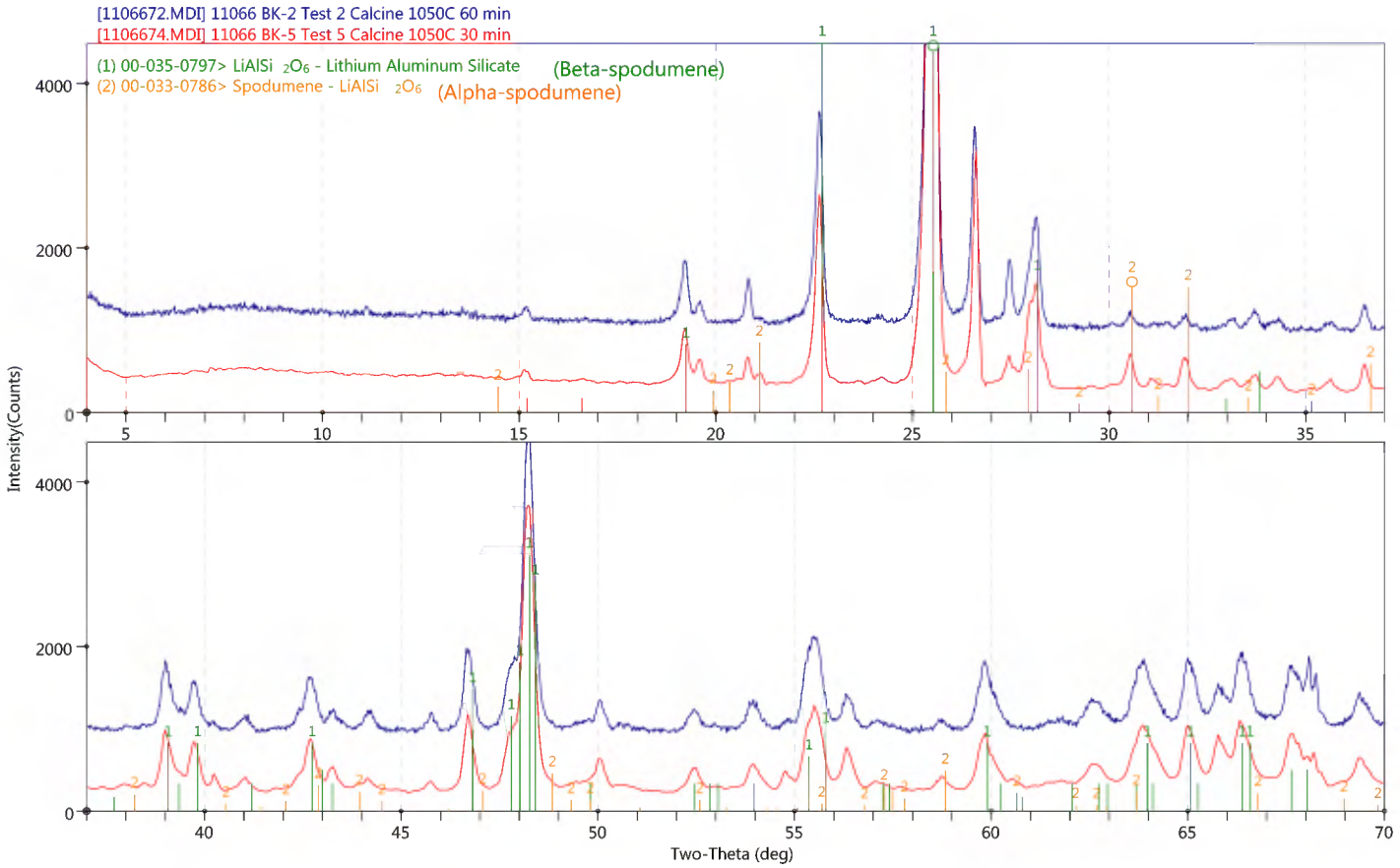


Figure 24. Overlay of XRD Patterns, Tests 2 and 5 Showing only α- and β-spodumene (1,050°C calcines at 30 and 60 min)

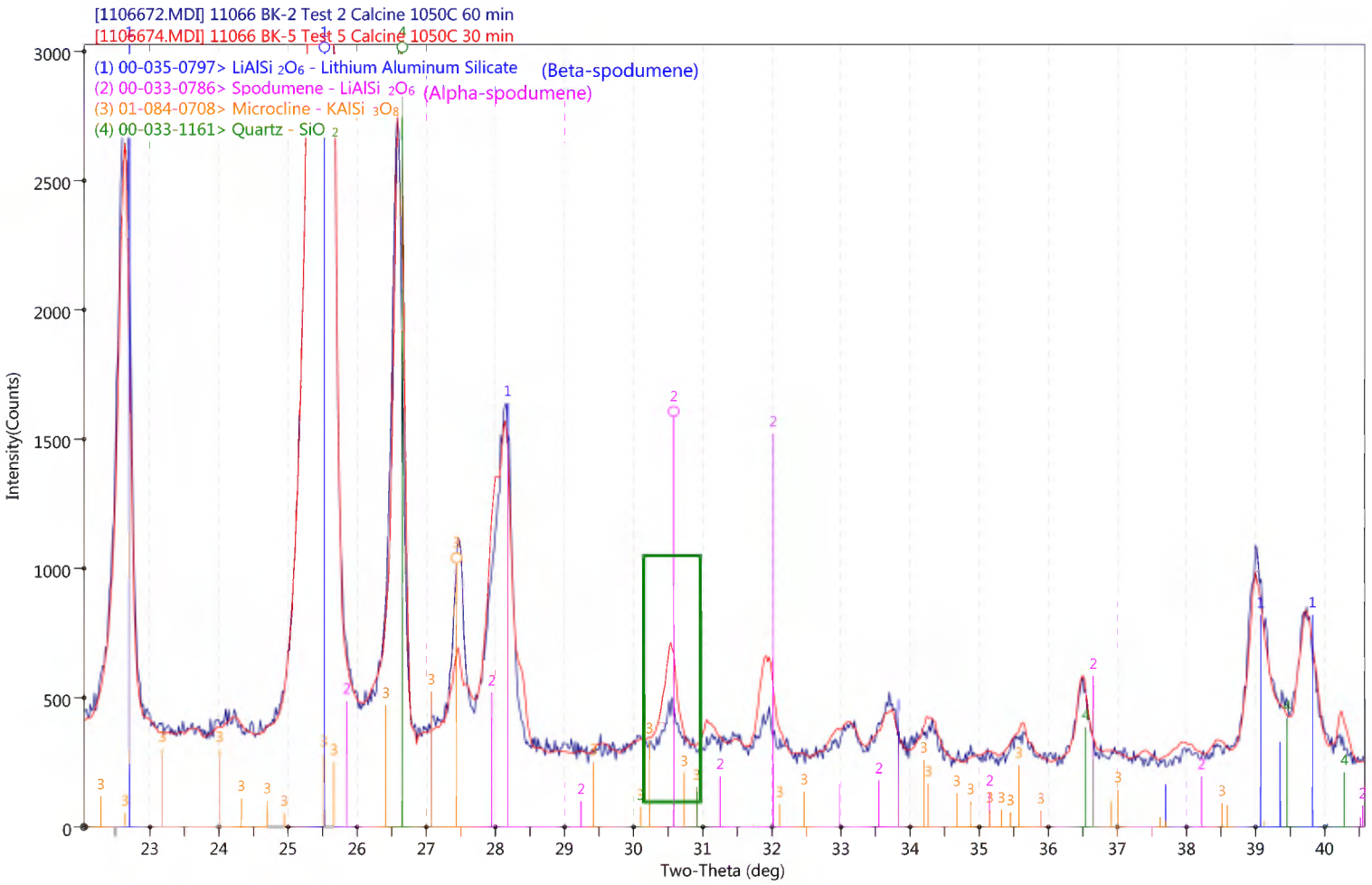


Figure 25. Expanded Area from Figure 24 (enclosed area shows main α-spodumene peak)

Differential thermal analyses were performed on the calcines from Experiments 1, 2, 3, and 5. These thermograms are shown in Appendix H. The calcines produced at higher temperatures show decreased endothermic reactions between 900 and 1,100°C, again indicating less α-spodumene in the samples calcined at higher temperatures.

Experiments 6 and 7 repeated the best conditions of 1,050°C and 60 min to produce additional calcine for the leaching and purification experiments. For the final two calcining experiments, the calcines from Experiments 1, 3, and 5 were combined and reprocessed at a higher temperature to reduce the α-spodumene in the original calcines. The calcines from all of the roasting experiments were blended together for the leaching and purification experiments.

LEACHING, PURIFICATION, AND PRODUCT RECOVERY

The blended calcine from the roasting experiments was used for a series of six acid bake and leaching experiments. Using the best acid bake and leaching conditions, a larger volume of leach liquor was prepared for solution purification and for the recovery of Li_2CO_3 product. These experiments are discussed below, and the data sheets for each experiment are provided in Appendix I.

ACID BAKE AND LEACHING

Seven acid bake and leaching experiments were completed on the calcined ore. For each experiment, the calcined ore was first mixed with concentrated H_2SO_4 and baked in an oven at 250°C for 1 h. After cooling, the acid-baked solids were leached in water at 10% solids and 60°C. For the first six experiments, the pH was maintained between 6.0 and 6.5 during the leach with a 10% lime slurry to minimize aluminum and iron dissolution. The seventh experiment investigated leaching without the addition of lime. For each experiment, the leached slurry was filtered through a Buchner filter and washed with DI water. The solids were dried in an oven and pulverized for analysis. The filtrate, wash, and dried solids were analyzed for lithium to determine the amount of lithium extracted from the ore.

The first three leach experiments (Experiments 1–3) examined the effect of decreasing grind size. For these three experiments, the acid addition was 434 kg/t calcine. The results, summarized in Table 25, indicated little difference in lithium extraction with calcine sizes ranging from as-received calcine at an 80% passing size (P_{80}) of 90 µm to calcine finely ground to a P_{80} of 28 µm. For each of these three experiments, the lithium extraction was 84–87%.

A review of the data from these first three experiments suggested that improved extraction might be obtained with more intimate mixing of the acid with the calcine before the acid bake step. This was tried in Experiment 4, using the same conditions as Experiment 3, except the acid was mixed with the calcined ore for 10 min before acid baking. As shown in Table 25, more intimate mixing of the acid with the calcine increased the lithium extraction from 87 to 92%.

Experiments 5 and 6 investigated increased acid addition as another possible improvement for lithium extraction. Experiment 5 was a repeat of Experiment 4 using the finely ground calcine, except 10% more acid was used. Experiment 6 repeated Experiment 1 with as-received calcine. The effect of additional acid on lithium extraction was pronounced with the coarser calcine, but did not change the lithium extraction of the finer calcine. As shown in Table 25, lithium extractions at the higher acid addition of 478 kg/t were 91% with the as-received calcine and 92% with the finely ground calcine.

Table 25. Summary of Acid Bake and Lithium Leaching Experiments with Lime Addition

Experiment	Sample for Leach	Li in the Calcine, %	H_2SO_4 Addition, kg/t	Wt Gain after Leach, %	Leach Residue, % Li	Li Extraction, %
1	As-received calcine (P_{80} = 90 µm)	2.90	434	25.7	0.323	86
6[a]	As-received calcine (P_{80} = 90 µm)	2.90	478	34.9	0.193	91
2	10-min grind (P_{80} = 74 µm)	2.68	434	31.0	0.333	84
3	2-h grind (P_{80} = 28 µm)	2.84	434	29.2	0.290	87
4[b]	2-h grind (P_{80} = 28 µm)	2.84	434	29.8	0.181	92
5[a]	2-h grind (P_{80} = 28 µm)	2.84	478	37.6	0.155	92

Acid bake conditions: 250°C, 1 h
Leach conditions: 10% solids, pH 6.0–6.5, 30 min, 60°C
[a]10-min acid mix and 10% more acid in acid bake step
[b]10-min acid mix in acid bake step

The lithium extractions from these six experiments plateaued at 92%, indicating that one or more refractory minerals were preventing higher extraction. A mineralogical examination of the leach residue identified the presence of a lithium–aluminum compound. One thought was that this species formed during the leach, with lithium coprecipitating with aluminum when lime was used to maintain the pH at 6.0–6.5. Experiment 7 was completed to determine if lithium extractions could be improved when the leach was conducted in an acid environment without the addition of lime.

For Experiment 7, the acid bake repeated the conditions used in Experiment 6. The acid-baked solids were split in half so that two leach conditions could be investigated. For Experiment 7a, the leach proceeded at 60°C at 30 min, while Experiment 7b was conducted at room temperature. For both of these experiments, no lime was added. Table 26 compares the results of these last two leaches with those of Experiment 6. A weight loss (negative weight gain) was seen in Experiments 7a and 7b because lime was not added and no gypsum formed. The lithium extractions were nearly identical at 91–92% for all three experiments listed in Table 26, indicating that the addition of lime during the leach was not responsible for preventing higher lithium extraction. The nearly equivalent lithium extractions for Experiments 7a and 7b suggest that an elevated temperature may not be needed for the leach.

Table 26. Effect of Lime Addition and Temperature during Lithium Leaching

Experiment	Li in the Calcine, %	H_2SO_4 Addition, kg/t	Leach Temperature, °C	Lime Added to Leach	Wt Gain after Leach, %	Leach Residue, % Li	Li Extraction, %
6	2.90	478	60	yes	34.9	0.193	91
7a	2.90	478	60	no	-7.4	0.059	92
7b	2.90	478	room temp	no	-6.2	0.063	91

As-received calcine (P_{80} = 90 µm)
Acid bake conditions: 250°C, 1 h
Leach conditions: 10% solids, 30 min

XRD Analyses of Acid Bake–Leach Feed and Leach Residues

The flotation concentrate feed to the acid bake–leach and selected leach residues were analyzed by XRD to determine the composition of the specific feed for the experiments and the form of unleached lithium. The feed showed β-spodumene as the only major lithium species, with possibly a trace of α-spodumene. In the residues, a separate unnamed lithium–aluminum silicate was detected, which evidently formed during the acid baking process. In this phase, the 100 intensity line is slightly offset from the major β-spodumene peak, and the β-spodumene peak at 22.7° 2-theta is missing. The mechanism of this transformation is not clear, and the height of the peak does not appear to be consistent with the high lithium extraction of up to 92%. However, at a residue assay of 0.2% Li, this could translate to perhaps 10% or more of this compound. In addition to this lithium-containing silicate, the patterns show the presence of gypsum in some of the residues. The XRD data are summarized in Table 27, and the individual patterns are shown in Figures 26–32.

Table 27. XRD Results of Leach Residues

Sample ID	Mineral Constituents		
	Major	Minor	Trace
Calcine Composite (3277-71-0)	β-spodumene	Quartz	Microcline α-spodumene
Leach Experiment 1 Residue (3277-72-3)	Li–Al silicate Gypsum	Quartz	Microcline Albite
Leach Experiment 2 Residue (3277-72-6)	Li–Al silicate Gypsum	Quartz	Microcline Albite
Leach Experiment 3 Residue (3277-73-5)	Li–Al silicate	Quartz	β-spodumene α-spodumene Microcline
Leach Experiment 6 Residue (3277-74-6)	Li–Al silicate Gypsum		α-spodumene Microcline
Leach Experiment 7a Residue (3277-74-9)	Li–Al silicate	Quartz	β-spodumene α-spodumene Microcline
Leach Experiment 7b Residue (3277-74-12)	Quartz	Li–Al silicate	β-spodumene α-spodumene Microcline

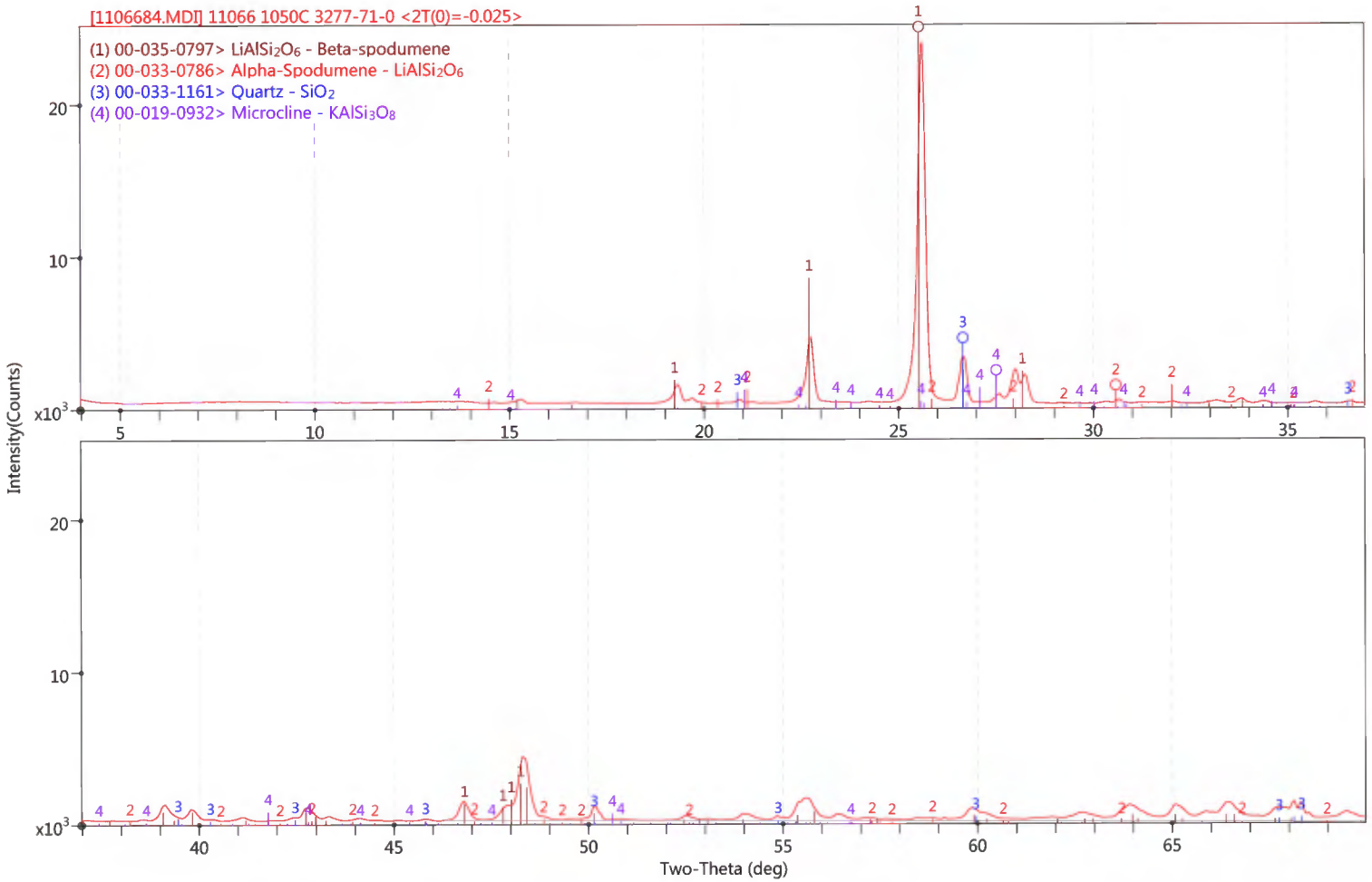


Figure 26. XRD Pattern of Flotation Concentrate (3277-71-0)

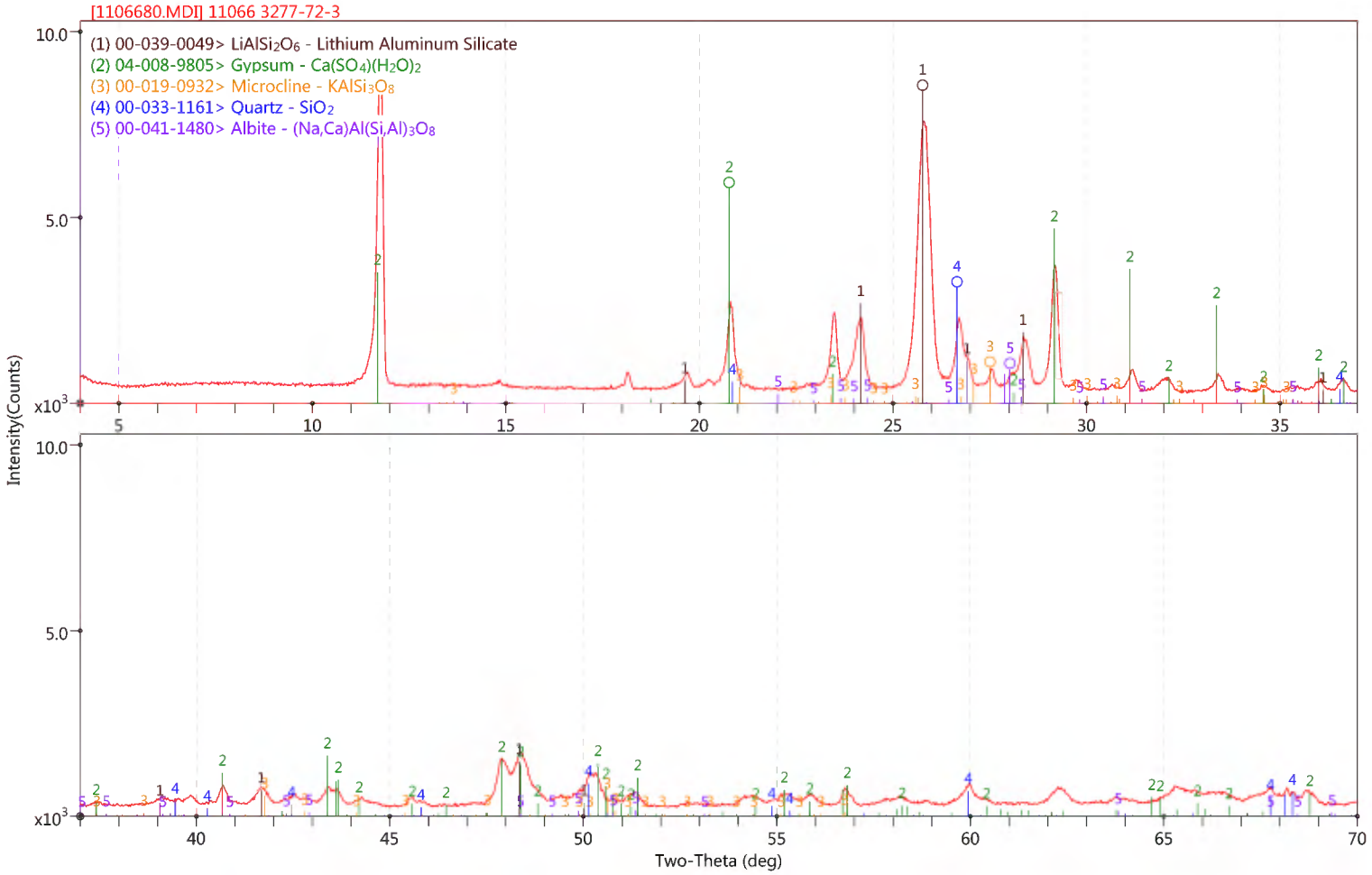


Figure 27. XRD Pattern of Leach Experiment 1 Residue (3277-72-3)

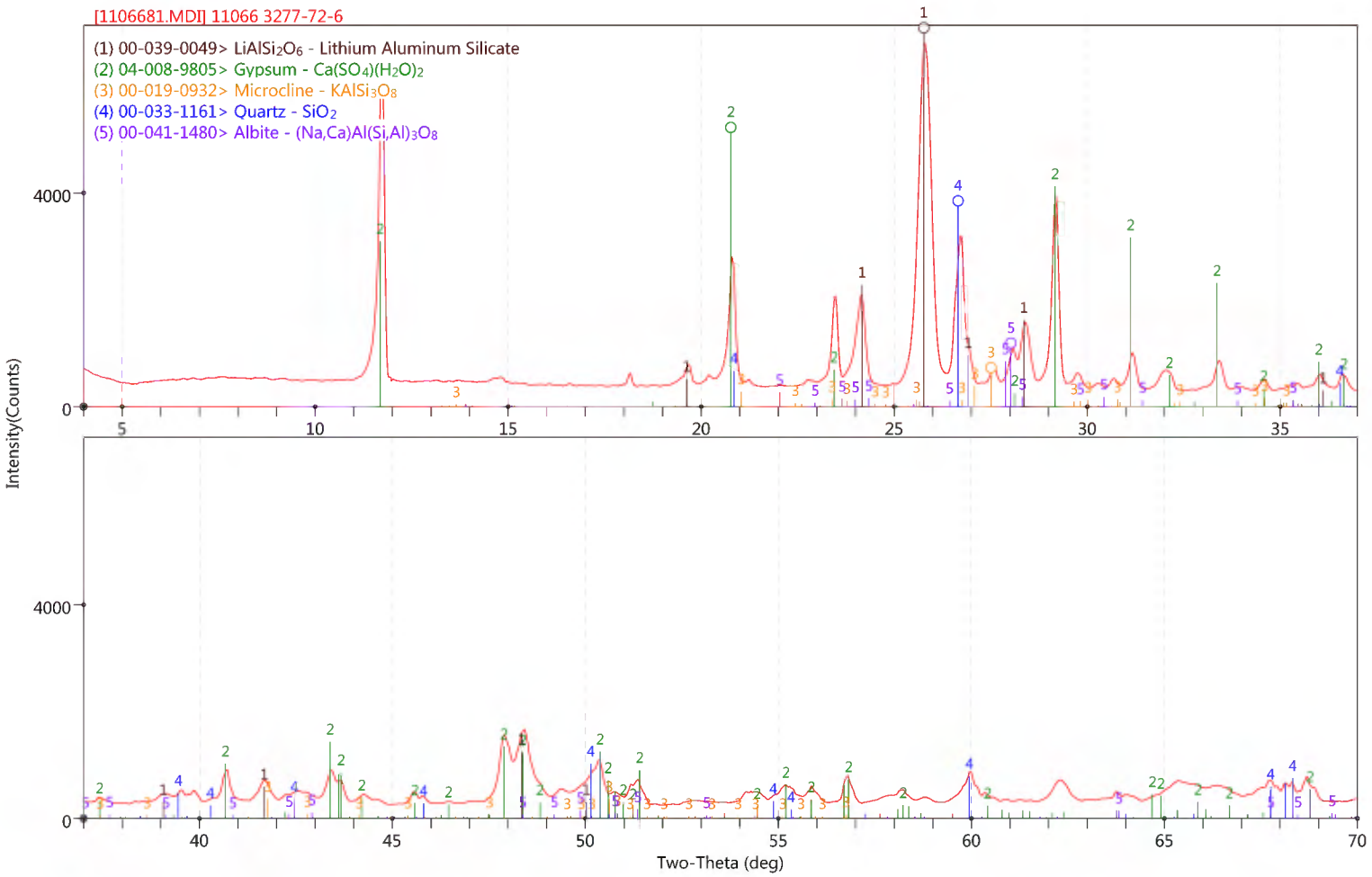


Figure 28. XRD Pattern of Leach Experiment 2 Residue (3277-72-6)

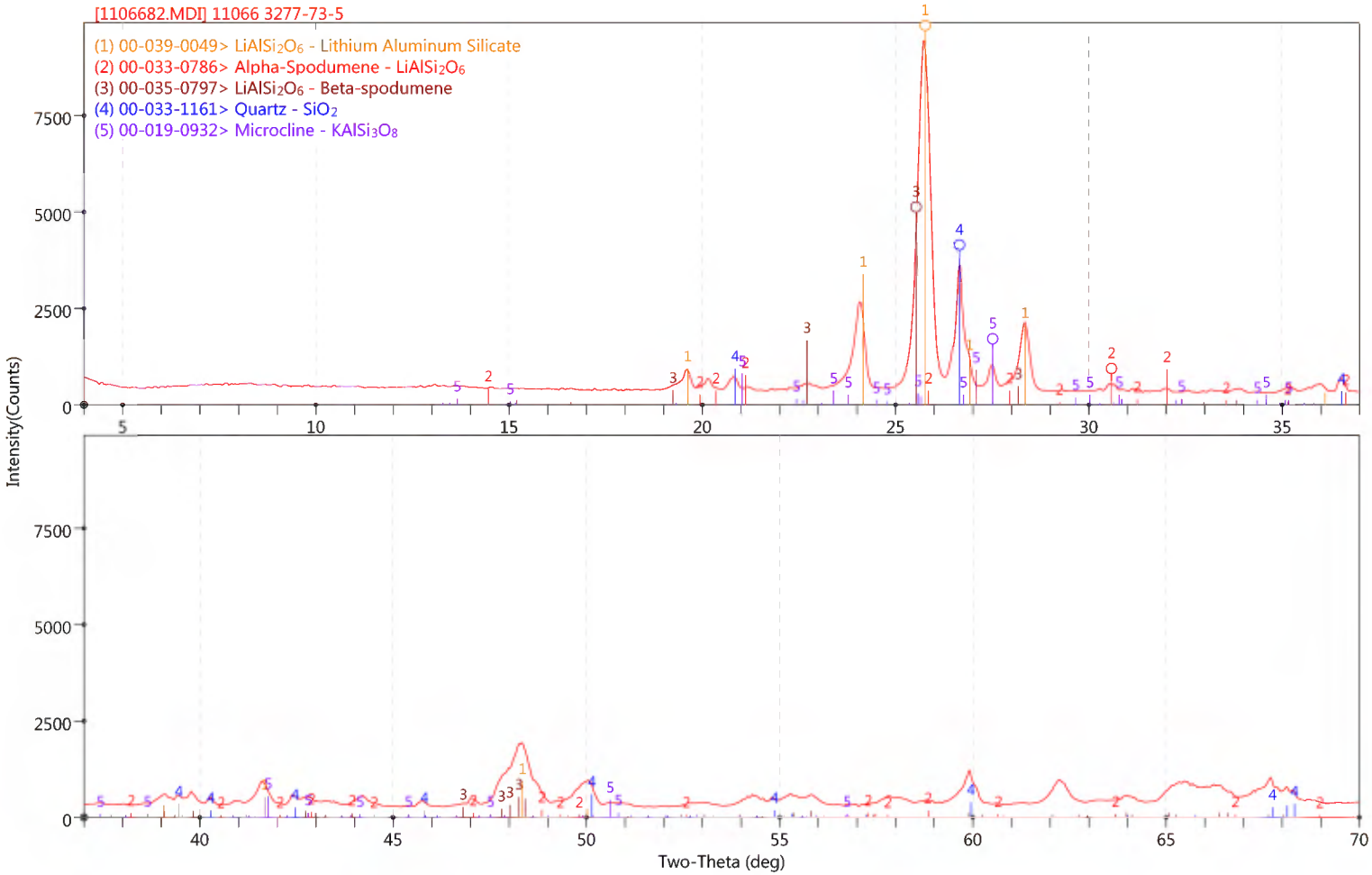


Figure 29. XRD Pattern of Leach Experiment 3 Residue (3277-73-5)

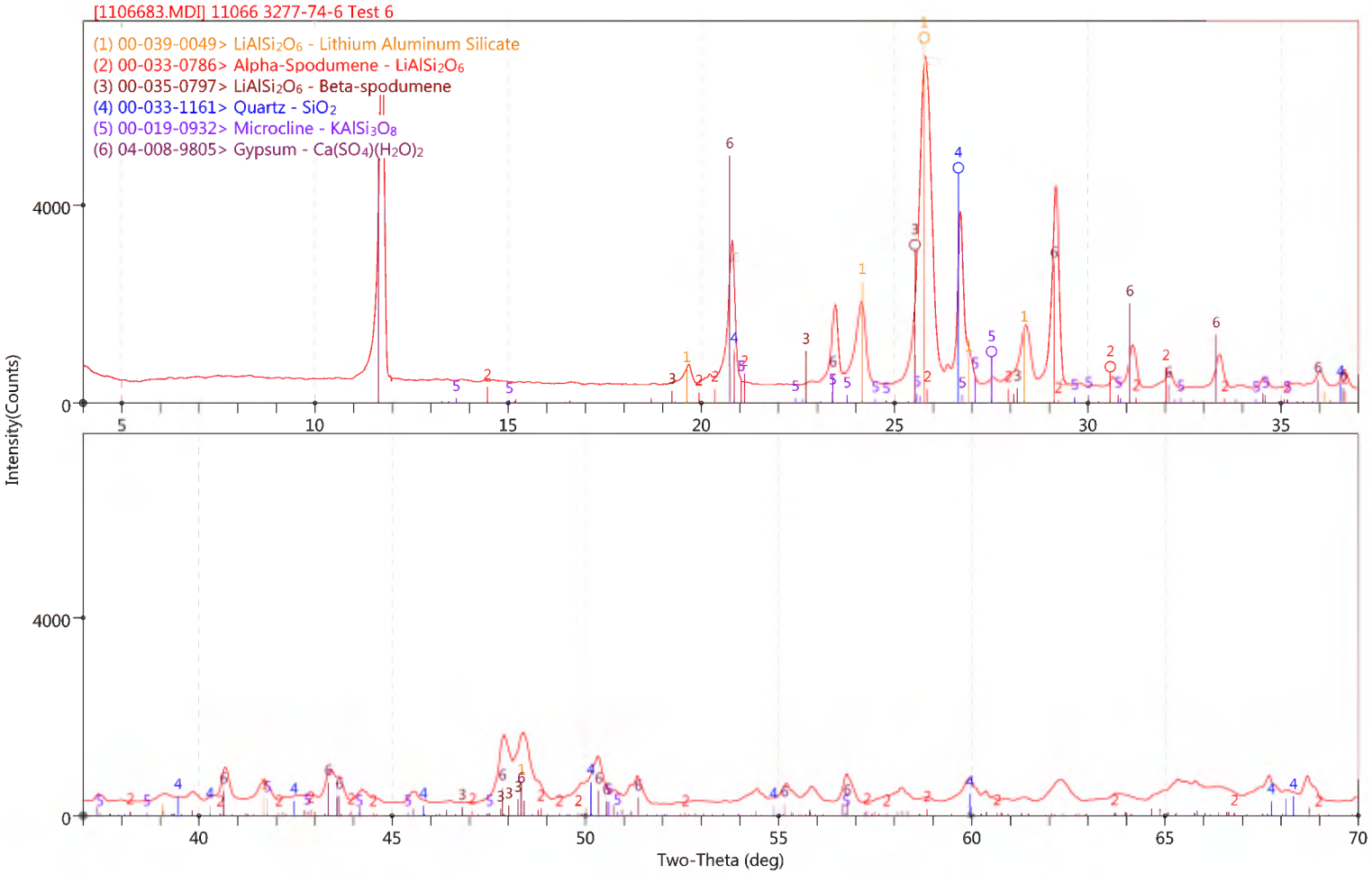


Figure 30. XRD Pattern of Leach Experiment 6 Residue (3277-74-6)

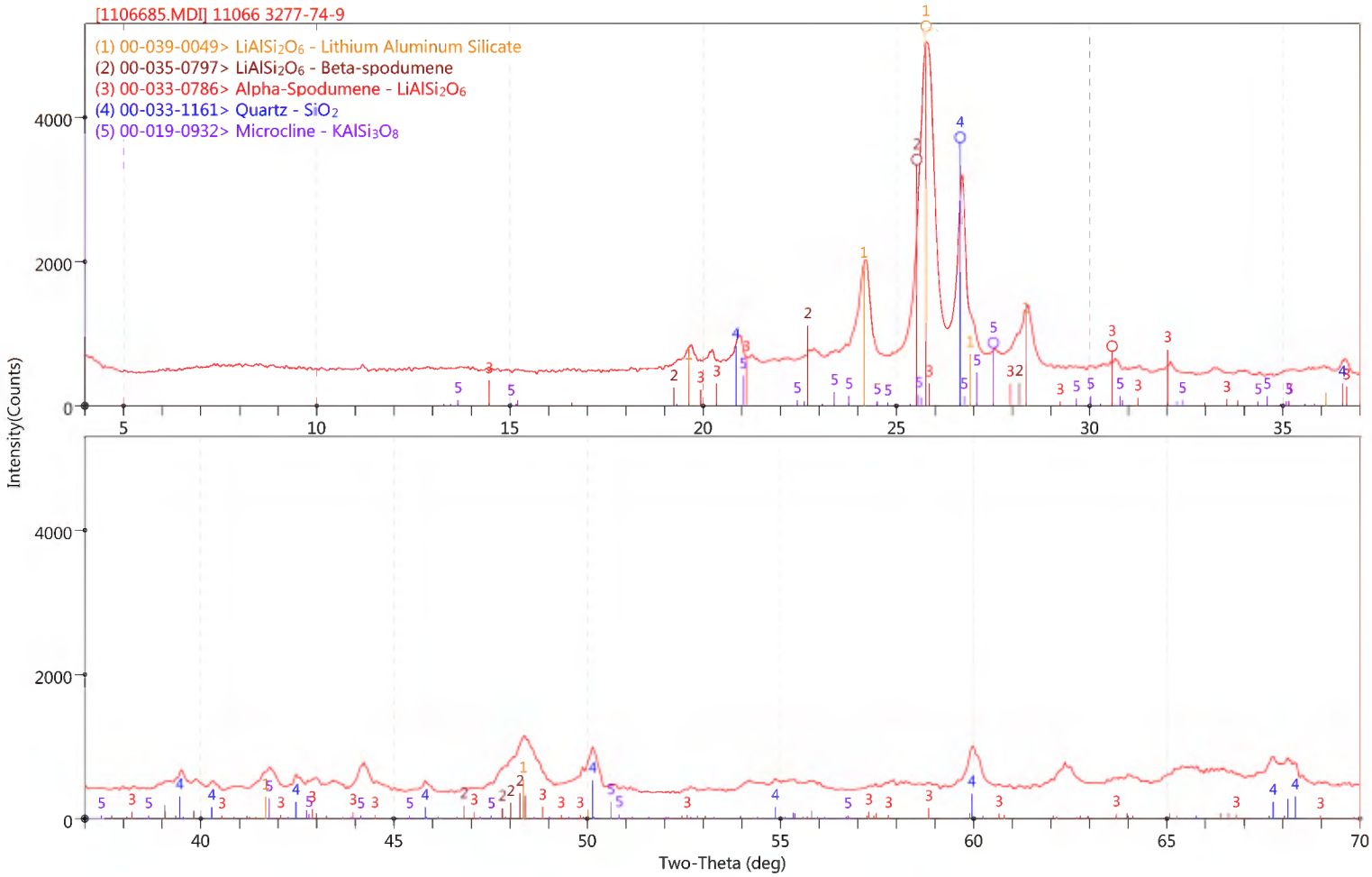


Figure 31. XRD Pattern of Leach Experiment 7a Residue (3277-74-9)

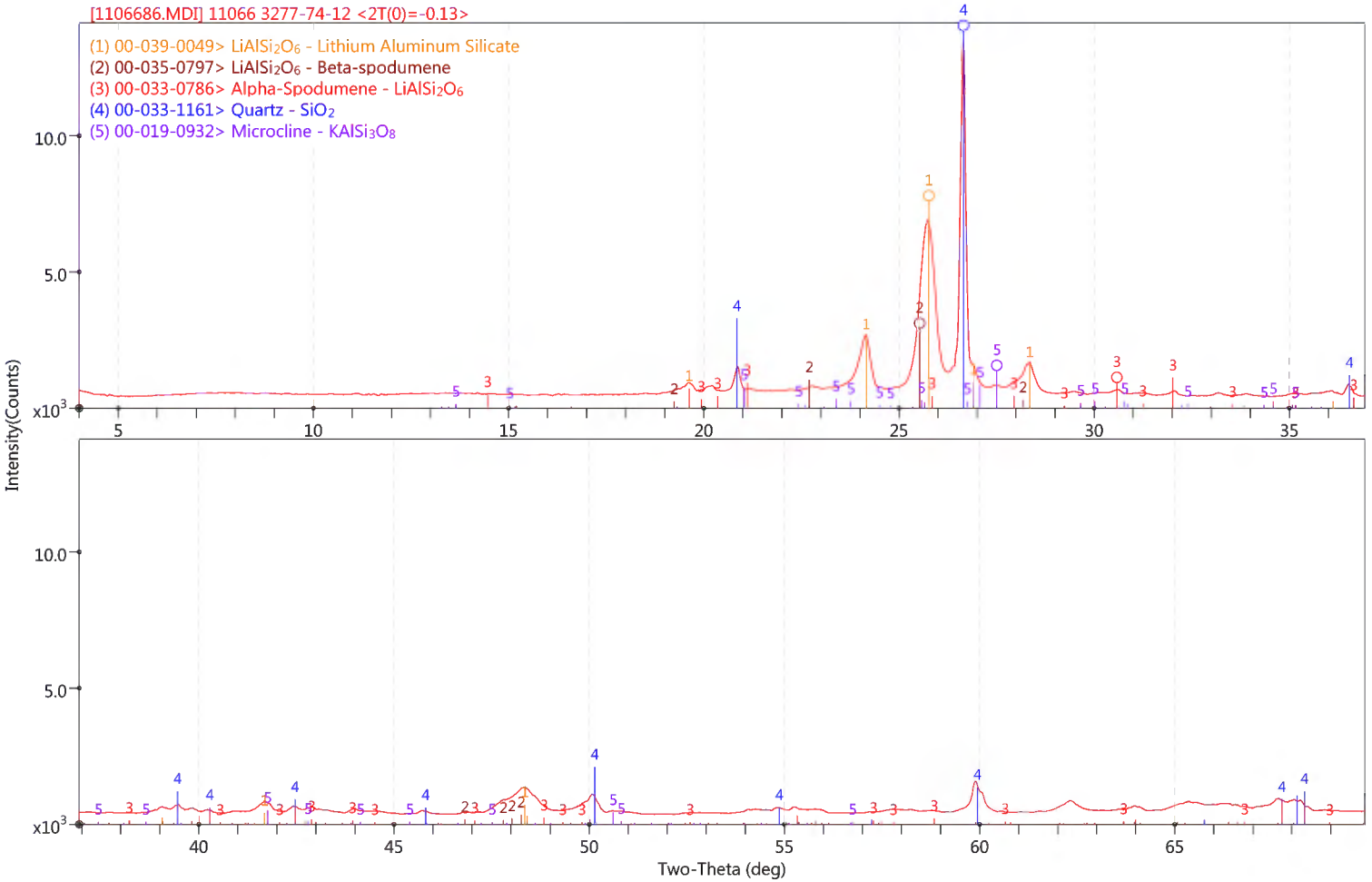


Figure 32. XRD Pattern of Leach Experiment 7b Residue (3277-74-12)

PURIFICATION AND Li_2CO_3 RECOVERY

A larger batch of as-received calcine (200 g) was acid-baked and leached following the same conditions used for Experiment 6. The lime required to maintain the pH between 6 and 6.5 was 197 kg/t. The resulting filtrate was boiled to half the original volume, giving a liquor containing 3.6 g/L Li and 0.093 g/L Mg. By concentrating the solution, the resulting liquor concentrations were more similar to those expected when leaching at a higher solids density, as is anticipated in the commercial operation.

The pH was increased to 11.5 at room temperature with a 10% lime slurry and mixed for 35 min to precipitate magnesium from solution. A negligible amount of lithium was lost, but the resulting liquor contained 0.73 g/L Ca.

The volume of the resulting liquor was again reduced in half by boiling, and the calcium in solution was removed with the addition of Na_2CO_3 as an 18% solution. Less than 0.3% of the lithium was lost in the resulting $CaCO_3$ precipitate. The lithium concentration in the filtrate was 8.0 g/L.

Additional Na_2CO_3 was added as a 24% solution, heating the solution to 90–93°C and mixing for 15 min to precipitate Li_2CO_3. The precipitation was conducted hot because of the inverse solubility of Li_2CO_3. The resulting slurry was centrifuged, and the solids were rinsed with a small amount of hot DI water and dried. The Li_2CO_3 product was analyzed, with the results shown in Table 28. Because Li_2CO_3 is partially soluble, 66% of the lithium in the feed liquor to this step was recovered in the Li_2CO_3 product. In the commercial operation, both the lithium-bearing mother liquor and wash would be recycled upstream in the process to recover additional lithium. A photomicrograph of the Li_2CO_3 product is shown in Figure 33.

Table 28. Analysis of Li_2CO_3 Product

Element	Assay, %
Li	17.1
CO_3^{2-}	72.1
Na	0.650
S	0.43
Mg	0.002
K	0.008
Ca	0.117
Fe	0.006
Al	0.020
Mn	0.001
Cu	nd
Pb	<0.01
Cr	nd
Cl^-	<0.01
PO_4^{3-}	<0.01
B	<0.005
Rb	<0.001

nd = not detected

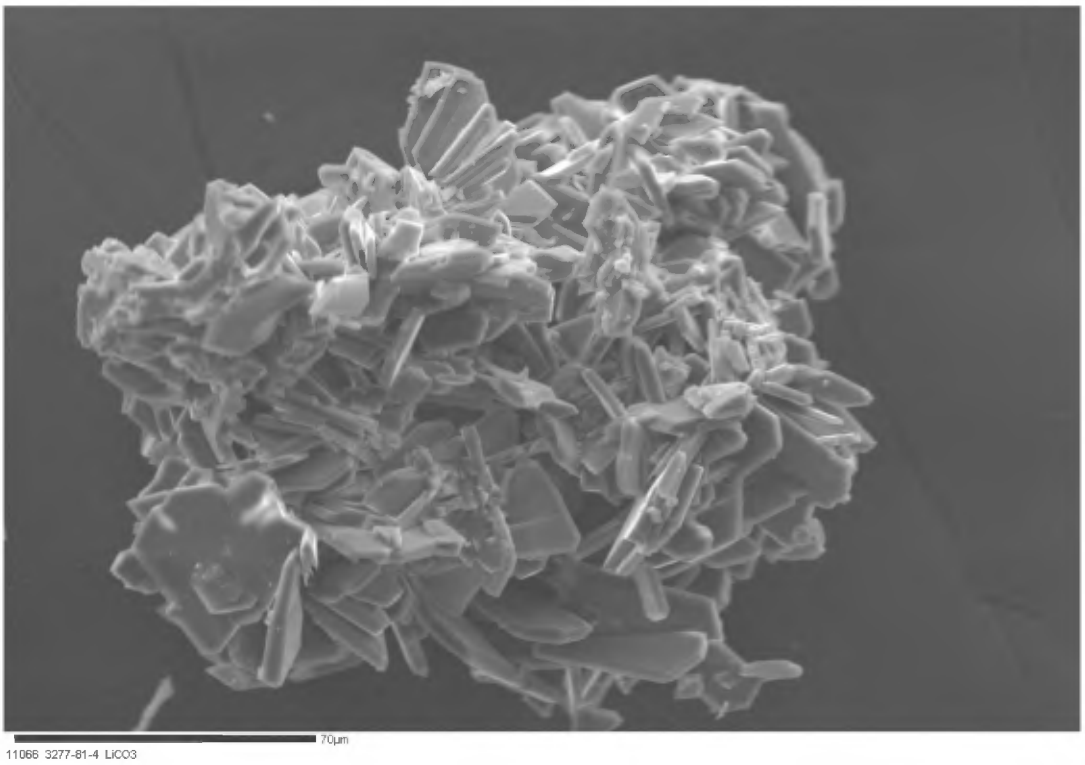


Figure 33. Photomicrograph of a Li_2CO_3 Product Particle

The Li_2CO_3 product analyzed in Table 28 contained appreciable sodium (0.65%) and sulfur (0.43%). These elevated impurity levels are most likely due to insufficient washing of the product, and these concentrations could be substantially reduced with additional wash water. However, additional lithium would be dissolved with more washing, which would require recycling back in the process to recover this lithium. It is expected that the calcium concentration in the final product could be reduced by adding additional Na_2CO_3 in the calcium-removal step. Another alternative to produce a lower-calcium product is to conduct the precipitation in two stages, with the less-pure Li_2CO_3 from the first stage recycled back to the process and the second-stage Li_2CO_3 collected as product.

CONCLUSIONS AND RECOMMENDATIONS

PRELIMINARY PROCESS CONDITIONS AND FLOWSHEET

Based on the data generated during this program, the preliminary process flowsheet for recovering Li_2CO_3 for the James Bay spodumene ore is presented in Figure 34. Briefly, the ore is crushed and ground to 80% passing 300 μm with 1.35 kg/t NaOH, deslimed using a hydrocyclone, and the cyclone underflow is wet-screened at 150 μm. Both fractions are floated separately using Sylfat FA-1 (tall oil fatty acid) collector at 0.75 kg/t in one rougher stage and two stages of cleaning. The concentrates from both circuits are combined and calcined at 1,050°C for 1 h to convert the naturally occurring α-spodumene in the concentrate to acid-soluble β-spodumene. Acid is added to the calcine and baked at 250°C to decompose the β-spodumene into Li_2SO_4, which is subsequently dissolved in the leaching step. Calcium oxide is added to the leach to suppress iron and aluminum solubility. After a solid–liquid separation step, magnesium is removed from solution by adding hydrated lime to precipitate $Mg(OH)_2$. The liquor from the subsequent solid–liquid separation step is concentrated by evaporation, and the calcium is removed by adding Na_2CO_3 to precipitate $CaCO_3$. The resulting liquor is heated to 90–93°C, and Li_2CO_3 is precipitated from solution. Because Li_2CO_3 is partially soluble, the resulting mother liquor contains significant lithium. This stream is recycled back to the lithium recovery step to reclaim additional lithium. A mother liquor bleed is necessary to control impurity buildup.

The process conditions for each step are summarized in Table 29. It should be kept in mind that some of these parameters have not been optimized. Also, the lithium recovery reported in the lithium precipitation step in Table 29 does not take into consideration that recycling the mother liquor from this step back into the process will significantly increase the overall lithium recovery of the process.

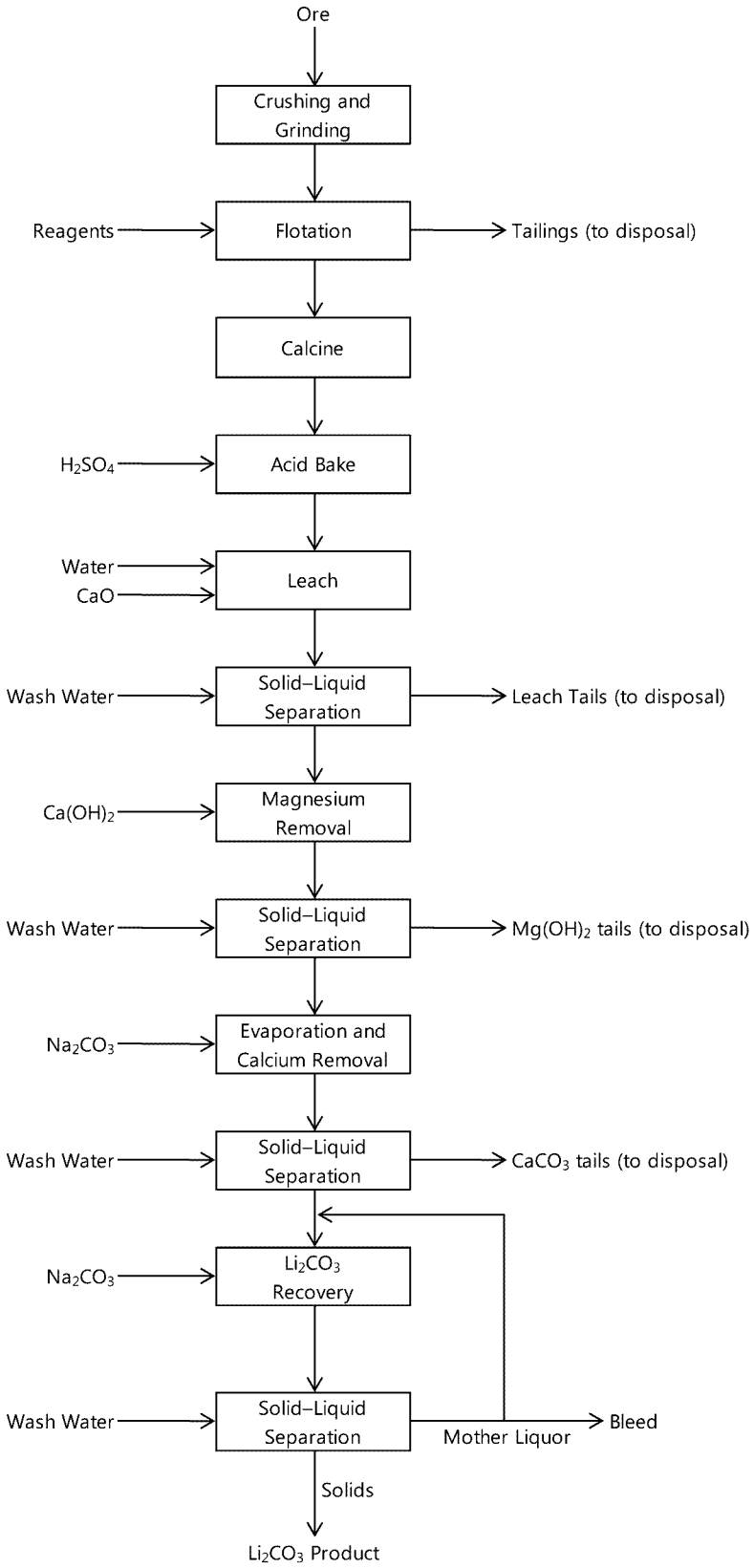


Figure 34. Preliminary Flowsheet for the Recovery of Li_2CO_3 from James Bay Spodumene

Table 29. Processing Conditions for the Recovery of Li_2CO_3 from James Bay Spodumene (Page 1 of 2)

Process Parameter	Value
Flotation	
Ore grind size, P_{80}, µm	300
NaOH added to grind, kg/t	1.35
Grind preparation	hydrocyclone desliming and screen at 150 µm
Desliming	
Mass of slimes rejected, %	11–20
Lithium loss in slimes, %	87–93
Coarse Fraction	
Particle size, µm	plus 150
Collector	Sylfat FA-1
Collector dose, kg/t	0.75
Rougher stages	1
Cleaner stages	2
Spodumene in the concentrate, %	80–85
Spodumene in the tails, %	<1
Fine Fraction	
Particle size, µm	minus 150
Collector	Sylfat FA-1
Collector dose, kg/t	0.75
Rougher stages	1
Cleaner stages	2
Spodumene in the concentrate, %	90
Spodumene in the tails, %	<1
Lithium recovery in the combined concentrate, %	80
Calcining	
Concentrate feed	combined flotation concentrates
Feed grind size	As-received (P_{80} = 90 µm)
Temperature, °C	1,050
Time, min	60
Calcine grind size, P_{80}, µm	90
Conversion of α-spodumene to β-spodumene	nearly complete
Acid Bake	
Feed grind size	As-received calcine
H_2SO_4 requirement, kg/t	478
Temperature, °C	250
Time, min	60

Table 29. Processing Conditions for the Recovery of Li_2CO_3 from James Bay Spodumene (Page 2 of 2)

Process Parameter	Value
Leaching	
Feed grind size	As-received acid-baked solids
Temperature, °C	60[a]
Time, min	30
Solids density, %	10
pH	6.0-6.5
Lime (CaO) requirement, kg/t	140
Weight gain, %	35
Lithium extraction, %	92
Li-to-Mg weight ratio in leach liquor	38.7:1
Magnesium Removal	
Temperature, °C	room temperature
pH	11.5
Hydrated lime ($Ca(OH)_2$) requirement, kg/kg Li	0.225
Ca in the purification liquor, g/L	0.73
Li recovery, %	99.97
Evaporation	
Final Li concentration, g/L	8.0
Calcium Removal	
Temperature, °C	room temperature
Na_2CO_3 requirement, kg/kg Li	0.60
Li recovery, %	99.7
Li_2CO_3 Precipitation	
Temperature, °C	90-93
Na_2CO_3 requirement, kg/kg Li	7.6
Li recovery without recycle, %	66[b]
Li in the mother liquor, g/L	2.3

[a]One experiment indicated that room-temperature leaching may be suitable.

[b]The mother liquor from this step contains significant lithium values. In the commercial operation, this mother liquor is recycled back in the process to recover additional lithium.

RECOMMENDATIONS FOR FURTHER DEVELOPMENT WORK

Additional laboratory work is recommended for a number of the processing steps shown in Figure 34 before proceeding to piloting this process.

- Because of the presence of coarse spodumene crystals, ore sorting should be investigated as a possible preconcentration option that would reduce the crushing–grinding costs and the size of

further downstream processes. A proposal for ore sorting evaluation was prepared by Terra Vision in February 2010; however, no work was done.

- Improvements in the lithium recovery from flotation may be possible with modifications to the flotation operation and should be investigated.

- The calcining conditions have been established in this program. These will need to be verified during continuous processing when the pilot plant proceeds.

- Three parameters for the leach step warrant further evaluation in the next phase of laboratory work.

    - The last leach experiment showed that leaching at room temperature gave similar lithium extractions as those at 60°C. However, lime was not added in this experiment, thus the resulting leach liquor would contain high aluminum and iron concentrations. This experiment should be repeated with lime addition to determine if room-temperature leaching is feasible.

    - The highest lithium extractions in the leach step were 92%, due to the presence of a lithium–aluminum silicate species. Further work is necessary to determine if this species is present in the ore or is formed during processing. Options for either preventing the formation of this species or conditions that allow it to be leached should be investigated to improve lithium extraction.

    - All of the leach tests in this program were performed at 10% solids. This resulted in rather dilute liquors that required evaporation to concentrate the solutions. Higher solids densities in the leach step should be evaluated to determine if leach liquors can be produced at higher lithium concentrations without adversely affecting lithium extraction.

- The purity of the Li_2CO_3 product made in this program needs to be evaluated by Lithium One. If this evaluation indicates that lower impurity levels are required, additional purification steps will need to be evaluated in the laboratory.

- The present program did not evaluate recycling lithium-bearing mother liquor from the Li_2CO_3 precipitation step. This will require further evaluation in the laboratory to determine solution impurity buildup and the bleed requirement to maintain product purity. The mother liquor may require evaporation before returning this stream to the process.

- Tracking and deportment of impurities through the process was not included in this current study. This should be undertaken in the next laboratory program.

- The solid–liquid separation steps in the flowsheet have not been evaluated. Laboratory tests to determine the appropriate solid–liquid separation technologies for each step are required. This would include flocculant screening, Kynch settling tests, and standard filtration tests for each step, as appropriate.

The results of this additional laboratory program, along with the data generated from this current work, would be used to design and operate a pilot plant to demonstrate the process on a continuous basis and to provide engineering data to scale the process to commercial size.

APPENDIX A

Sample Identification

Sample 1A Split Core

HRI	Client ID		Interval received	Weight, kg	Comments
	Dike	Interval			
52461-1	7.2	753022–753027	753022–753023	9.8	no 753024
			753025–753027	8.6	
		753037–753045	753037–753039	8.8	
			753041–753043	10.8	
			753044–753045	3.2	
		753068–753072	753068–753072	9.8	
		753085–753087	753085–753087	8.9	
		753209–753211	753209–753211	7.5	
		753309–753313	753309–753313	10.3	
		753407–753413	753407–753409	8.6	
			753411–753413	7.3	
		753269–753271	753269–753271	4.4	
		753366–753369	753366–753369	10.6	
		753461–753465	753461–753463	7.1	
			753464–753465	4.4	
Total				**120.1**	
52461-2	7.6	753142–753148	753142–753144	9.5	
			753145–753148	12.3	
		753229–753239	753229–753232	7.3	
			753233–753235	7.4	
			753236–753239	10.7	
		753326–753333	753326–753329	11.8	
			753331–753333	6.8	
		753423–753426	753423–753426	11.8	
		753534–753541	753534–753536	8.9	
			753537–753541	9.2	
		279576–279581	279576–279577	6.3	
			279578–279581	6.2	
		279672–279675	279672–279675	10.0	
		279728–279734	279728–279730	5.4	
			279731–279734	9.8	
		279851–279854	279851–279854	8.5	
Total				**141.9**	
52461-3	8.3	279562–279565	279562–279565	10.4	
		279606–279615	279606–279608	8.3	
			279609–279612	9.8	
			279613–279615	10.6	
		279654–279657	279654–279657	9.8	
		279756–279763	279756–279758	8.8	
			279759–279763	12.4	
		279823–279834	279823–279825	8.1	
			279826–279829	12.3	
			279831–279834	9.4	
		279891–279893	279891–279893	7.4	
		279991–279994	279991–279994	8.1	
		105–107	105–107	7.0	
Total				**122.4**	

Sample 1A Split Core

HRI	Client ID		Interval received	Weight, kg	Comments
	Dike	Interval			
52461-4	8.7	124–136	124–126	8.8	
			128–130	6.4	
			131–133	13.2	
			134–136	8.6	
		618–632	618–620	11.9	
			621–623	6.3	
			624–626	9.4	
			627–629	9.8	
			630–632	9.3	
		658–659	658–659	4.7	
		234–237	234–237	12.7	
		331–349	331–333	8.4	
			334–336	10.2	
			337–339	10.1	
			340–342	10.2	
			344–346	8.2	no 343
			347–349	6.9	
Total				**155.1**	
52461-5	9.2	171–175	171–173	7.4	
			174–175	4.6	
		245–254	245–248	8.5	
			249–251	6.2	
			252–254	5.9	
		255–257	255–257	6.1	
		354–357	354–357	11.8	
		399–402	399–402	12.8	
		465–476	465–467	6.7	
			468–470	10.1	
			471–473	9.7	
			474–476	8.0	
		1520–1522	1520–1522	8.7	
Total				**106.5**	
52461-6	10.4	1545–1546	1545–1546	4.6	
		1614–1618	1614–1618	10.8	
		1741–1745	1741–1743	8.1	
			1744–1745	3.8	
		1893–1897	1893–1895	9.1	
			1896–1897	9.1	
		2454–2461	2454–2457	12.6	
			2458–2461	12.6	
		2063–2068	2063–2065	9.4	
			2066–2068	9.3	
		2074–2079	2074–2076	9.6	
			2077–2079	9.0	
Total				**108.0**	
52461-7	11.2 (11.4 included)	1597–1609	1597–1599	11.4	
			1600–1602	10.6	
			1603–1606	10.9	
			1607–1609	8.6	
		1610–1613	1610–1613	10.4	
		1663–1677	1663–1665	10.2	
			1666–1669	8.2	
			1669–1671	7.9	
			1672–1674	9.2	
			1675–1677	7.1	
		1727–1735	1727–1730	9.1	
			1731–1733	9.9	
			1734–1735	5.2	
		1834–1848	1834–1836	8.6	
			1837–1840	12.9	
			1841–1843	10.4	
			1844–1848	13.2	
Total				**163.8**	

Sample 1A Split Core

HRI	Client ID		Interval received	Weight, kg	Comments
	Dike	Interval			
52461-8	12.2	2468–2482	2468–2470	8.5	
			2472–2474	10.0	no 2471
			2475–2477	9.8	
			2478–2482	12.6	
		2501–2516	2501–2504	8.5	
			2505–2507	9.7	
			2508–2510	10.1	
			2511–2513	6.1	
			2514–2516	9.1	
		2342–2352	2342–2344	8.5	
			2345–2349	11.9	
			2350–2352	9.2	
		2382–2386	2382–2384	7.9	
			2385–2386	4.4	
Total				**126.3**	
52461-9	13.2	1181–1184	1181–1184	14.6	
		1302–1310	1302–1304	8.6	
			1305–1307	9.0	
			1308–1309	5.7	no 1310
		2207–2235	2207–2210	11.7	
			2212–2215	12.3	
			2216–2218	9.9	
			2220–2222	12.6	no 2219
			2223–2225	9.3	
			2226–2228	10.0	
			2231–2233	9.9	no 2229 or 2230
			2233–2235	7.2	
Total				**120.8**	
52461-10	14.2	1089–1107	1089–1093	12.8	
			1093–1096	9.8	
			1097–1099	9.8	
			1100–1103	13.3	
			1104–1107	12.1	
		1198–1216	1198–1200	9.1	
			1201–1203	9.6	
			1204–1207	9.8	
			1208–1210	10.0	
			1211–1213	10.2	
			1214–1216	7.6	
		2118–2127	2118–2120	6.8	
			2121–2123	9.6	
			2125–2127	13.3	no 2124
Total				**143.8**	
52461-11	15.1		842–845[a]	10.1	
		859–893	859–861	9.9	
			862–863	8.5	
			864–867	7.6	
			868–870	9.6	
			871–873	9.3	
			874–877	9.6	
			878–881	11.4	
			882–884	11.2	
			885–887	9.6	
			888–890	9.9	
			891–893	9.7	
		1036–1047	1036–1038	9.6	
			1039–1045?	9.2	not labeled with 1045
			1046–1047	5.9	
Total				**141.1**	

[a]not on client data sheet

Sample 1B Assay Reject

HRI	Client ID		Received	Weight, kg	Comments
	Dike	Interval			
52346-1	7.2	753022–753027	753022	3.6	
			753023	3.1	
			753024	3.4	
			753025	3.2	
			753026	3.1	
			753027	1.9	
		753037–753045	753037	1.7	
			753038	3.1	
			753039	3.1	
			753041	3.0	
			753042	2.8	
			753043	2.7	
			753044	1.3	
			753045	0.9	
		753068–753072	753068	2.1	wet
			753069	3.1	wet
			753071	3.1	wet
			753072	2.2	
		753085–753087	753085	1.8	
			753086	3.0	
			753087	2.9	
		753209–753211	753209	3.1	wet
			753211	3.4	
		753309–753313	753309	1.5	
			753311	3.0	
			753312	3.1	
			753313	1.8	
		753407–753413	753407	2.2	
			753408	3.4	wet
			753409	3.1	
			753411	2.6	wet
			753412	2.5	
			753413	2.2	
		753269–753271	753269	1.6	
			753271	2.8	
		753366–753369	753366	2.3	
			753367	3.3	wet
			753368	2.2	
			753369	2.1	
		753461–753465	753461	2.0	wet
			753462	2.6	
			753463	2.6	
			753464	2.0	
			753465	2.1	
Total				**112.4**	

Sample 1B Assay Reject

HRI	Client ID		Received	Weight, kg	Comments
	Dike	Interval			
52346-2	7.6	753142–753148	753142	2.9	
			753143	3.0	
			753144	3.1	
			753145	3.0	
			753146	3.2	
			753147	3.6	
			753148	2.1	
		753229–753239	753229	1.4	
			753231	3.0	
			753232	2.7	
			753233	3.0	
			753234	3.0	
			753235	3.2	wet
			753236	3.2	wet
			753237	3.1	
			753238	1.9	
			753239	2.0	
		753326–753333	753326	2.0	
			753327	3.2	wet
			753328	3.1	
			753329	3.2	
			753331	2.6	
			753332	2.2	
			753333	2.1	
		753423–753426	753423	2.1	
			753424	3.2	wet
			753425	3.2	wet
			753426	3.0	wet
		753534–753541	753534	2.2	
			753535	3.1	
			753536	3.1	
			753537	3.2	
			753538	2.0	wet
			753539	1.5	
			753541	2.5	
		279576–279581	279576	3.2	
			279577	3.2	
			279578	2.2	wet
			279579	1.8	
			279581	2.1	
		279672–279675	279672	1.9	
			279673	3.1	
			279674	2.2	
			279675	1.9	
		279728–279734	279728	2.0	
			279729	3.0	
			279731	2.9	
			279732	1.6	
			279733	1.7	
			279734	1.9	
		279851–279854	279851	1.0	
			279852	3.3	
			279853	1.9	
			279854	2.1	
Total				**137.8**	

Sample 1B Assay Reject

HRI	Client ID		Received	Weight, kg	Comments
	Dike	Interval			
52346-3	8.3	279562–279565	279562	3.1	
			279563	2.6	
			279564	2.1	
			279565	2.1	
		279606–279615	279606	3.2	wet
			279607	3.2	wet
			279608	3.2	wet
			279609	3.1	wet
			279611	3.0	wet
			279612	3.1	wet
			279613	3.1	wet
			279614	3.2	wet
			279615	2.4	wet
		279654–279657	279654	2.0	
			279655	2.2	
			279656	2.6	
			279657	2.1	
		279756–279763	279756	2.0	
			279757	3.1	
			279758	3.2	
			279759	3.2	
			279761	3.1	
			279762	3.0	
			279763	2.7	
		279823–279834	279823	1.9	
			279824	3.1	
			279825	3.1	
			279826	1.2	
			279827	3.1	
			279828	3.2	
			279829	2.6	wet
			279831	3.3	
			279832	2.5	
			279833	1.1	
			279834	2.2	
		279891–279893	279891	1.8	
			279892	3.1	
			279893	0.7	
		279991–279994		0.0	not received
		105–107	0105	2.2	
			0106	3.3	
			0107	2.4	
Total				**107.3**	

Sample 1B Assay Reject

HRI	Client ID		Received	Weight, kg	Comments
	Dike	Interval			
52346-4	8.7	124–136	0124	2.1	
			0125	3.1	
			0126	3.1	
			0128	3.2	
			0129	2.9	
			0130	2.9	
			0131	3.2	
			0132	2.8	
			0133	2.7	
			0134	2.8	
			0135	2.7	
			0136	2.2	
		618–632	0618	3.0	
			0619	2.9	
			0620	2.9	
			0621	3.0	
			0622	3.2	
			0624	3.0	
			0625	3.1	
			0626	3.1	
			0627	3.3	
			0628	2.9	
			0629	2.8	
			0630	2.9	
			0631	3.0	
			0632	3.0	
		658–659	0658	2.3	
			0659	2.1	
		234–237	0234	3.2	
			0235	3.0	
			0236	3.1	
			0237	3.1	
		331–349	0331	3.0	
			0332	2.9	
			0334	3.0	
			0335	3.0	
			0336	3.1	
			0337	3.1	
			0338	3.0	
			0339	3.1	
			0340	3.2	
			0341	3.0	
			0342	3.0	
			0343	3.0	
			0345	2.0	
			0346	1.7	
			0347	2.2	
			0348	1.9	
			0349	1.9	
Total				**138.2**	

Sample 1B Assay Reject

HRI	Client ID		Received	Weight, kg	Comments
	Dike	Interval			
52346-5	9.2	171–175	0171	2.2	
			0172	2.2	
			0173	2.3	
			0174	2.6	
			0175	2.1	
		245–254	0245	2.1	
			0246	3.4	
			0247	3.1	
			0249	3.3	
			0250	3.3	
			0252	1.6	
			0253	1.6	
			0254	2.0	
		255–257	0255	2.0	
			0256	1.7	
			0257	1.9	
		354–357	0354	2.5	
			0355	1.0	
			0356	3.0	
			0357	3.1	
		399–402	0399	3.0	
			0400	3.0	
			0401	3.0	
			0402	3.1	
		465–476	0465	3.0	
			0466	3.0	
			0468	2.9	
			0469	2.9	
			0470	3.0	
			0471	3.1	
			0472	3.2	
			0473	2.9	
			0474	2.9	
			0475	3.4	
			0476	2.0	
		1520–1522	1520	0.0	missing
			1521	0.0	missing
			1522	0.0	missing
Total				**91.4**	

Sample 1B Assay Reject

HRI	Client ID		Received	Weight, kg	Comments
	Dike	Interval			
52346-6	10.4	1545–1546	1545	2.1	
			1546	2.1	
		1614–1618	1614	2.2	
			1615	3.3	
			1616	0.0	missing
			1617	3.0	
			1618	2.0	
		1741–1745	1741	2.1	
			1742	3.0	
			1743	3.0	
			1744	1.4	
			1745	1.8	
		1893–1897	1893	2.8	
			1894	3.2	
			1895	3.1	
			1896	3.2	
			1897	3.1	
		2454–2461	2454	2.1	
			2455	3.2	wet
			2456	3.2	
			2457	3.2	
			2458	3.1	
			2459	3.2	
			2460	3.3	
			2461	0.8	wet
		2063–2068	2063	3.3	
			2064	3.2	
			2065	2.5	
			2066	2.7	
			2067	3.0	
			2068	3.0	
		2074–2079	2074	3.1	
			2075	3.0	
			2076	2.9	
			2077	3.3	
			2078	3.2	
			2079	2.5	
Total				**99.1**	

Sample 1B Assay Reject

HRI	Client ID		Received	Weight, kg	Comments
	Dike	Interval			
52346-7	11.2 (11.4 included)	1597–1609	1597	2.9	
			1598	3.1	
			1599	3.2	
			1600	2.9	
			1601	2.9	
			1602	3.0	
			1603	3.1	wet
			1604	2.9	
			1606	3.0	
			1607	3.1	
			1608	2.4	
			1609	1.9	
		1610–1613	1610	0.0	missing
			1611	3.0	
			1612	2.5	
			1613	2.1	
		1663–1677	1663	1.9	
			1664	3.0	
			1665	3.2	
			1666	3.2	
			1668	3.1	
			1669	3.2	
			1670	2.9	
			1671	3.1	
			1672	2.8	
			1673	3.0	
			1674	3.0	
			1675	1.9	
			1676	2.0	
			1677	0.8	
		1727–1735	1727	1.8	
			1728	0.0	missing
			1729	2.4	
			1730	2.8	
			1731	3.1	
			1732	2.8	
			1733	2.7	
			1734	2.5	
			1735	1.5	
		1834–1848	1834	2.0	
			1835	3.1	
			1836	3.1	
			1837	3.2	
			1838	3.0	
			1839	3.0	
			1840	3.0	
			1841	1.5	
			1842	2.7	
			1843	3.4	wet
			1844	3.2	
			1845	3.0	
			1846	2.0	
			1847	2.1	
			1848	2.0	
Total				**138.4**	

Sample 1B Assay Reject

HRI	Client ID		Received	Weight, kg	Comments
	Dike	Interval			
52346-8	12.2	2468–2482	2468	1.7	
			2469	2.9	
			2470	3.0	
			2471	0.0	missing
			2472	2.9	
			2473	3.3	
			2474	2.9	
			2475	2.4	
			2476	2.8	
			2477	2.9	
			2478	2.8	
			2479	3.1	
			2481	3.0	
			2482	3.0	
		2501–2516	2501	2.0	
			2503	3.1	
			2504	3.2	
			2505	2.7	
			2506	2.3	
			2507	3.0	
			2508	3.0	
			2509	2.9	
			2510	3.1	
			2511	2.9	
			2513	2.9	indicated excluded
			2514	3.4	
			2515	3.0	
			2516	2.4	
		2342–2352	2342	1.6	
			2343	2.8	
			2344	2.7	wet
			2345	2.5	
			2346	3.1	
			2347	2.7	
			2348	2.9	
			2350	2.9	
			2351	2.3	
			2352	2.0	
		2382–2386	2382	3.1	
			2383	2.9	
			2384	1.6	
			2385	2.0	
			2386	2.0	
Total				**113.3**	

Sample 1B Assay Reject

HRI	Client ID		Received	Weight, kg	Comments
	Dike	Interval			
52346-9	13.2	1181–1184	1181	2.6	
			1182	3.0	
			1183	3.0	
			1184	3.1	
		1302–1310	1302	1.9	
			1303	3.0	
			1304	3.0	
			1305	3.1	
			1306	3.0	
			1307	2.9	
			1308	0.9	
			1309	2.0	
			1310	2.9	
		2207–2235	2207	2.0	
			2208	3.0	
			2209	3.1	
			2210	3.1	
			2212	3.1	
			2213	3.1	
			2214	3.1	
			2215	3.3	
			2216	3.1	
			2217	3.1	
			2218	3.1	
			2219	3.1	
			2220	3.2	
			2221	0.0	missing
			2222	0.0	missing
			2223	0.0	missing
			2224	0.0	missing
			2225	0.0	missing
			2226	0.0	missing
			2227	0.0	missing
			2228	0.0	missing
			2229	0.0	missing
			2230	0.0	missing
			2231	3.1	
			2232	3.0	
			2233	3.2	
			2234	1.8	
			2235	1.8	
Total				**86.8**	

Sample 1B Assay Reject

HRI	Client ID		Received	Weight, kg	Comments
	Dike	Interval			
52346-10	14.2	1089–1107	1089	3.1	
			1090	2.9	
			1091	3.0	
			1092	2.7	
			1093	2.9	
			1095	3.1	
			1096	2.9	
			1097	2.9	
			1098	2.9	
			1099	2.8	
			1100	3.0	
			1101	3.0	
			1102	2.5	
			1103	2.9	
			1104	1.2	
			1105	2.9	
			1106	2.6	
			1107	1.6	
		1198–1216	1198	1.6	
			1199	3.2	
			1200	2.8	
			1201	3.0	
			1202	2.6	
			1203	2.9	
			1204	3.1	
			1205	3.1	
			1206	3.1	
			1208	3.1	
			1209	3.1	
			1210	3.0	
			1211	2.9	
			1212	3.2	
			1213	3.2	
			1214	3.2	
			1215	2.4	
			1216	2.1	
		2118–2127	2118	3.0	
			2120	2.7	
			2121	2.9	
			2122	3.1	
			2123	3.2	
			2124	2.9	
			2125	3.0	
			2126	3.0	
			2127	3.0	
Total				**127.1**	

Sample 1B Assay Reject

HRI	Client ID		Received	Weight, kg	Comments
	Dike	Interval			
52346-11	15.1	859–893	859	3.1	
			860	3.0	
			861	2.8	
			862	3.3	
			863	2.9	
			864	3.1	
			866	3.1	
			867	2.9	
			868	2.9	
			869	2.8	
			870	2.5	
			871	3.2	
			872	2.9	
			873	3.0	
			874	3.2	
			876	2.9	
			877	2.9	
			878	2.9	
			879	3.0	
			880	3.0	
			881	3.0	
			882	3.1	
			883	3.1	
			884	3.7	
			885	1.3	
			886	3.1	
			887	2.9	
			888	2.9	
			889	2.9	
			890	3.0	
			891	3.0	
			892	3.0	
			893	3.0	
		1036–1047	1036	3.2	
			1037	3.2	
			1038	3.2	
			1039	3.0	
			1040	3.3	
			1041	3.3	
			1042	2.9	
			1044	2.9	
			1045	3.1	
			1046	3.0	
			1047	1.9	
Total				**130.1**	

APPENDIX B

Sample Characterization

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO. 2148-43
LOCATION

DYKE NO.	Dyke 8.3	Dyke 8.3	Dyke 8.3
SAMPLE IDENTIFICATION	279562-279565	279562-279565	279562-279565
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10	2/19/10	2/19/10
TEST DATA			
Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	437.09	378.98	396.10
Wt. of Soaked Sample in Distilled Water (gms)	284.60	238.90	252.00
Wt. of Dried Sample (gms)	436.55	378.29	395.51
SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS			
Bulk Specific Gravity	2.86	2.70	2.74
Absorption, weight (%)	0.12	0.18	0.15
Average Absorption, weight (%)	0.15		
Average Bulk Specific Gravity	2.77		

Note: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

*Dimensions and number of specimens are not based on C-97 standard recommendations.

Data entry by: BKL
Data checked by: HN
FileName: HZSA8365

Date: 02/23/2010
Date: 2/23/2010

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO. 2148-43
LOCATION

DYKE NO.	Dyke 7.2	Dyke 7.2	Dyke 7.2
SAMPLE IDENTIFICATION	753366-753369	753366-753369	753366-753369
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10	2/19/10	2/19/10
TEST DATA			
Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	389.62	366.68	428.80
Wt. of Soaked Sample in Distilled Water (gms)	245.30	232.00	278.70
Wt. of Dried Sample (gms)	389.19	366.20	428.42
SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS			
Bulk Specific Gravity	2.70	2.72	2.85
Absorption, weight (%)	0.11	0.13	0.09
Average Absorption, weight (%)	0.11		
Average Bulk Specific Gravity	2.76		

Note: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

*Dimensions and number of specimens are not based on C-97 standard recommendations.

Data entry by: BKL
Data checked by: HN
FileName: HZSA7269

Date: 02/23/2010
Date: 2/23/2010

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO. 2148-43
LOCATION

DYKE NO.	Dyke 9.2	Dyke 9.2	Dyke 9.2
SAMPLE IDENTIFICATION	399-402	399-402	399-402
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10	2/19/10	2/19/10
TEST DATA			
Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	339.42	255.30	381.81
Wt. of Soaked Sample in Distilled Water (gms)	211.90	162.30	240.80
Wt. of Dried Sample (gms)	338.66	255.02	381.30
SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS			
Bulk Specific Gravity	2.66	2.74	2.70
Absorption, weight (%)	0.22	0.11	0.13
Average Absorption, weight (%)	0.16		
Average Bulk Specific Gravity	2.70		

Note: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

*Dimensions and number of specimens are not based on C-97standard recommendations.

Data entry by: BKL
Data checked by: HN
FileName: HZSA9202

Date: 02/23/2010
Date: 2/23/2010

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO. 2148-43
LOCATION

DYKE NO.	Dyke 7.2	Dyke 7.2	Dyke 7.2
SAMPLE IDENTIFICATION	753025-753027	753025-753027	753025-753027
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10	2/19/10	2/19/10
TEST DATA			
Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	383.47	365.38	414.50
Wt. of Soaked Sample in Distilled Water (gms)	244.10	233.00	265.80
Wt. of Dried Sample (gms)	382.95	364.81	413.89
SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS			
Bulk Specific Gravity	2.75	2.76	2.78
Absorption, weight (%)	0.14	0.16	0.15
Average Absorption, weight (%)	0.15		
Average Bulk Specific Gravity	2.76		

Note: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

*Dimensions and number of specimens are not based on C-97 standard recommendations.

Data entry by: BKL
Data checked by: HN
FileName: HZSA7227

Date: 02/23/2010
Date: 2/23/2010

ATT

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO. 2148-43
LOCATION

DYKE NO.	Dyke 11.2	Dyke 11.2	Dyke 11.2
SAMPLE IDENTIFICATION	1610-1613	1610-1613	1610-1613
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10	2/19/10	2/19/10
TEST DATA			
Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	494.20	402.66	266.20
Wt. of Soaked Sample in Distilled Water (gms)	316.50	256.30	165.40
Wt. of Dried Sample (gms)	493.69	402.23	265.90
SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS			
Bulk Specific Gravity	2.78	2.75	2.64
Absorption, weight (%)	0.10	0.11	0.11
Average Absorption, weight (%)	0.11		
Average Bulk Specific Gravity	2.72		

Note: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

*Dimensions and number of specimens are not based on C-97 standard recommendations.

Data entry by: BKL
Data checked by: HN
FileName: HZSA112A

Date: 02/23/2010
Date: 2/23/2010

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO. 2148-43
LOCATION

DYKE NO.	Dyke 13.2	Dyke 13.2	Dyke 13.2
SAMPLE IDENTIFICATION	2226-2228	2226-2228	2226-2228
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10	2/19/10	2/19/10
TEST DATA			
Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	335.92	261.25	368.38
Wt. of Soaked Sample in Distilled Water (gms)	213.50	165.40	236.60
Wt. of Dried Sample (gms)	335.46	260.81	367.88
SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS			
Bulk Specific Gravity	2.74	2.72	2.79
Absorption, weight (%)	0.14	0.17	0.14
Average Absorption, weight (%)	0.15		
Average Bulk Specific Gravity	2.75		

Note: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

*Dimensions and number of specimens are not based on C-97 standard recommendations.

Data entry by: BKL
Data checked by: HN
FileName: HZSA132A

Date: 02/23/2010
Date: 2/23/2010

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO. 2148-43
LOCATION

DYKE NO.	Dyke 7.2	Dyke 7.2	Dyke 7.2
SAMPLE IDENTIFICATION	753407-753409	753407-753409	753407-753409
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10	2/19/10	2/19/10
TEST DATA			
Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	367.06	392.08	350.03
Wt. of Soaked Sample in Distilled Water (gms)	231.20	249.80	224.70
Wt. of Dried Sample (gms)	366.62	391.64	349.58

SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS

Bulk Specific Gravity	2.70	2.75	2.79
Absorption, weight (%)	0.12	0.11	0.13
Average Absorption, weight (%)	0.12		
Average Bulk Specific Gravity	2.75		

Note: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

*Dimensions and number of specimens are not based on C-97 standard recommendations.

Data entry by: BKL
Data checked by: HN
FileName: HZSA7209

Date: 02/23/2010
Date: 2/23/2010

ADVANCED TERRA TESTING, INC. ATT

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO. 2148-43
LOCATION

DYKE NO.	Dyke 8.3	Dyke 8.3	Dyke 8.3
SAMPLE IDENTIFICATION	105-107	105-107	105-107
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10	2/19/10	2/19/10
TEST DATA			
Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	281.03	410.90	286.35
Wt. of Soaked Sample in Distilled Water (gms)	176.30	259.00	177.50
Wt. of Dried Sample (gms)	280.60	410.43	285.90
SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS			
Bulk Specific Gravity	2.68	2.70	2.63
Absorption, weight (%)	0.15	0.11	0.16
Average Absorption, weight (%)	0.14		
Average Bulk Specific Gravity	2.67		

Note: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

*Dimensions and number of specimens are not based on C-97 standard recommendations.

Data entry by: BKL
Data checked by: HN
FileName: HZSA8307

Date: 02/23/2010
Date: 2/23/2010

ADVANCED TERRA TESTING, INC. ATT

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO. 2148-43
LOCATION

DYKE NO.	Dyke 7.2	Dyke 7.2	Dyke 7.2
SAMPLE IDENTIFICATION	753085-753087	753085-753087	753085-753087
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10	2/19/10	2/19/10
TEST DATA			
Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	311.94	369.98	344.19
Wt. of Soaked Sample in Distilled Water (gms)	202.60	239.20	221.50
Wt. of Dried Sample (gms)	311.60	369.58	343.81
SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS			
Bulk Specific Gravity	2.85	2.83	2.80
Absorption, weight (%)	0.11	0.11	0.11
Average Absorption, weight (%)	0.11		
Average Bulk Specific Gravity	2.83		

Note: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

*Dimensions and number of specimens are not based on C-97 standard recommendations.

Data entry by: BKL
Data checked by: HN
FileName: HASA7287

Date: 02/23/2010
Date: 2/23/2010

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO. 2148-43
LOCATION

DYKE NO.	Dyke 9.2	Dyke 9.2	Dyke 9.2
SAMPLE IDENTIFICATION	354-357	354-357	354-357
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10	2/19/10	2/19/10
TEST DATA			
Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	405.72	297.83	296.66
Wt. of Soaked Sample in Distilled Water (gms)	255.00	186.50	182.80
Wt. of Dried Sample (gms)	405.26	297.41	296.18

SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS

Bulk Specific Gravity	2.69	2.67	2.60
Absorption, weight (%)	0.11	0.14	0.16
Average Absorption, weight (%)	0.14		
Average Bulk Specific Gravity	2.65		

Note: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

*Dimensions and number of specimens are not based on C-97 standard recommendations.

Data entry by: BKL
Data checked by: HN
FileName: HZSA9257

Date: 02/23/2010
Date: 2/23/2010

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO. 2148-43
LOCATION

DYKE NO.	Dyke 10.4	Dyke 10.4	Dyke 10.4
SAMPLE IDENTIFICATION	1545-1546	1545-1546	1545-1546
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10.	2/19/10.	2/19/10.
TEST DATA			
Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	351.36	212.58	290.94
Wt. of Soaked Sample in Distilled Water (gms)	218.00	142.30	185.80
Wt. of Dried Sample (gms)	350.85	212.41	290.52
SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS			
Bulk Specific Gravity	2.63	3.02	2.76
Absorption, weight (%)	0.15	0.08	0.14
Average Absorption, weight (%)	0.12		
Average Bulk Specific Gravity	2.81		

NOte: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

*Dimensions and number of specimens are not based on C-97 standard recommendations.

Data entry by: BKL
Data checked by: HV
FileName: HZSA104A

Date: 02/23/2010
Date: 2/23/2010

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO. 2148-43
LOCATION

DYKE NO.	Dyke 8.7	Dyke 8.7	Dyke 8.7
SAMPLE IDENTIFICATION	234-237	234-237	234-237
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10	2/19/10	2/19/10
TEST DATA			
Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	421.00	314.02	356.67
Wt. of Soaked Sample in Distilled Water (gms)	268.60	196.90	228.90
Wt. of Dried Sample (gms)	420.58	313.57	356.26

SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS

Bulk Specific Gravity	2.76	2.68	2.79
Absorption, weight (%)	0.10	0.14	0.12
Average Absorption, weight (%)	0.12		
Average Bulk Specific Gravity	2.74		

Note: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

*Dimensions and number of specimens are not based on C-97 standard recommendations.

Data entry by: BKL
Data checked by: HN
FileName: HZSA8734

Date: 02/23/2010
Date: 2/23/2010

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO. 2148-43
LOCATION

DYKE NO.	Dyke 8.7	Dyke 8.7	Dyke 8.7
SAMPLE IDENTIFICATION	658-659	658-659	658-659
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10	2/19/10	2/19/10
TEST DATA			
Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	202.48	389.53	374.67
Wt. of Soaked Sample in Distilled Water (gms)	127.80	250.10	240.60
Wt. of Dried Sample (gms)	202.20	389.12	374.23
SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS			
Bulk Specific Gravity	2.71	2.79	2.79
Absorption, weight (%)	0.14	0.11	0.12
Average Absorption, weight (%)	0.12		
Average Bulk Specific Gravity	2.76		

Note: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

*Dimensions and number of specimens are not based on C-97standard recommendations.

Data entry by: BKL Date: 02/23/2010
Data checked by: HN Date: 2/23/10
FileName: HZSA8758

ABSORPTION AND BULK SPECIFIC GRAVITY OF DIMENSION STONE
*ASTM C 97

CLIENT Hazen Research JOB NO.: 2148-43
PROJECT

DYKE NO.	Dyke 7.6	Dyke 7.6	Dyke 7.6
SAMPLE IDENTIFICATION	753534-753536	753534-753536	753534-753536
SAMPLE NO.	A	B	C
DATE SAMPLED			
DATE TESTED	2/19/10 BKL	2/19/10 BKL	2/19/10 BKL

TEST DATA

Temperature of Distilled Water. (C)	20	20	20
Wt. of Soaked Surface-Dried in air Sample (gms)	275.30	231.85	461.30
Wt. of Soaked Sample in Distilled Water (gms)	178.10	151.20	292.60
Wt. of Dried Sample (gms)	274.97	231.55	460.68

SPECIFIC GRAVITY & ABSORPTION DETERMINATIONS

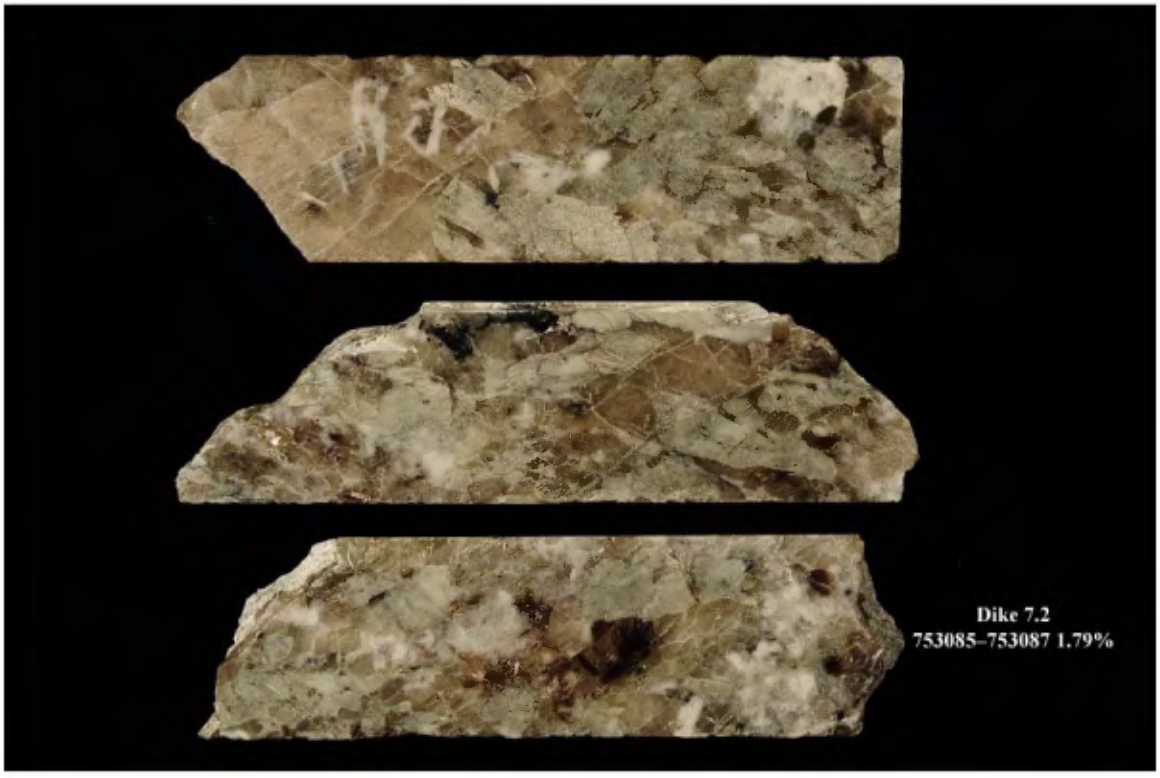
Bulk Specific Gravity	2.83	2.87	2.73
Absorption, weight (%)	0.12	0.13	0.13
Average Absorption, weight (%)	0.13		
Average Bulk Specific Gravity	2.81		

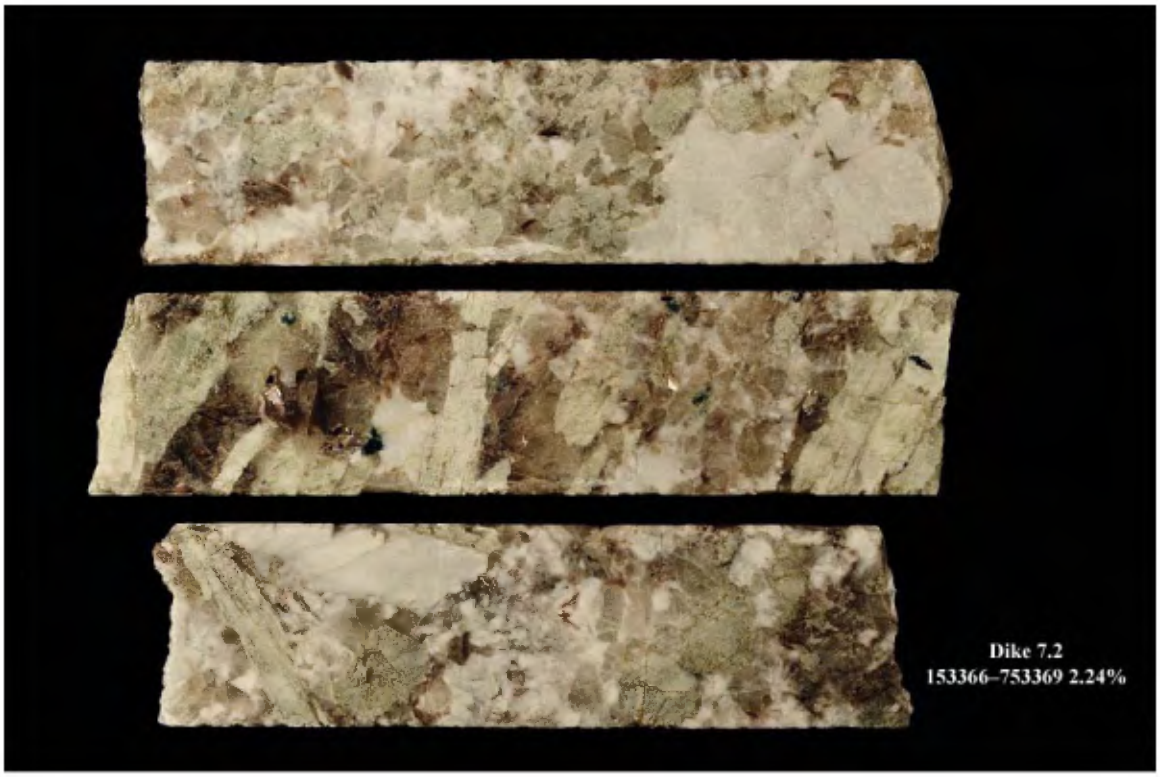
Note: Specimens were dried for 48h in a ventilated oven at 60+/-2C, then after cooling and weighing were immersed completely in distilled water at 22+/-2C for 48h, surface dry with a damp cloth and weighed.

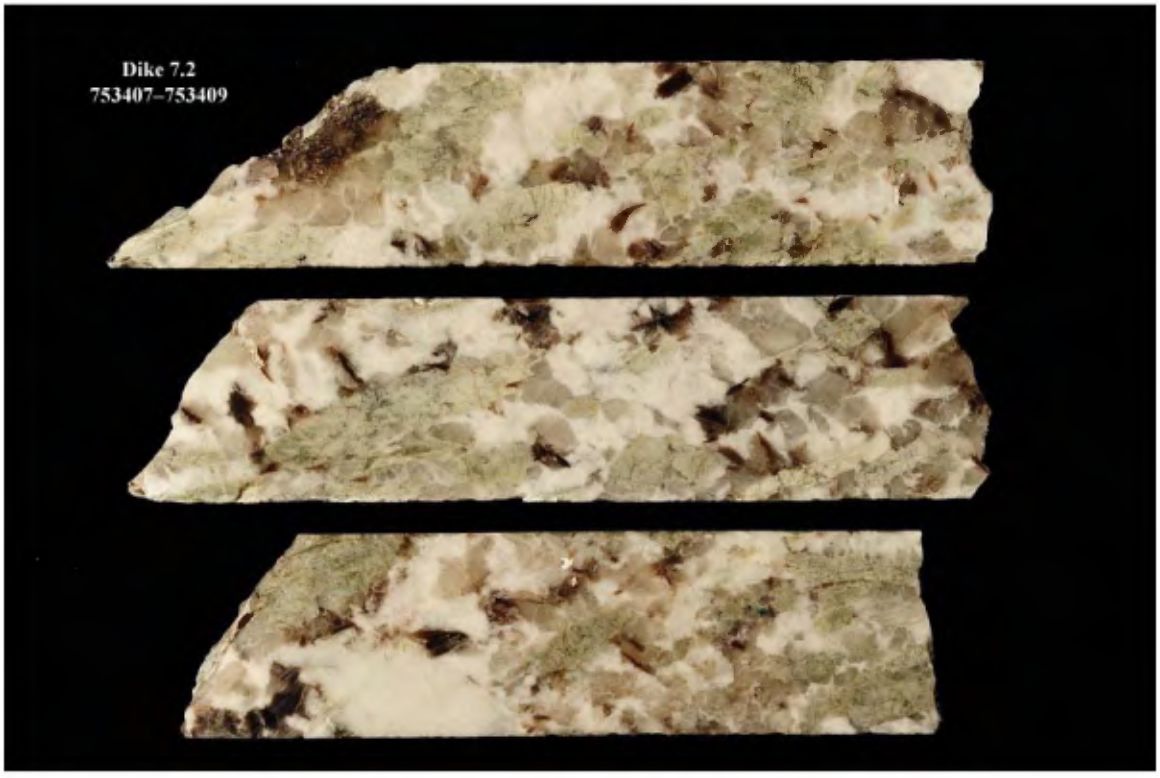
*Dimensions and number of specimens are not based on C-97standard recommendations.

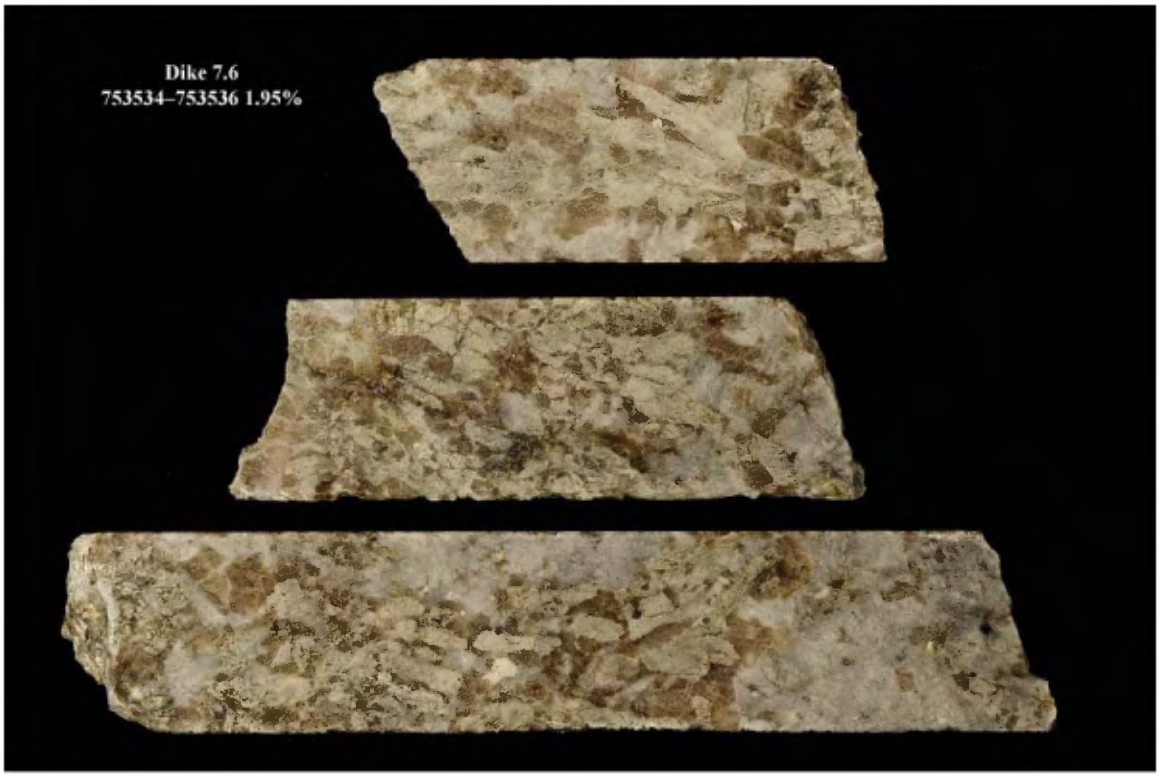
Data entry by: BKL Date: 02/23/2010
Data checked by: HN Date: 2/22/10
FileName: HZSA7636


Dike 7.2
753025–753027 2.07%

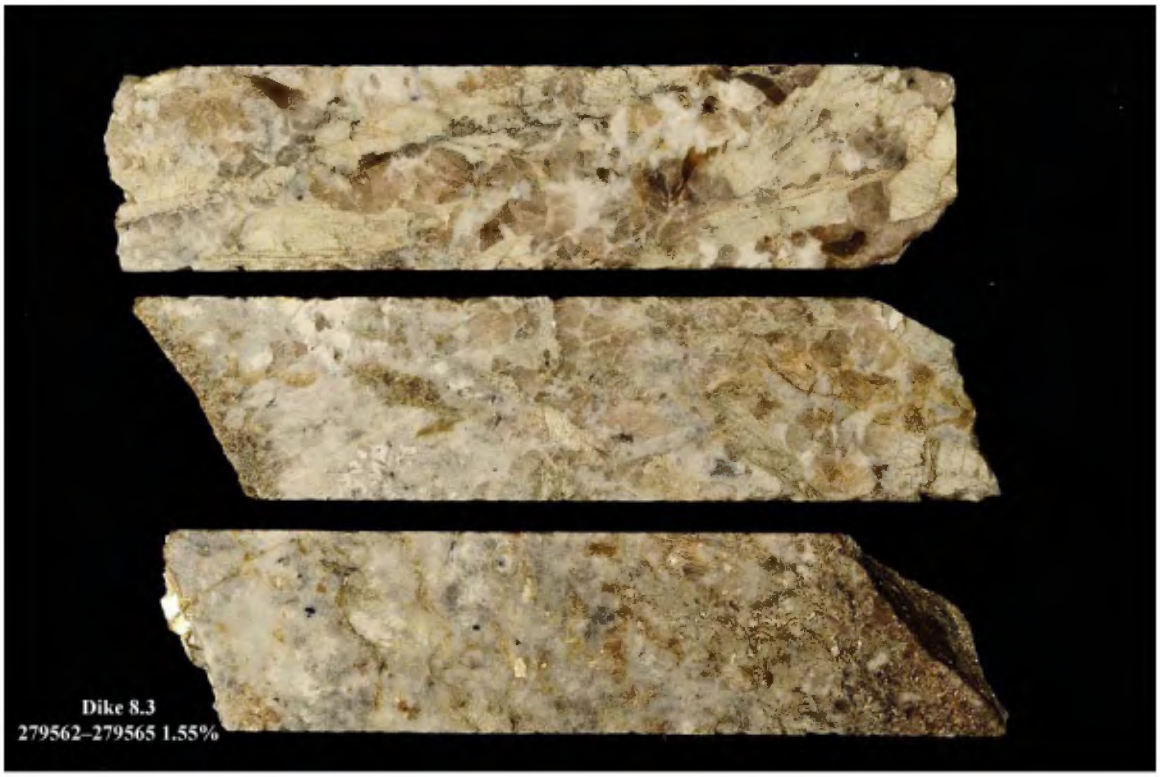

Dike 7.2
753085–753087 1.79%

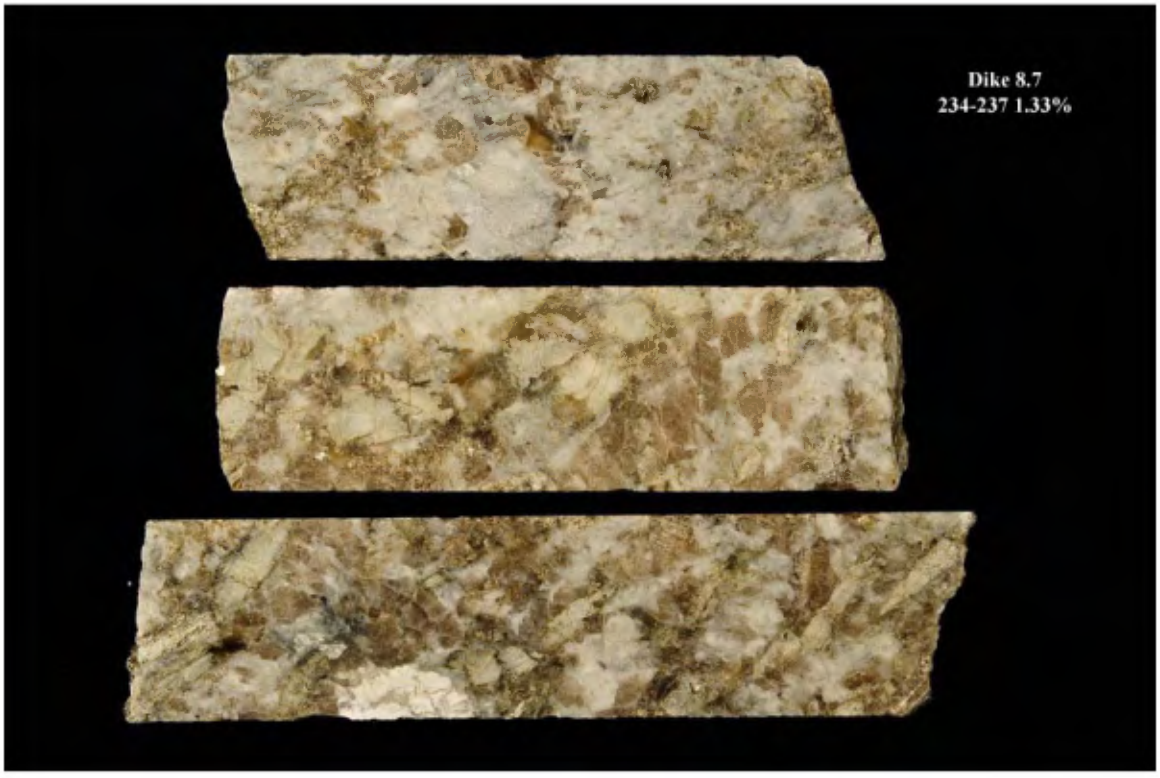

Dike 7.2
153366–753369 2.24%

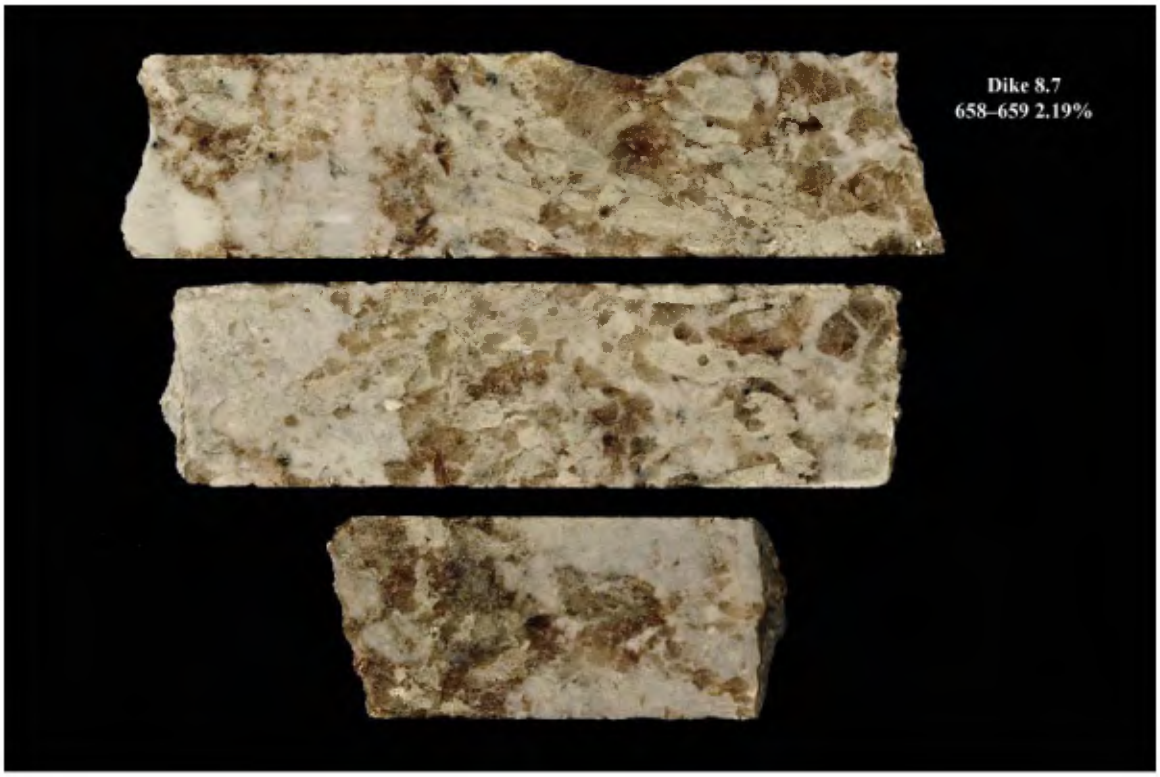

Dike 7.2
753407–753409

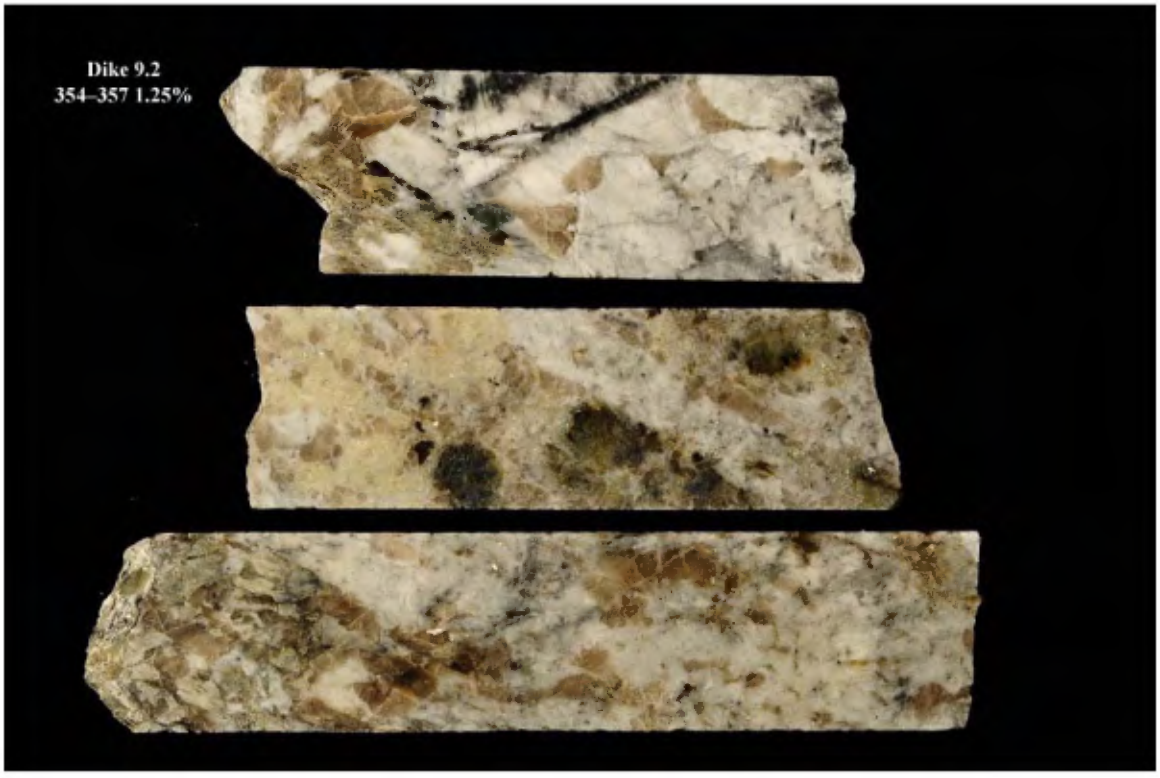

Dike 7.6
753534–753536 1.95%

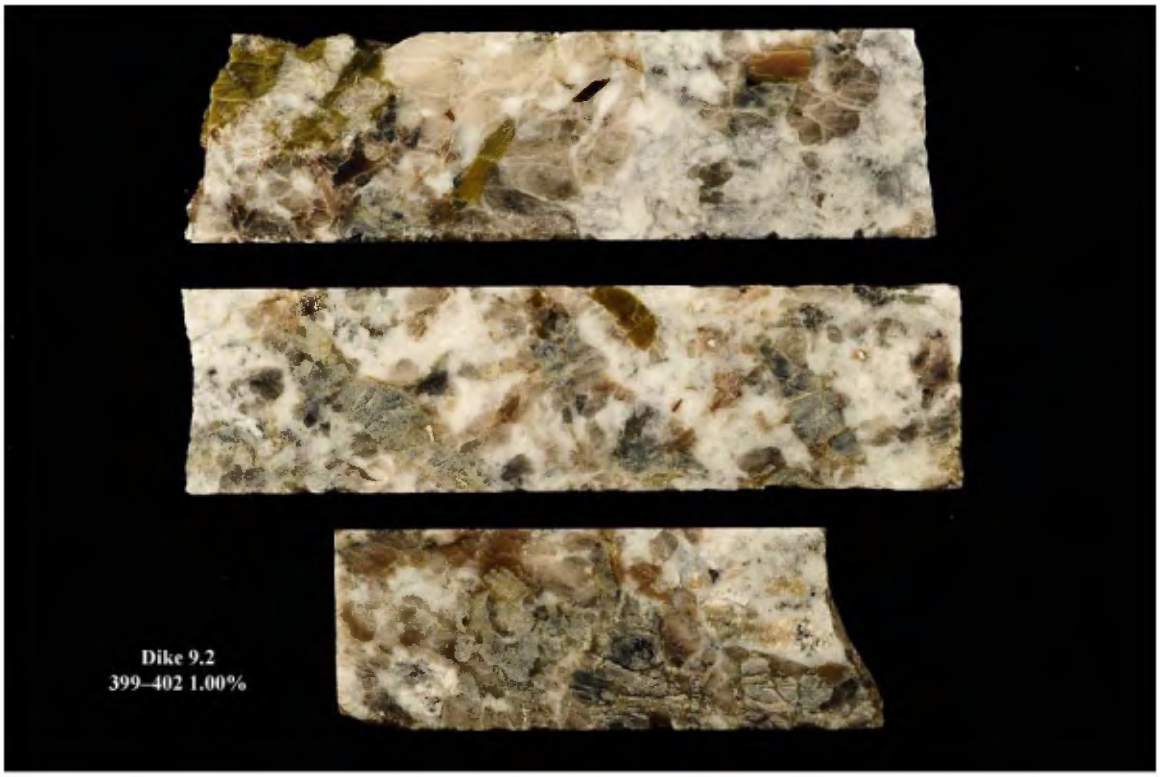

Dike 8.3
105–107 1.44%


Dike 8.3
279562–279565 1.55%

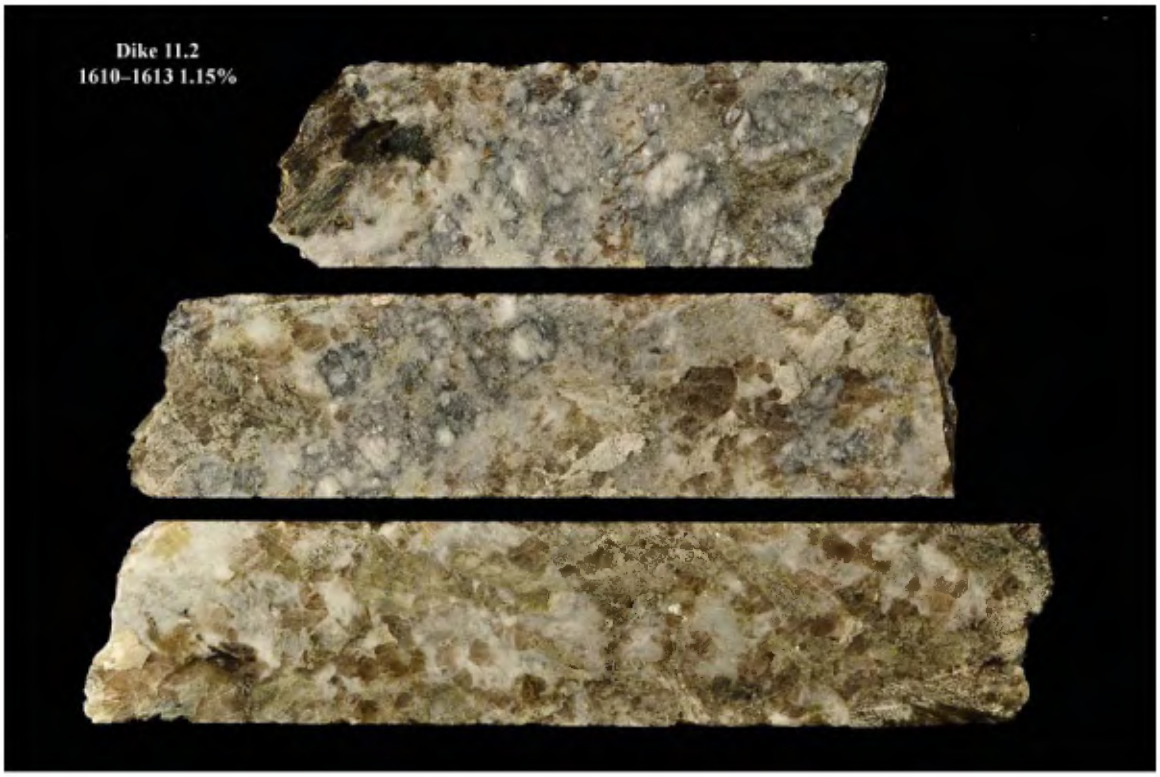

Dike 8.7
234-237 1.33%


Dike 8.7
658–659 2.19%


Dike 9.2
354–357 1.25%


Dike 9.2
399–402 1.00%


Dike 10.4
1545–1546 2.46%


Dike 11.2
1610–1613 1.15%


Dike 13.2
2226–2228 1.83%

APPENDIX C

XRD Analyses of Dike Samples

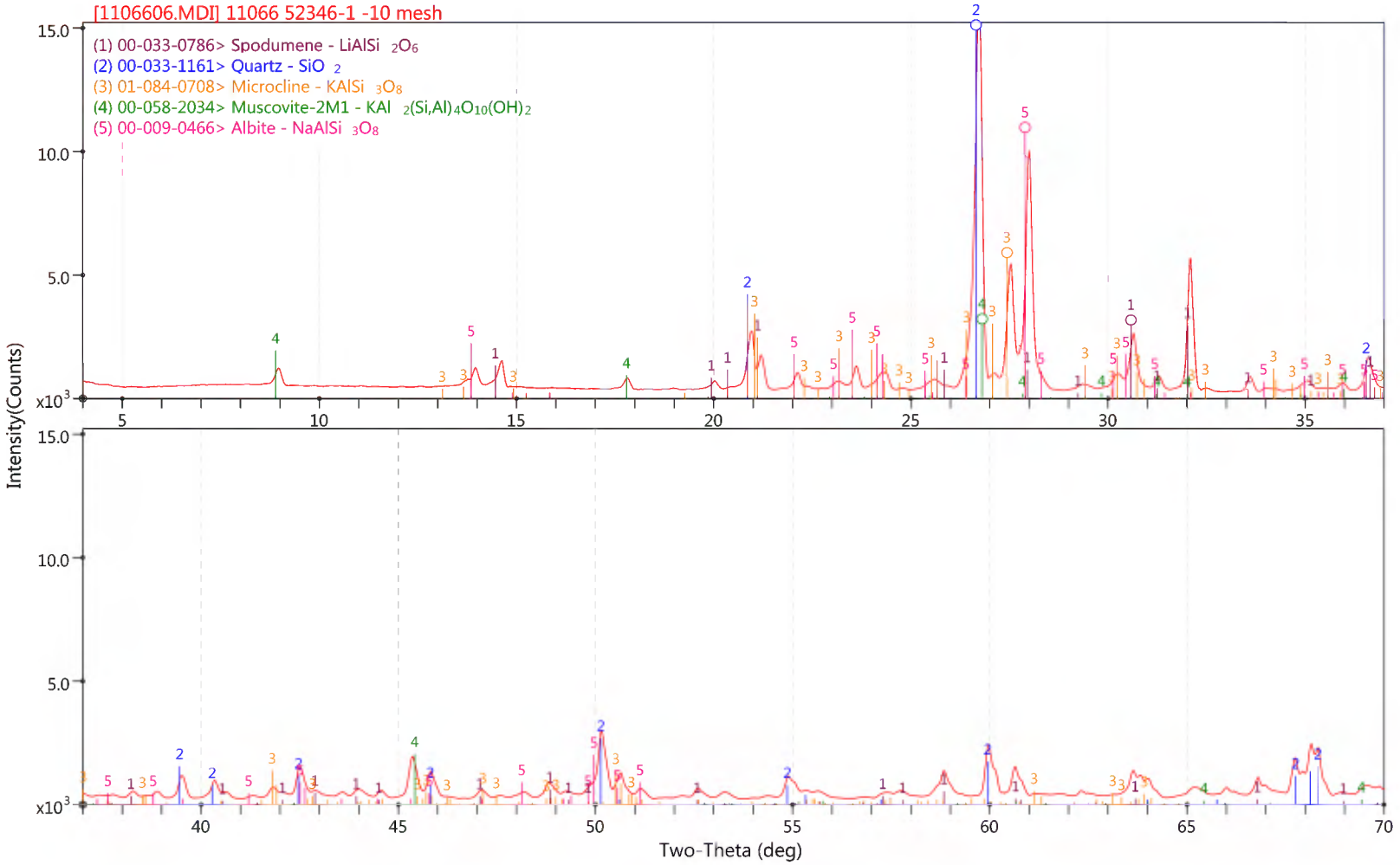


Figure C1. XRD Pattern, Dike 7.2, Minus 1.7 mm

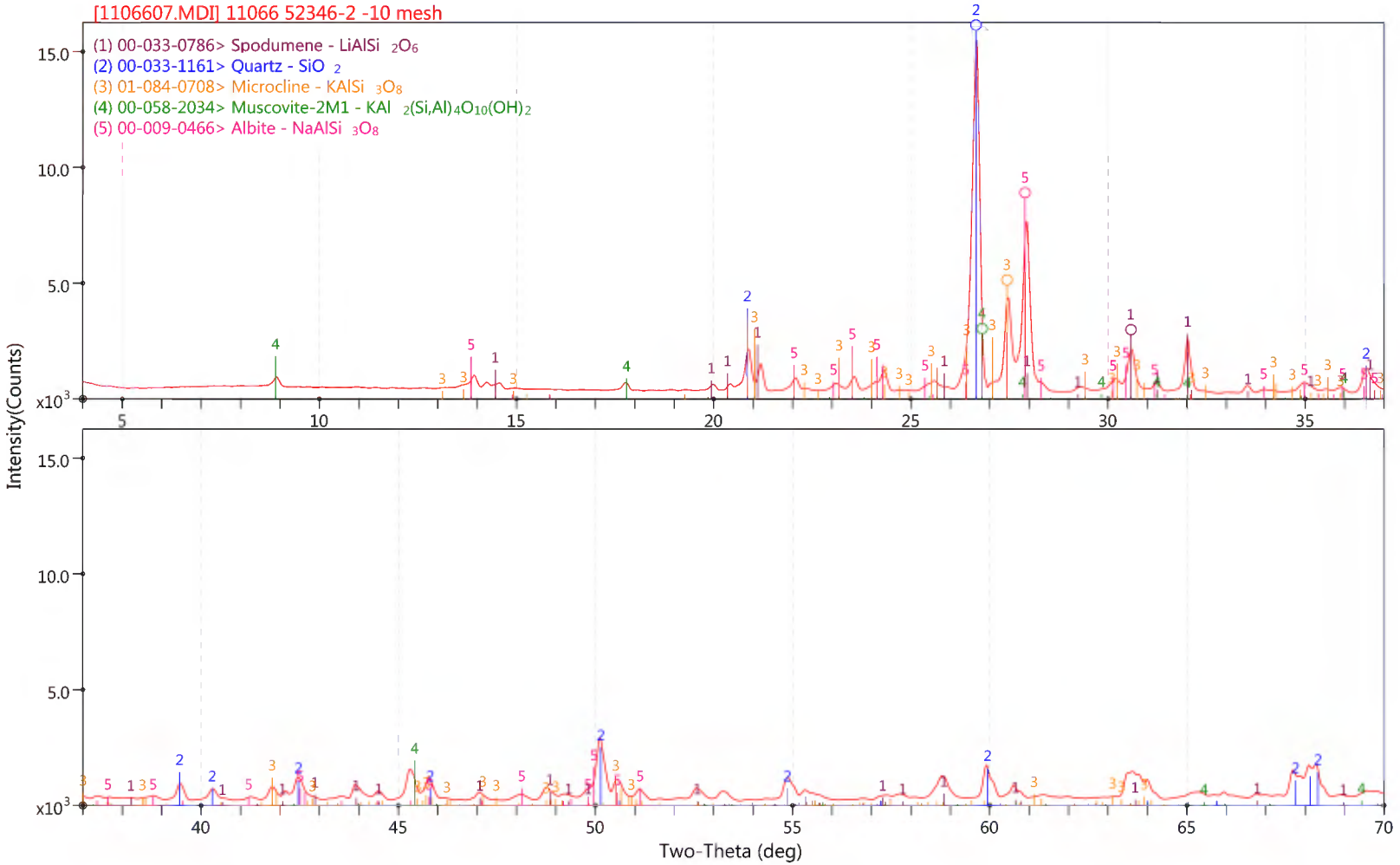


Figure C2. XRD Pattern, Dike 7.6, Minus 1.7 mm

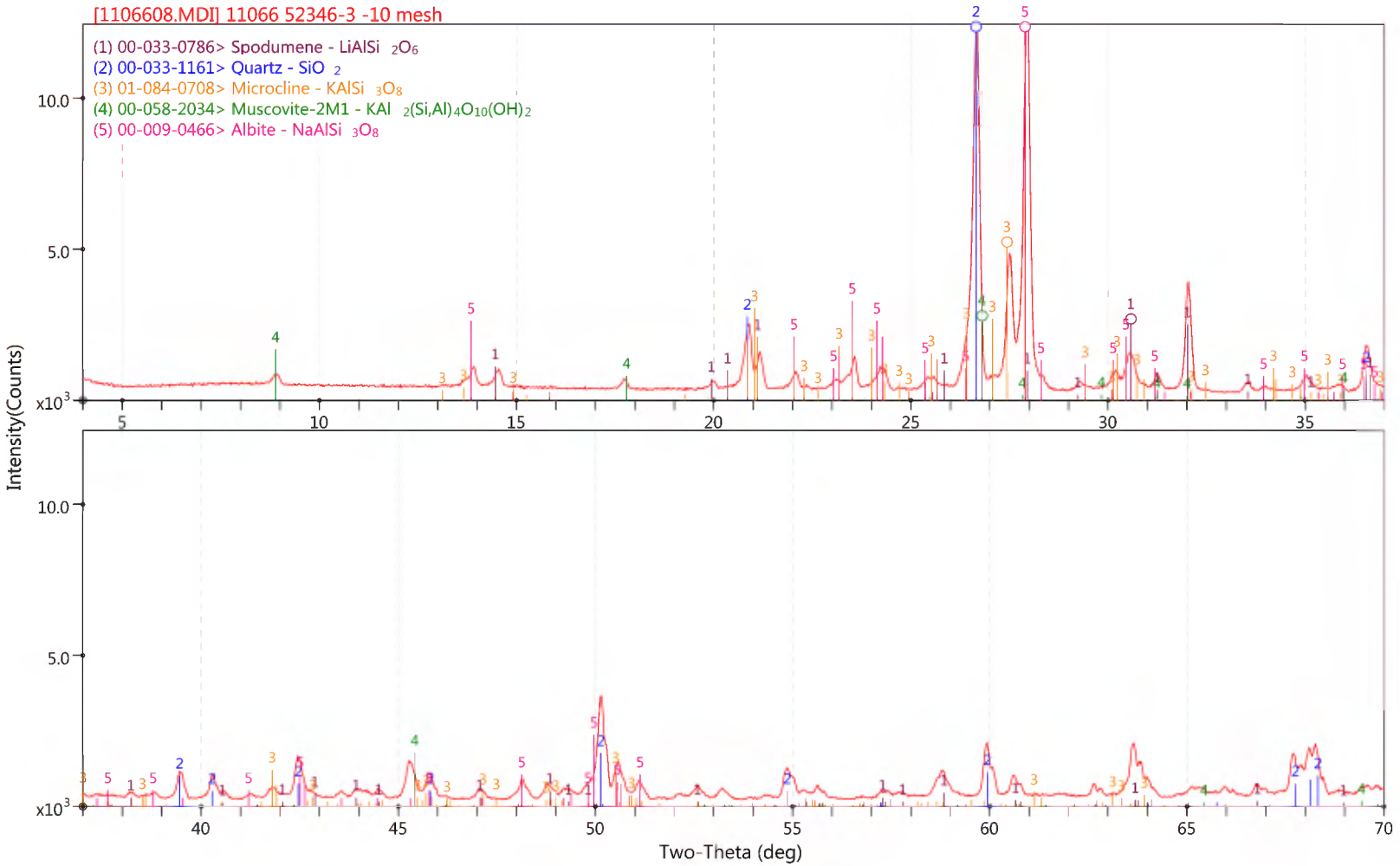


Figure C3. XRD Pattern, Dike 8.3, Minus 1.7 mm

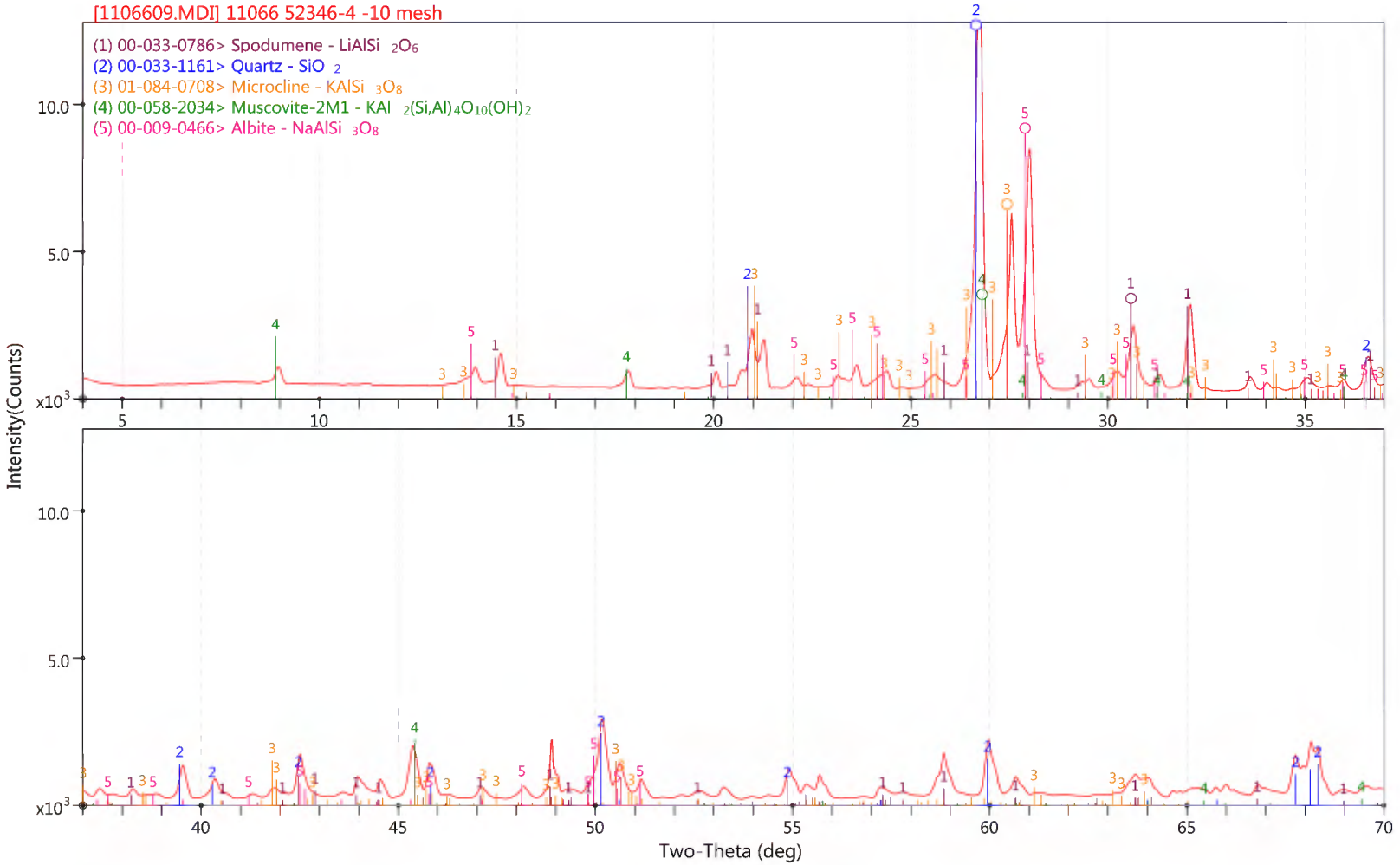


Figure C4. XRD Pattern, Dike 8.7, Minus 1.7 mm

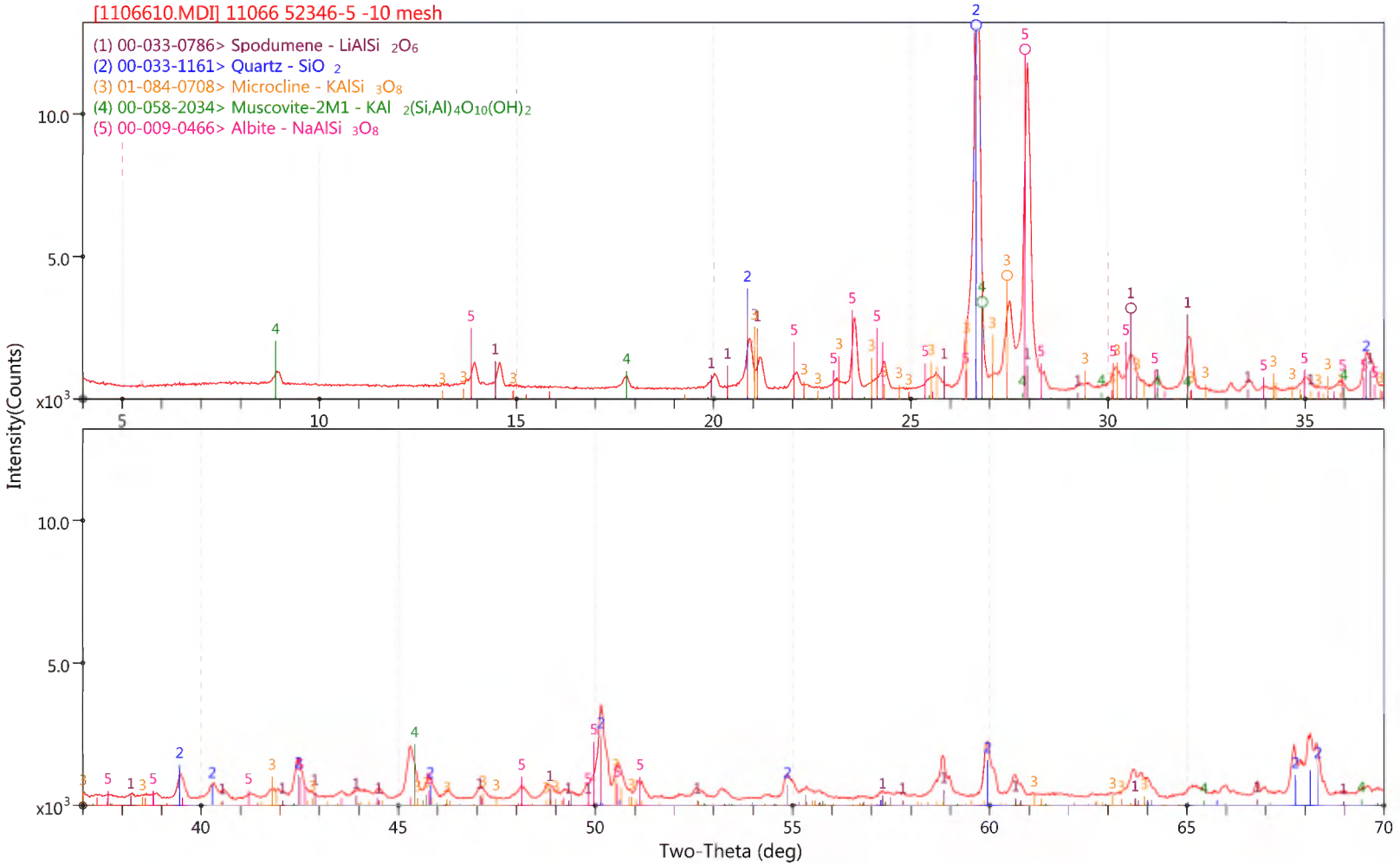


Figure C5. XRD Pattern, Dike 9.2, Minus 1.7 mm

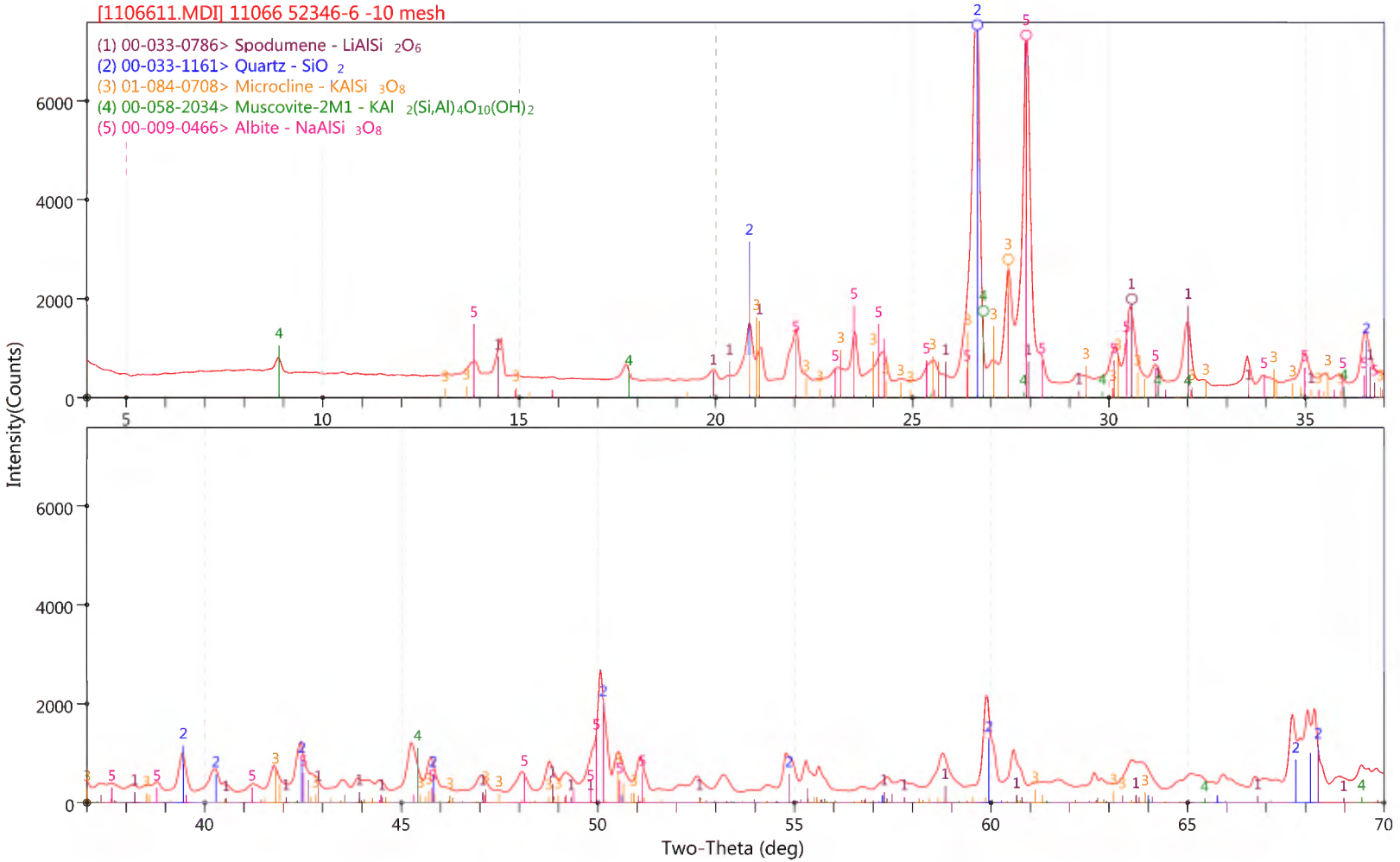


Figure C6. XRD Pattern, Dike 10.4, Minus 1.7 mm

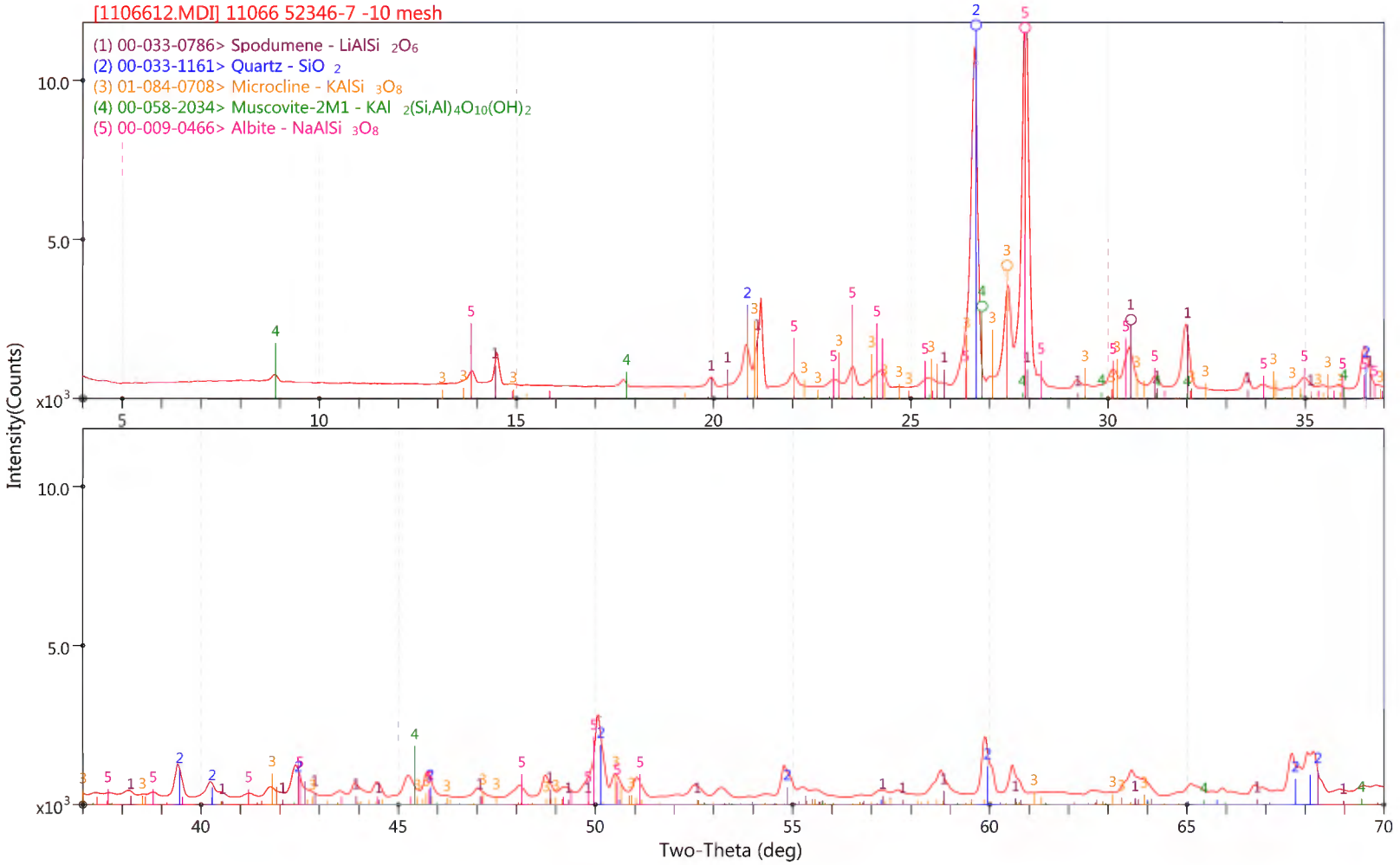


Figure C7. XRD Pattern, Dike 11.2, Minus 1.7 mm

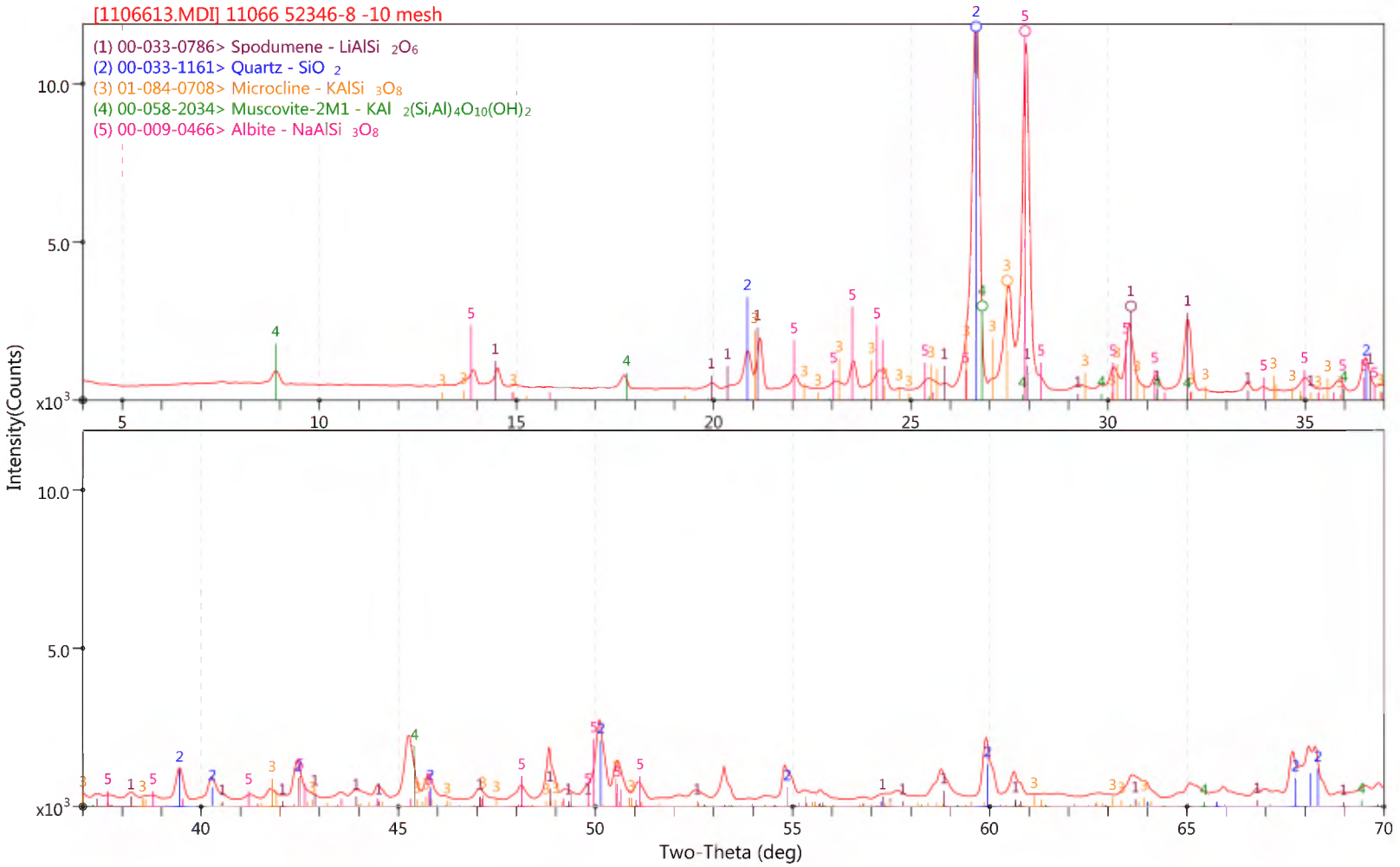


Figure C8. XRD Pattern, Dike 12.2, Minus 1.7 mm

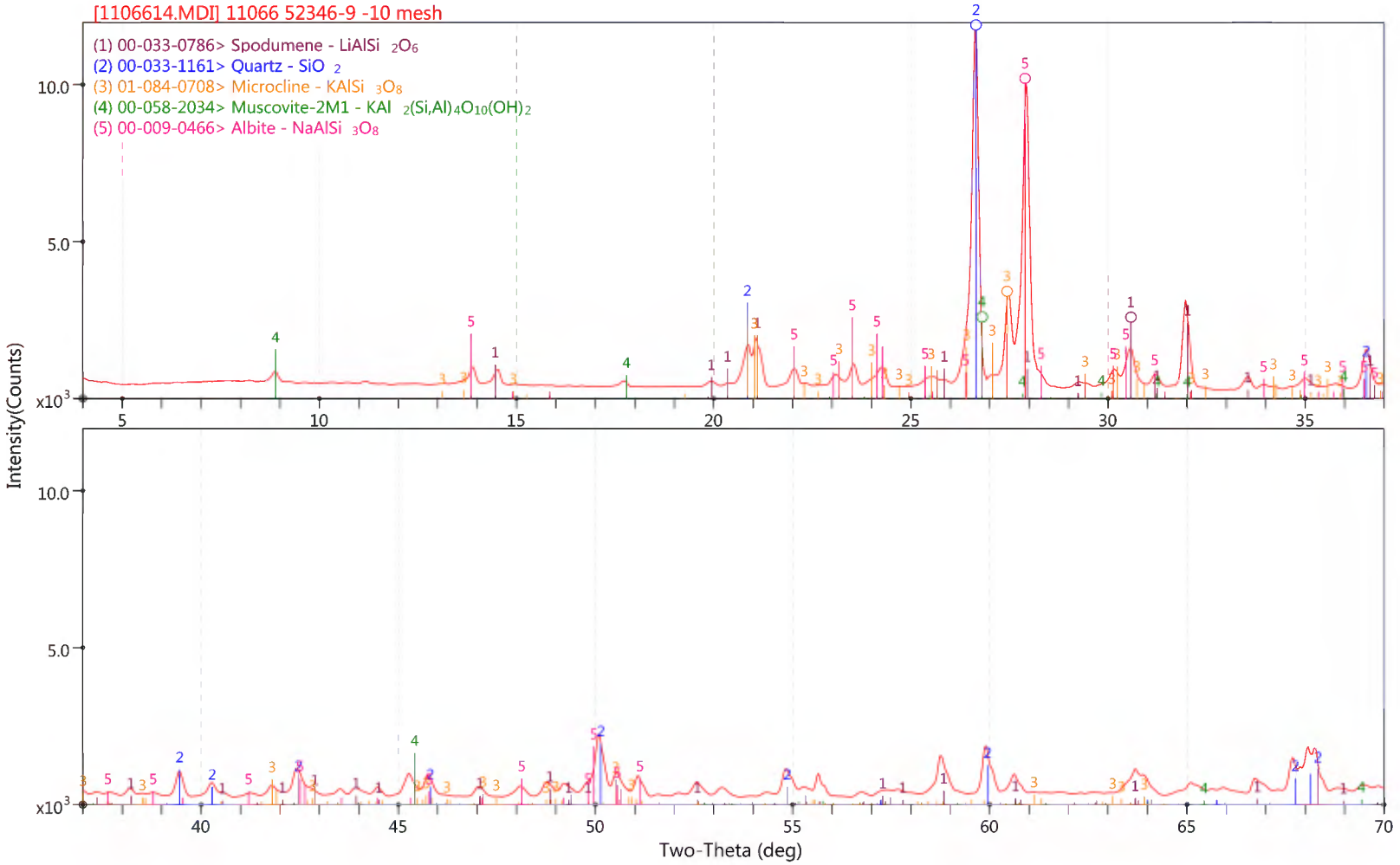


Figure C9. XRD Pattern, Dike 13.2, Minus 1.7 mm

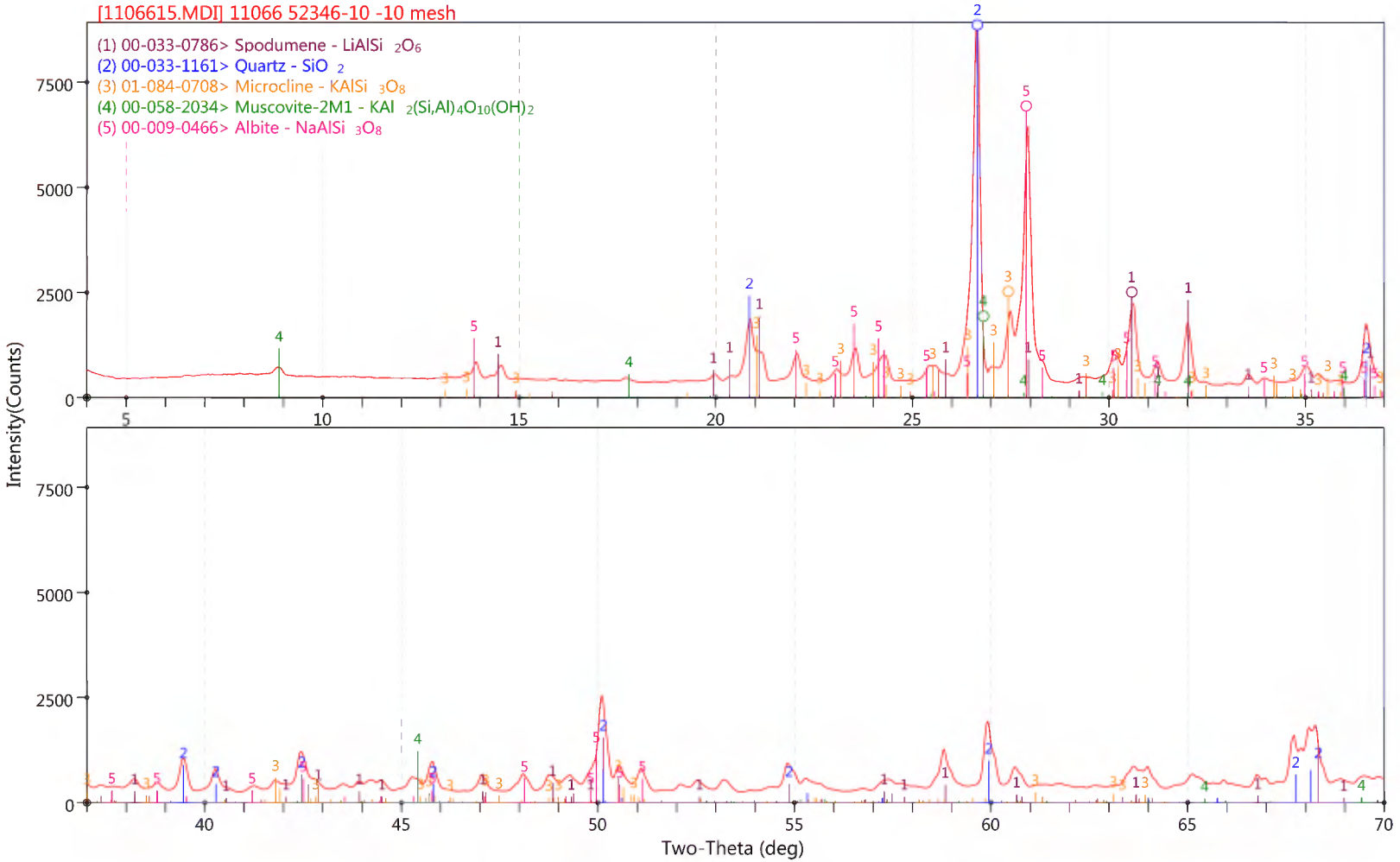


Figure C10. XRD Pattern, Dike 14.2, Minus 1.7 mm

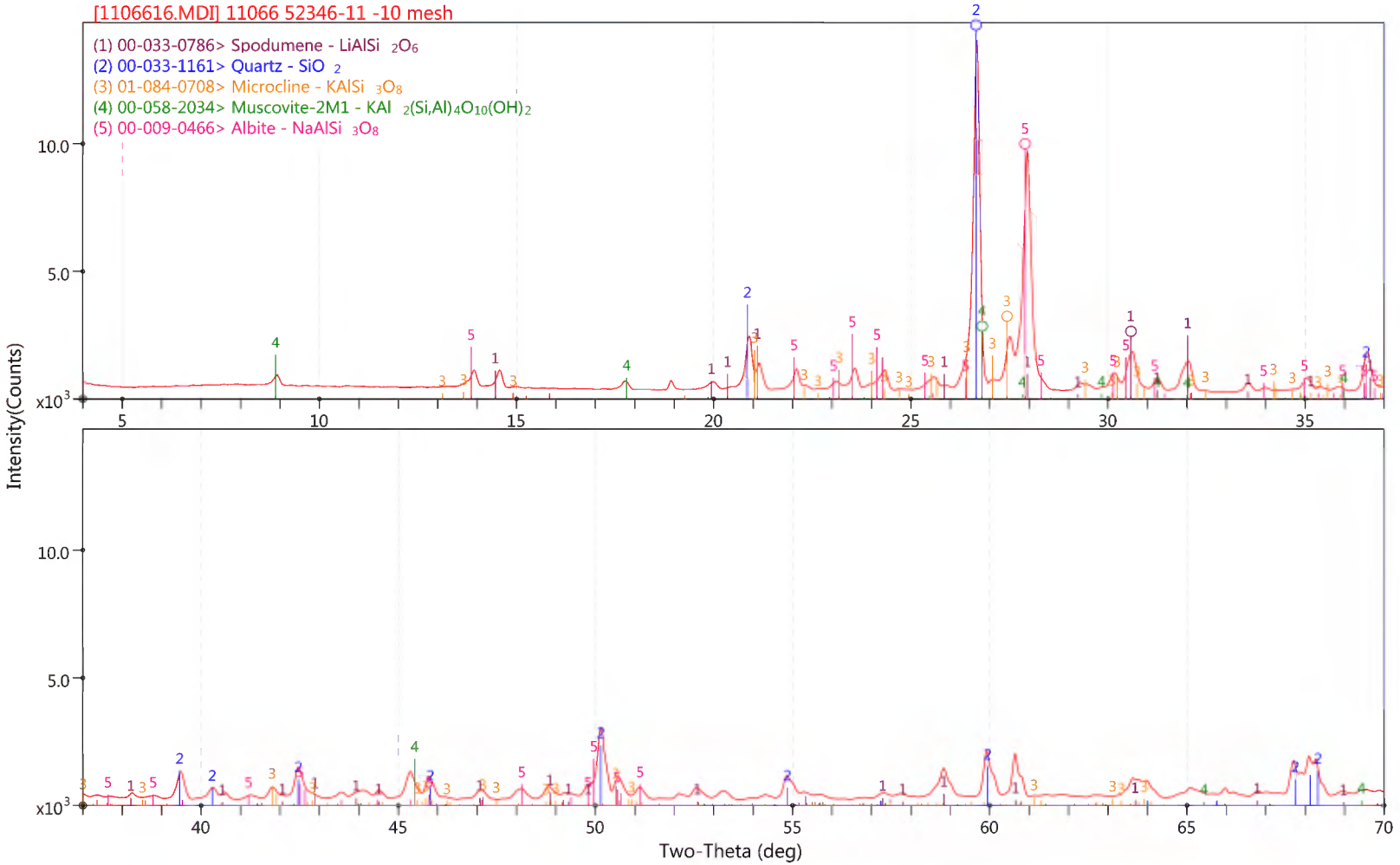


Figure C11. XRD Pattern, Dike 15.1, Minus 1.7 mm

APPENDIX D

Comminution Report

HAZEN RESEARCH, INC.

4601 Indiana Street · Golden, Colorado 80403 USA
Phone: (303) 279-4501 · Fax: (303) 278-1528
www.hazenresearch.com

August 26, 2010

E-mail Delivery

Mr. Lain Scarr
Lithium One Incorporated
9137 Ridgeline Blvd, Suite 250
Highlands Ranch, CO 80129

Subject: Comminution Testing
Hazen Project 11066-01

Dear Mr. Scarr:

As requested, the samples received at Hazen Research, Inc. on June 11, 2010, were subjected to semiautogeneous (SAG) mill comminution (SMC), Bond ball mill work index (BW_i), and Bond abrasion work index (A_i) testing. Upon receipt, the samples were assigned Hazen numbers for internal identification and future reference, as shown in Table 1. The table also summarizes the BW_i and A_i values; the complete results are enclosed.

Table 1. Sample Identification

HRI	Client ID	BW_i, kWh/t	A_i, g
52461-1	Dike 7.2	18.0	0.4659
52461-2	Dike 7.6	17.5	0.4244
52461-3	Dike 8.3	17.6	0.4443
52461-4	Dike 8.7	17.5	0.3936
52461-5	Dike 9.2	17.6	0.4151
52461-6	Dike 10.4	17.6	0.4224
52461-7	Dike 11.2	18.2	0.4380
52461-8	Dike 12.2	17.0	0.4174
52461-9	Dike 13.2	16.9	0.4136
52461-10	Dike 14.2	17.4	0.4280
52461-11	Dike 15.1	17.9	0.3750

The SMC test was developed by SMC Testing Pty Ltd (SMCT) to provide a cost-effective means of obtaining drop-weight parameters from drill core samples, as well as in situations in which limited quantities of material are available. The results of the evaluations were sent to SMCT to determine the JKSimMet parameters using their database. Table 2 is a summary of the parameters determined by SMCT from the results of the SMC evaluation on your samples. The complete results are enclosed.

Table 2. Summary of SMC Breakage Evaluations

Parameter	Value										
	52461-1	**52461-2**	**52461-3**	**52461-4**	**52461-5**	**52461-6**	**52461-7**	**52461-8**	**52461-9**	**52461-10**	**52461-11**
sg (by weighing in water and air)	2.53	2.69	2.71	2.76	2.76	2.69	2.74	2.73	2.72	2.71	2.74
SMCT Parameters											
A (maximum breakage)	59.8	65.2	64.6	64.3	66.8	63.0	60.5	62.7	63.5	63.4	63.0
b (relation between energy and impact breakage)	1.31	1.10	1.11	1.27	1.00	1.06	1.40	1.13	1.21	1.27	1.23
A × b (overall AG–SAG hardness)	78.3	71.7	71.7	81.7	66.8	66.8	84.7	70.9	76.8	80.5	77.5
SMC Test DW_i, kWh/m^3	3.23	3.77	3.79	3.38	4.14	4.03	3.22	3.84	3.54	3.39	3.54
DW_i, %	19	24	25	20	29	27	19	25	22	20	22
M_{ia}, kWh/t	11.6	12.3	12.3	11.0	12.9	13.0	10.7	12.3	11.6	11.2	11.5
M_{ih}, kWh/t	7.4	8.1	8.1	7.1	8.7	8.7	6.8	8.2	7.5	7.2	7.5
M_{ic}, kWh/t	3.8	4.2	4.2	3.7	4.5	4.5	3.5	4.2	3.9	3.7	3.9
t_a	0.80	0.69	0.68	0.77	0.63	0.64	0.80	0.67	0.73	0.76	0.73

This letter report completes the work authorized for these samples. Hazen looks forward to assisting you with comminution testing in the future. Please contact me if you have questions concerning this project.

Regards,

Daniel W. Gillespie
Project Engineer

DWG/gmr

Enclosures

Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2737**

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number, BW_i.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices. The sample was stage crushed to minus 6 mesh. This material was used as feed for the grindability test.

Sample:
Client Identified: Dike 7.2
Sample No: 52461-1

Results:

P_1 = 100% passing size of product	149	µm
P_{80} = 80% passing size of product	131	µm
F_{80} = 80% passing size of feed	2,454	µm
Gpr = Grams per revolution	1.36	g

Calculation of a Bond Ball Mill Work Index:

$$BW_i = \frac{44.5}{P_1^{0.23} \times Gpr^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)}$$

BW_i = 16.3 kWh/st

BW_i = 18.0 kWh/t

Standard Bond Ball Mill Grindability Test

Date: **Aug-10**
Project No: **11066-01**
Test No: **2737**

Test Conditions and Notes

100% passing size of product:		100	mesh
Test feed weight (700 cm^3):		1175.7	g
Percent minus 100 mesh in feed:		6.4	%
Target product weights:	+ 100 mesh	839.8	g
	- 100 mesh	335.9	g

Test Ball Charge

Ball Size in.	Number of Balls	Weight g
1.45	43	8,806
1.17	67	7,209
1.00	10	672
0.75	71	2,009
0.61	94	1,428
Totals:	285	20,125

			Undersize			Undersize Product		
Stage No.	Revolutions	New Feed g	In Feed g	To Be Ground g	U'Size In Product g	Total g	Per Mill Revolutions g	Circ. Load %
1	100	1175.7	74.8	261.1	193.6	118.8	1.19	507.3
2	272	193.6	12.3	323.6	344.6	332.3	1.22	241.2
3	257	344.6	21.9	314.0	361.9	340.0	1.32	224.9
4	237	361.9	23.0	312.9	342.4	319.4	1.35	243.4
5	233	342.4	21.8	314.1	338.1	316.3	1.36	247.7
6	232	338.1	21.5	314.4	337.3	315.8	1.36	248.6
7	231	337.3	21.5	314.5	333.3	311.8	1.35	252.7

Average of Last Three Stages: 1.36 249.7

Standard Bond Ball Mill Grindability Test

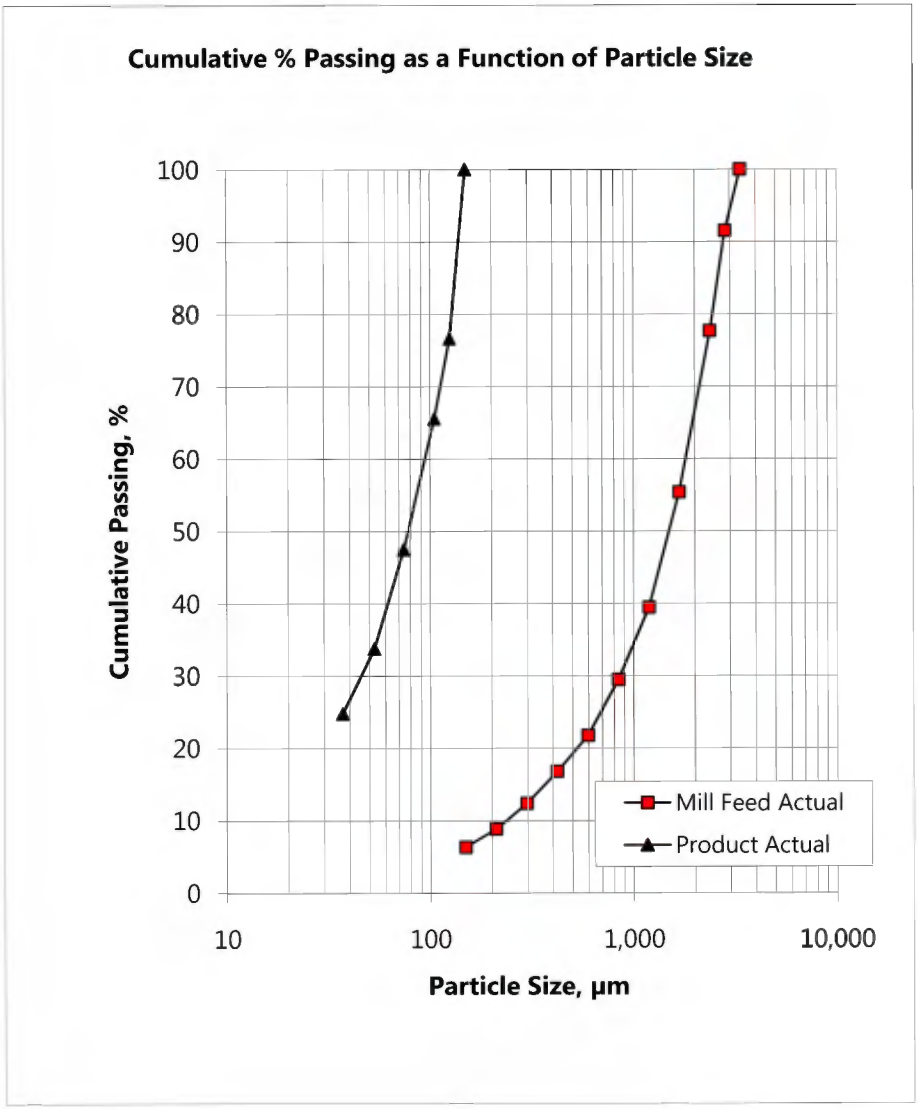
Date: **Aug-10**
Project No: **11066-01**
Test No: **2737**

Detailed Particle Analyses

Screen Size (Retained)		Mill Feed, -6 mesh				Circ. Load, +100 mesh			Test Product, -100 mesh			
Tyler Mesh	µm	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain	g	Wt. %	Pass	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain
6	3,360	0.0	0.0	100.0	0.0	0.0	0.0	100.0				
7	2,830	99.7	8.5	91.5	8.5	16.5	2.0	98.0				
8	2,380	162.8	13.8	77.7	22.3	32.9	3.9	94.1				
10	1,680	261.8	22.3	55.4	44.6	64.3	7.6	86.5				
14	1,190	187.1	15.9	39.5	60.5	52.7	6.3	80.2				
20	841	117.5	10.0	29.5	70.5	41.7	5.0	75.3				
28	595	90.7	7.7	21.8	78.2	50.7	6.0	69.3				
35	420	58.5	5.0	16.8	83.2	61.8	7.3	61.9				
48	297	51.9	4.4	12.4	87.6	114.7	13.6	48.3				
65	210	41.5	3.5	8.9	91.1	205.2	24.4	24.0				
100	149	29.4	2.5	6.4	93.6	201.8	24.0	0.0				
-100	-149	74.8	6.4	0.0	100.0	0.0	0.0	0.0				
	Totals:	1175.7	100.0			842.3	100.0					
100	149								0.0	0.0	100.0	0.0
115	125								49.1	23.4	76.6	23.4
150	105								23.2	11.1	65.5	34.5
200	74								37.9	18.1	47.5	52.5
270	53								28.7	13.7	33.8	66.2
400	37								18.9	9.0	24.8	75.2
-400	-37								52.0	24.8	0.0	100.0
								Totals:	209.8	100.0		

Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2737**



Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2738**

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number, BW_i.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices. The sample was stage crushed to minus 6 mesh. This material was used as feed for the grindability test.

Sample:
Client Identified: Dike 7.6
Sample No: 52461-2

Results:

P_1 = 100% passing size of product	149	µm
P_{80} = 80% passing size of product	132	µm
F_{80} = 80% passing size of feed	2,427	µm
Gpr = Grams per revolution	1.42	g

Calculation of a Bond Ball Mill Work Index:

$$BW_i = \frac{44.5}{P_1^{0.23} \times Gpr^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)}$$

BW_i = 15.9 kWh/st

BW_i = 17.5 kWh/t

Standard Bond Ball Mill Grindability Test

Date: **Aug-10**
Project No: **11066-01**
Test No: **2738**

Test Conditions and Notes

100% passing size of product:		100	mesh
Test feed weight (700 cm^3):		1203.7	g
Percent minus 100 mesh in feed:		8.4	%
Target product weights:	+ 100 mesh	859.8	g
	- 100 mesh	343.9	g

Test Ball Charge

Ball Size in.	Number of Balls	Weight g
1.45	43	8,806
1.17	67	7,209
1.00	10	672
0.75	71	2,009
0.61	94	1,428
Totals:	285	20,125

			Undersize			Undersize Product		
Stage No.	Revolutions	New Feed g	In Feed g	To Be Ground g	U'Size In Product g	Total g	Per Mill Revolutions g	Circ. Load %
1	100	1203.7	101.7	242.2	234.4	132.7	1.33	413.5
2	244	234.4	19.8	324.1	342.3	322.5	1.32	251.7
3	238	342.3	28.9	315.0	362.2	333.3	1.40	232.3
4	224	362.2	30.6	313.3	349.0	318.4	1.42	244.9
5	221	349.0	29.5	314.4	342.5	313.0	1.42	251.4
6	222	342.5	28.9	315.0	342.9	314.0	1.41	251.0
7	223	342.9	29.0	314.9	344.7	315.7	1.42	249.2

Average of Last Three Stages: 1.42 250.6

Standard Bond Ball Mill Grindability Test

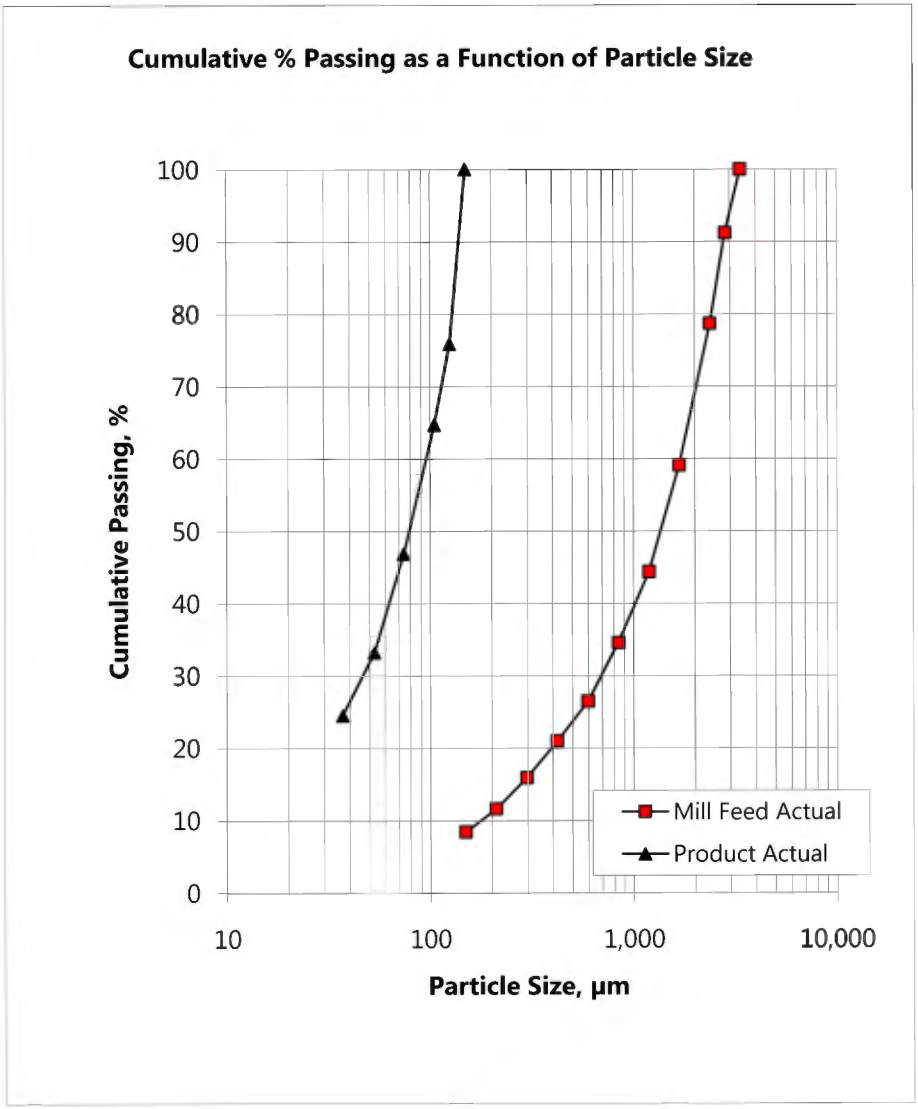
Date: **Aug-10**
Project No: **11066-01**
Test No: **2738**

Detailed Particle Analyses

Screen Size (Retained)		Mill Feed, -6 mesh				Circ. Load, +100 mesh			Test Product, -100 mesh			
Tyler Mesh	µm	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain	g	Wt. %	Pass	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain
6	3,360	0.0	0.0	100.0	0.0	0.0	0.0	100.0				
7	2,830	105.6	8.8	91.2	8.8	17.7	2.1	97.9				
8	2,380	150.6	12.5	78.7	21.3	34.6	4.0	93.9				
10	1,680	236.0	19.6	59.1	40.9	66.1	7.7	86.2				
14	1,190	177.0	14.7	44.4	55.6	53.9	6.3	79.9				
20	841	117.8	9.8	34.6	65.4	41.1	4.8	75.2				
28	595	97.7	8.1	26.5	73.5	51.8	6.0	69.1				
35	420	66.1	5.5	21.0	79.0	65.6	7.6	61.5				
48	297	61.1	5.1	15.9	84.1	134.5	15.7	45.8				
65	210	51.8	4.3	11.6	88.4	244.6	28.5	17.4				
100	149	38.3	3.2	8.4	91.6	149.1	17.4	0.0				
-100	-149	101.7	8.4	0.0	100.0	0.0	0.0	0.0				
	Totals:	1203.7	100.0			859.0	100.0					
100	149								0.0	0.0	100.0	0.0
115	125								50.7	24.2	75.8	24.2
150	105								23.4	11.1	64.7	35.3
200	74								37.4	17.8	46.9	53.1
270	53								28.6	13.6	33.3	66.7
400	37								18.3	8.7	24.5	75.5
-400	-37								51.5	24.5	0.0	100.0
								Totals:	209.9	100.0		

Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2738**



Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2739**

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number, BW_i.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices. The sample was stage crushed to minus 6 mesh. This material was used as feed for the grindability test.

Sample:
Client Identified: Dike 8.3
Sample No: 52461-3

Results:

P_1 = 100% passing size of product	149	µm
P_{80} = 80% passing size of product	132	µm
F_{80} = 80% passing size of feed	2,482	µm
Gpr = Grams per revolution	1.40	g

Calculation of a Bond Ball Mill Work Index:

$$BW_i = \frac{44.5}{P_1^{0.23} \times Gpr^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)}$$

BW_i = 16.0 kWh/st

BW_i = 17.6 kWh/t

Standard Bond Ball Mill Grindability Test

Date: **Aug-10**
Project No: **11066-01**
Test No: **2739**

Test Conditions and Notes

100% passing size of product:		100	mesh
Test feed weight (700 cm^3):		1227.2	g
Percent minus 100 mesh in feed:		7.6	%
Target product weights:	+ 100 mesh	876.6	g
	- 100 mesh	350.6	g

Test Ball Charge

Ball Size in.	Number of Balls	Weight g
1.45	43	8,806
1.17	67	7,209
1.00	10	672
0.75	71	2,009
0.61	94	1,428
Totals:	285	20,125

			Undersize			Undersize Product		
Stage No.	Revolutions	New Feed g	In Feed g	To Be Ground g	U'Size In Product g	Total g	Per Mill Revolutions g	Circ. Load %
1	100	1227.2	93.2	257.4	239.3	146.1	1.46	412.8
2	228	239.3	18.2	332.5	344.1	325.9	1.43	256.6
3	227	344.1	26.1	324.5	359.4	333.3	1.47	241.5
4	220	359.4	27.3	323.3	347.9	320.6	1.46	252.7
5	223	347.9	26.4	324.2	362.8	336.4	1.51	238.3
6	214	362.8	27.6	323.1	338.0	310.4	1.45	263.1
7	224	338.0	25.7	325.0	345.5	319.8	1.43	255.2
8	227	345.5	26.2	324.4	347.4	321.2	1.41	253.3
9	229	347.4	26.4	324.2	349.5	323.1	1.41	251.1
10	230	349.5	26.5	324.1	349.6	323.1	1.40	251.0
11	231	349.6	26.6	324.1	348.9	322.3	1.40	251.7
12	232	348.9	26.5	324.1	352.1	325.6	1.40	248.5
				Average of Last Three Stages:			1.40	250.4

Standard Bond Ball Mill Grindability Test

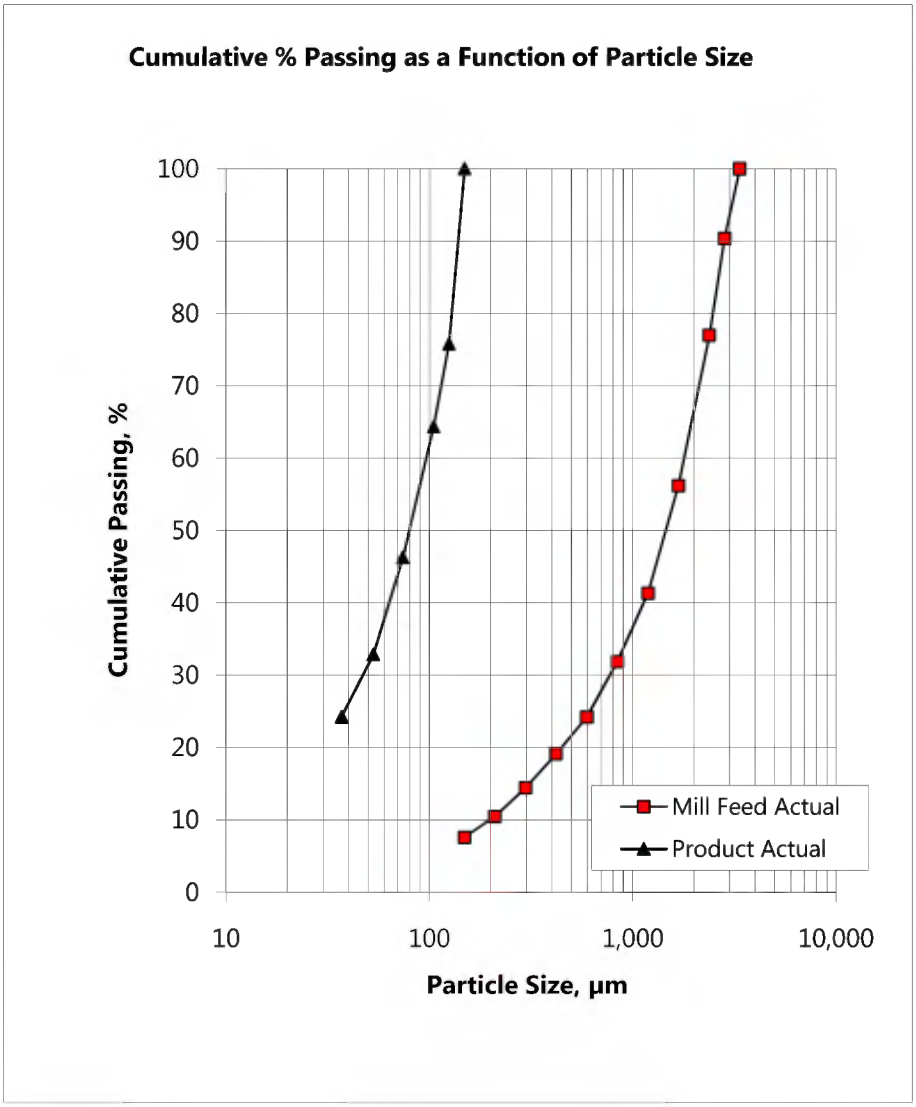
Date: **Aug-10**
Project No: **11066-01**
Test No: **2739**

Detailed Particle Analyses

Screen Size (Retained)		Mill Feed, -6 mesh				Circ. Load, +100 mesh			Test Product, -100 mesh			
Tyler Mesh	µm	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain	g	Wt. %	Pass	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain
6	3,360	0.0	0.0	100.0	0.0	0.0	0.0	100.0				
7	2,830	118.3	9.6	90.4	9.6	20.4	2.3	97.7				
8	2,380	164.4	13.4	77.0	23.0	36.3	4.1	93.5				
10	1,680	254.8	20.8	56.2	43.8	67.4	7.7	85.8				
14	1,190	182.4	14.9	41.3	58.7	53.9	6.2	79.7				
20	841	116.2	9.5	31.9	68.1	42.2	4.8	74.8				
28	595	93.8	7.6	24.2	75.8	50.7	5.8	69.0				
35	420	62.8	5.1	19.1	80.9	62.4	7.1	61.9				
48	297	57.4	4.7	14.4	85.6	113.6	13.0	48.9				
65	210	48.7	4.0	10.5	89.5	208.4	23.8	25.1				
100	149	35.2	2.9	7.6	92.4	219.8	25.1	0.0				
-100	-149	93.2	7.6	0.0	100.0	0.0	0.0	0.0				
	Totals:	1227.2	100.0			875.1	100.0					
100	149								0.0	0.0	100.0	0.0
115	125								49.2	24.2	75.8	24.2
150	105								23.2	11.4	64.3	35.7
200	74								36.7	18.1	46.3	53.7
270	53								27.2	13.4	32.9	67.1
400	37								17.5	8.6	24.2	75.8
-400	-37								49.2	24.2	0.0	100.0
								Totals:	203.0	100.0		

Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2739**



Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2740**

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number, BW_i.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices. The sample was stage crushed to minus 6 mesh. This material was used as feed for the grindability test.

Sample:
Client Identified: Dike 8.7
Sample No: 52461-4

Results:

P_1 = 100% passing size of product	149	µm
P_{80} = 80% passing size of product	132	µm
F_{80} = 80% passing size of feed	2,522	µm
Gpr = Grams per revolution	1.40	g

Calculation of a Bond Ball Mill Work Index:

$$BW_i = \frac{44.5}{P_1^{0.23} \times Gpr^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)}$$

BW_i = 15.9 kWh/st

BW_i = 17.5 kWh/t

Standard Bond Ball Mill Grindability Test

Date: **Aug-10**
Project No: **11066-01**
Test No: **2740**

Test Conditions and Notes

100% passing size of product:		100	mesh
Test feed weight (700 cm^3):		1190.0	g
Percent minus 100 mesh in feed:		7.1	%
Target product weights:	+ 100 mesh	850.0	g
	- 100 mesh	340.0	g

Test Ball Charge

Ball Size in.	Number of Balls	Weight g
1.45	43	8,806
1.17	67	7,209
1.00	10	672
0.75	71	2,009
0.61	94	1,428
Totals:	285	20,125

			Undersize			Undersize Product		
Stage No.	Revolutions	New Feed g	In Feed g	To Be Ground g	U'Size In Product g	Total g	Per Mill Revolutions g	Circ. Load %
1	100	1190.0	84.4	255.6	224.3	139.9	1.40	430.5
2	232	224.3	15.9	324.1	337.1	321.2	1.38	253.0
3	228	337.1	23.9	316.1	352.7	328.8	1.44	237.4
4	218	352.7	25.0	315.0	353.3	328.3	1.51	236.8
5	209	353.3	25.1	314.9	327.1	302.0	1.45	263.8
6	219	327.1	23.2	316.8	335.2	312.0	1.42	255.0
7	222	335.2	23.8	316.2	338.1	314.3	1.42	252.0
8	223	338.1	24.0	316.0	338.5	314.5	1.41	251.6
9	224	338.5	24.0	316.0	337.9	313.9	1.40	252.2
10	226	337.9	24.0	316.0	339.3	315.3	1.40	250.7
11	226	339.3	24.1	315.9	338.3	314.2	1.39	251.8
12	227	338.3	24.0	316.0	346.2	322.2	1.42	243.7
				Average of Last Three Stages:			1.40	248.7

Standard Bond Ball Mill Grindability Test

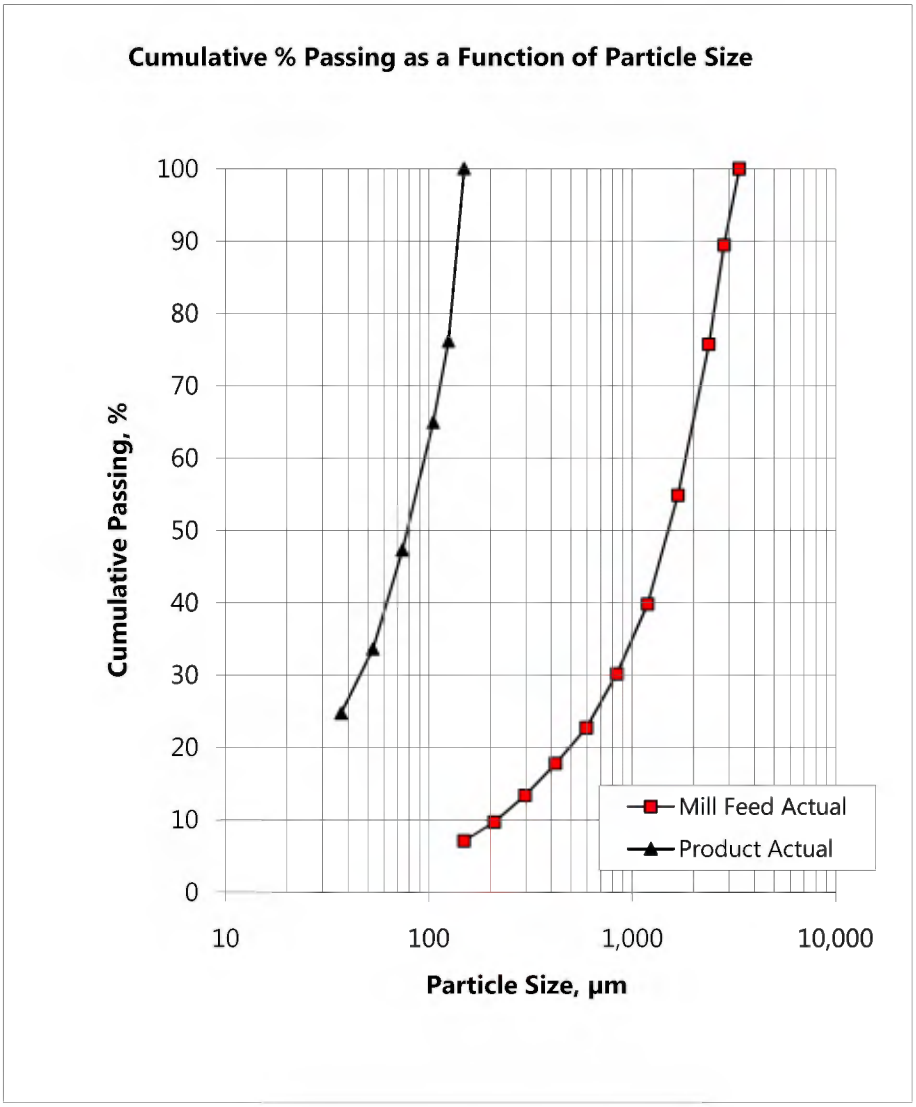
Date: **Aug-10**
Project No: **11066-01**
Test No: **2740**

Detailed Particle Analyses

Screen Size (Retained)		Mill Feed, -6 mesh				Circ. Load, +100 mesh			Test Product, -100 mesh			
Tyler Mesh	µm	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain	g	Wt. %	Pass	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain
6	3,360	0.0	0.0	100.0	0.0	0.0	0.0	100.0				
7	2,830	125.3	10.5	89.5	10.5	20.1	2.4	97.6				
8	2,380	163.6	13.7	75.7	24.3	36.1	4.3	93.3				
10	1,680	248.0	20.8	54.9	45.1	66.7	7.9	85.4				
14	1,190	179.3	15.1	39.8	60.2	53.6	6.4	79.1				
20	841	114.5	9.6	30.2	69.8	41.6	4.9	74.1				
28	595	88.8	7.5	22.7	77.3	50.7	6.0	68.1				
35	420	58.8	4.9	17.8	82.2	63.4	7.5	60.6				
48	297	52.6	4.4	13.4	86.6	118.7	14.1	46.5				
65	210	43.6	3.7	9.7	90.3	222.6	26.4	20.1				
100	149	31.1	2.6	7.1	92.9	169.9	20.1	0.0				
-100	-149	84.4	7.1	0.0	100.0	0.0	0.0	0.0				
	Totals:	1190.0	100.0			843.4	100.0					
100	149								0.0	0.0	100.0	0.0
115	125								48.9	23.9	76.1	23.9
150	105								23.0	11.2	64.9	35.1
200	74								36.2	17.7	47.3	52.7
270	53								28.0	13.7	33.6	66.4
400	37								18.2	8.9	24.7	75.3
-400	-37								50.7	24.7	0.0	100.0
								Totals:	205.0	100.0		

Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2740**



Standard Bond Ball Mill Grindability Test

Date:	**Jan-00**
Project No:	**11066-01**
Test No:	**2741**

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number, BW_i.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices. The sample was stage crushed to minus 6 mesh. This material was used as feed for the grindability test.

Sample:
Client Identified: Dike 9.2
Sample No: 52461-5

Results:

P_1 = 100% passing size of product	149	µm
P_{80} = 80% passing size of product	133	µm
F_{80} = 80% passing size of feed	2,477	µm
Gpr = Grams per revolution	1.41	g

Calculation of a Bond Ball Mill Work Index:

$$BW_i = \frac{44.5}{P_1^{0.23} \times Gpr^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)}$$

BW_i = 15.9 kWh/st

BW_i = 17.6 kWh/t

Standard Bond Ball Mill Grindability Test

Date: **Jan-00**
Project No: **11066-01**
Test No: **2741**

Test Conditions and Notes

100% passing size of product:		100	mesh
Test feed weight (700 cm^3):		1179.8	g
Percent minus 100 mesh in feed:		7.5	%
Target product weights:	+ 100 mesh	842.7	g
	- 100 mesh	337.1	g

Test Ball Charge

Ball Size in.	Number of Balls	Weight g
1.45	43	8,806
1.17	67	7,209
1.00	10	672
0.75	71	2,009
0.61	94	1,428
Totals:	285	20,125

			Undersize			Undersize Product		
Stage No.	Revolutions	New Feed g	In Feed g	To Be Ground g	U'Size In Product g	Total g	Per Mill Revolutions g	Circ. Load %
1	100	1179.8	88.4	248.7	235.2	146.8	1.47	401.6
2	218	235.2	17.6	319.5	328.9	311.3	1.43	258.7
3	219	328.9	24.6	312.4	349.1	324.5	1.48	238.0
4	210	349.1	26.2	310.9	336.6	310.4	1.48	250.5
5	211	336.6	25.2	311.9	334.8	309.6	1.47	252.4
6	213	334.8	25.1	312.0	333.4	308.3	1.45	253.9
7	216	333.4	25.0	312.1	333.2	308.2	1.43	254.1
8	219	333.2	25.0	312.1	336.8	311.8	1.42	250.3
9	219	336.8	25.2	311.8	335.2	310.0	1.42	252.0
10	220	335.2	25.1	312.0	335.2	310.1	1.41	252.0
11	221	335.2	25.1	312.0	334.5	309.4	1.40	252.7
12	223	334.5	25.1	312.0	340.9	315.8	1.42	246.1
				Average of Last Three Stages:			1.41	250.3

Standard Bond Ball Mill Grindability Test

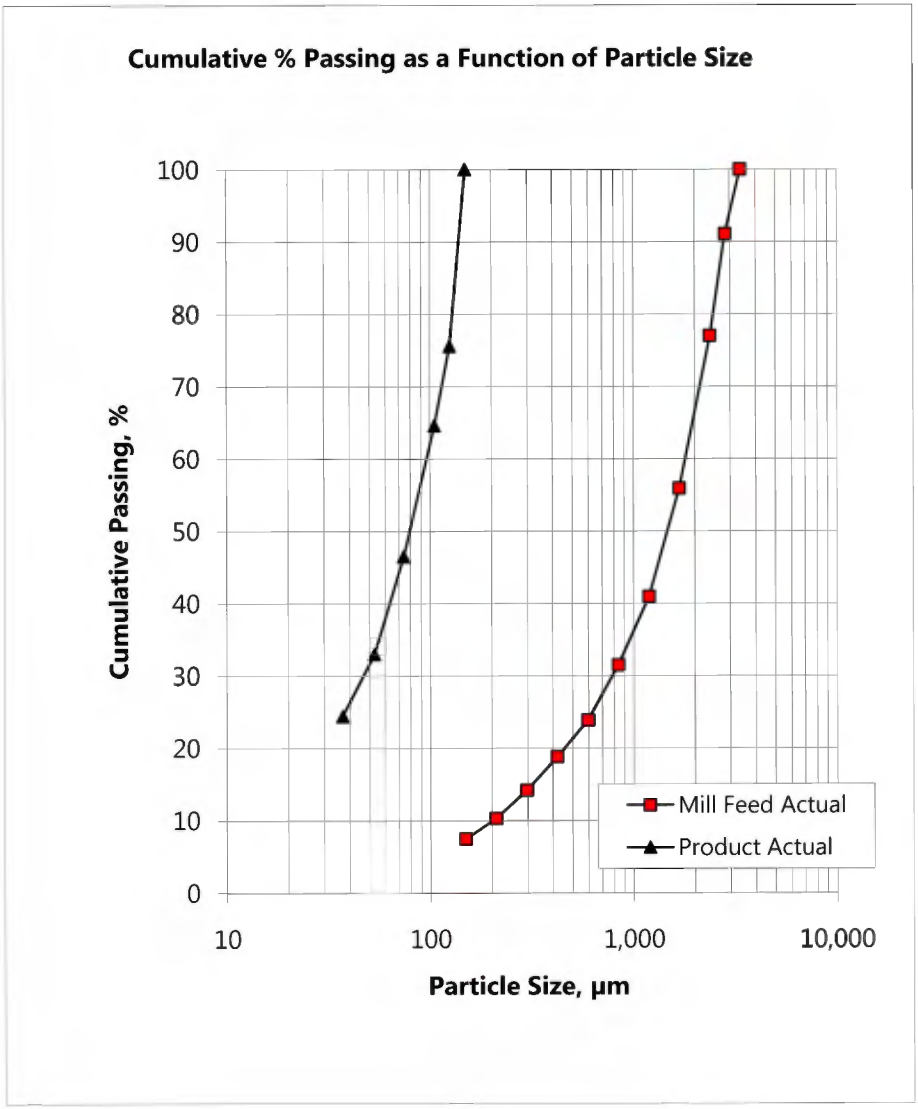
Date: **Jan-00**
Project No: **11066-01**
Test No: **2741**

Detailed Particle Analyses

Screen Size (Retained)		Mill Feed, -6 mesh				Circ. Load, +100 mesh			Test Product, -100 mesh			
Tyler Mesh	µm	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain	g	Wt. %	Pass	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain
6	3,360	0.0	0.0	100.0	0.0	0.0	0.0	100.0				
7	2,830	105.1	8.9	91.1	8.9	20.1	2.4	97.6				
8	2,380	166.1	14.1	77.0	23.0	35.5	4.2	93.4				
10	1,680	248.9	21.1	55.9	44.1	65.5	7.8	85.6				
14	1,190	176.4	15.0	41.0	59.0	51.6	6.2	79.4				
20	841	111.5	9.5	31.5	68.5	40.6	4.8	74.6				
28	595	89.7	7.6	23.9	76.1	48.9	5.8	68.7				
35	420	59.6	5.1	18.9	81.1	59.5	7.1	61.6				
48	297	54.9	4.7	14.2	85.8	108.5	12.9	48.7				
65	210	45.7	3.9	10.3	89.7	195.2	23.3	25.4				
100	149	33.5	2.8	7.5	92.5	213.2	25.4	0.0				
-100	-149	88.4	7.5	0.0	100.0	0.0	0.0	0.0				
	Totals:	1179.8	100.0			838.6	100.0					
100	149								0.0	0.0	100.0	0.0
115	125								49.6	24.4	75.6	24.4
150	105								22.3	11.0	64.6	35.4
200	74								36.7	18.1	46.5	53.5
270	53								27.4	13.5	33.0	67.0
400	37								17.4	8.6	24.4	75.6
-400	-37								49.6	24.4	0.0	100.0
								Totals:	203.0	100.0		

Standard Bond Ball Mill Grindability Test

Date:	**Jan-00**
Project No:	**11066-01**
Test No:	**2741**



Standard Bond Ball Mill Grindability Test

Date:	**Jan-00**
Project No:	**11066-01**
Test No:	**2742**

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number, BW_i.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices. The sample was stage crushed to minus 6 mesh. This material was used as feed for the grindability test.

Sample:
Client Identified: Dike 10.4
Sample No: 52461-6

Results:

P_1 = 100% passing size of product	149	µm
P_{80} = 80% passing size of product	133	µm
F_{80} = 80% passing size of feed	2,402	µm
Gpr = Grams per revolution	1.42	g

Calculation of a Bond Ball Mill Work Index:

$$BW_i = \frac{44.5}{P_1^{0.23} \times Gpr^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)}$$

BW_i = 16.0 kWh/st

BW_i = 17.6 kWh/t

Standard Bond Ball Mill Grindability Test

Date: **Jan-00**
Project No: **11066-01**
Test No: **2742**

Test Conditions and Notes

100% passing size of product:		100	mesh
Test feed weight (700 cm^3):		1202.8	g
Percent minus 100 mesh in feed:		8.3	%
Target product weights:	+ 100 mesh	859.1	g
	- 100 mesh	343.7	g

Test Ball Charge

Ball Size in.	Number of Balls	Weight g
1.45	43	8,806
1.17	67	7,209
1.00	10	672
0.75	71	2,009
0.61	94	1,428
Totals:	285	20,125

			Undersize			Undersize Product		
Stage No.	Revolutions	New Feed g	In Feed g	To Be Ground g	U'Size In Product g	Total g	Per Mill Revolutions g	Circ. Load %
1	100	1202.8	100.4	243.3	248.2	147.8	1.48	384.6
2	219	248.2	20.7	322.9	337.0	316.3	1.44	256.9
3	218	337.0	28.1	315.5	350.7	322.6	1.48	243.0
4	213	350.7	29.3	314.4	345.8	316.5	1.49	247.8
5	212	345.8	28.9	314.8	339.9	311.0	1.47	253.9
6	215	339.9	28.4	315.3	339.2	310.8	1.45	254.6
7	218	339.2	28.3	315.3	342.2	313.9	1.44	251.5
8	219	342.2	28.6	315.1	342.6	314.0	1.43	251.1
9	220	342.6	28.6	315.1	342.8	314.2	1.43	250.9
10	221	342.8	28.6	315.0	343.6	315.0	1.43	250.1
11	221	343.6	28.7	315.0	341.4	312.7	1.42	252.3
12	223	341.4	28.5	315.2	344.3	315.8	1.42	249.3
				Average of Last Three Stages:			1.42	250.6

Standard Bond Ball Mill Grindability Test

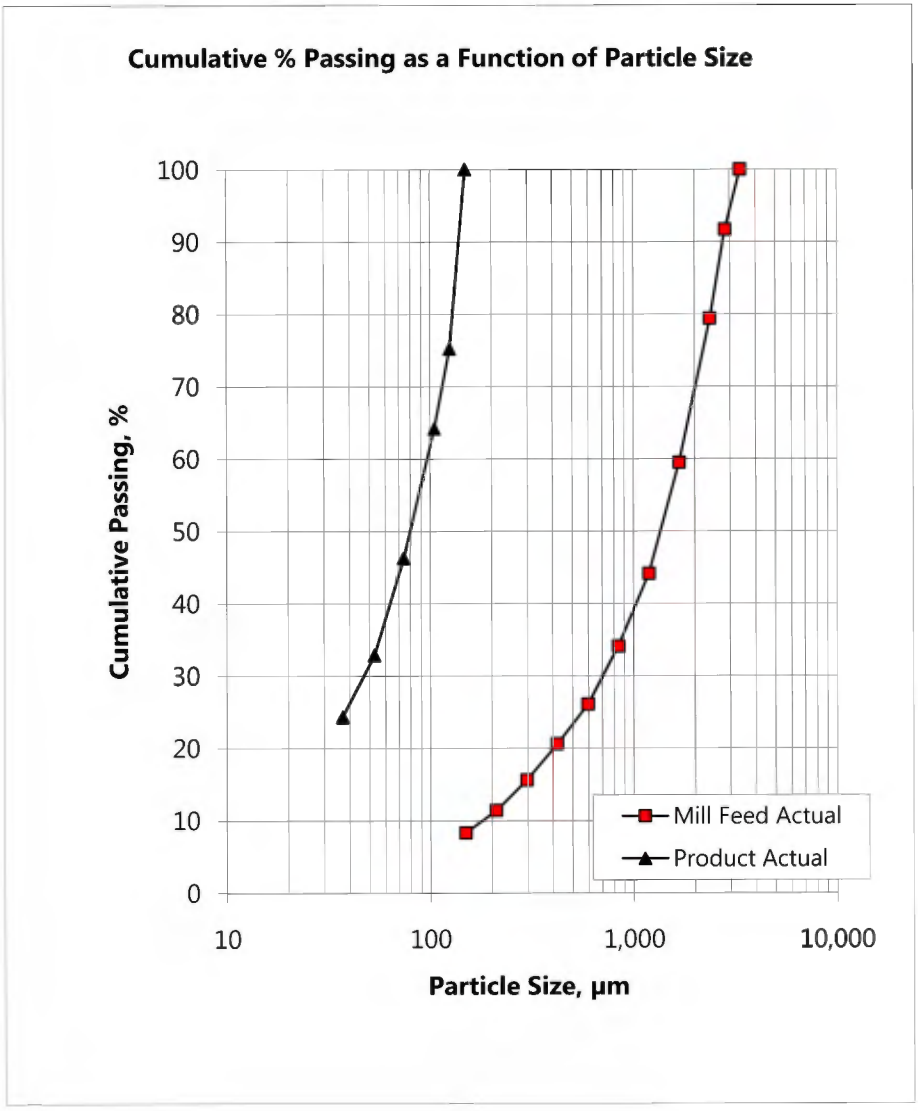
Date: **Jan-00**
Project No: **11066-01**
Test No: **2742**

Detailed Particle Analyses

Screen Size (Retained)		Mill Feed, -6 mesh				Circ. Load, +100 mesh			Test Product, -100 mesh			
Tyler Mesh	µm	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain	g	Wt. %	Pass	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain
6	3,360	0.0	0.0	100.0	0.0	0.0	0.0	100.0				
7	2,830	99.9	8.3	91.7	8.3	14.0	1.6	98.4				
8	2,380	147.8	12.3	79.4	20.6	28.3	3.3	95.1				
10	1,680	239.3	19.9	59.5	40.5	63.9	7.4	87.6				
14	1,190	184.5	15.3	44.2	55.8	54.8	6.4	81.2				
20	841	120.9	10.1	34.1	65.9	43.3	5.0	76.2				
28	595	96.3	8.0	26.1	73.9	50.9	5.9	70.3				
35	420	66.1	5.5	20.6	79.4	62.3	7.3	63.0				
48	297	60.1	5.0	15.6	84.4	111.7	13.0	50.0				
65	210	50.5	4.2	11.4	88.6	204.9	23.9	26.1				
100	149	37.0	3.1	8.3	91.7	224.4	26.1	0.0				
-100	-149	100.4	8.3	0.0	100.0	0.0	0.0	0.0				
	Totals:	1202.8	100.0			858.5	100.0					
100	149								0.0	0.0	100.0	0.0
115	125								50.3	24.8	75.2	24.8
150	105								22.4	11.0	64.2	35.8
200	74								36.3	17.9	46.3	53.7
270	53								27.2	13.4	32.9	67.1
400	37								17.3	8.5	24.3	75.7
-400	-37								49.4	24.3	0.0	100.0
								Totals:	202.9	100.0		

Standard Bond Ball Mill Grindability Test

Date:	**Jan-00**
Project No:	**11066-01**
Test No:	**2742**



Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2743**

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number, BW_i.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices. The sample was stage crushed to minus 6 mesh. This material was used as feed for the grindability test.

Sample:
Client Identified: Dike 11.2
Sample No: 52461-7

Results:

P_1 = 100% passing size of product	149	µm
P_{80} = 80% passing size of product	131	µm
F_{80} = 80% passing size of feed	2,466	µm
Gpr = Grams per revolution	1.34	g

Calculation of a Bond Ball Mill Work Index:

$$BW_i = \frac{44.5}{P_1^{0.23} \times Gpr^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)}$$

BW_i = 16.5 kWh/st

BW_i = 18.2 kWh/t

Standard Bond Ball Mill Grindability Test

Date: **Aug-10**
Project No: **11066-01**
Test No: **2743**

Test Conditions and Notes

100% passing size of product:		100	mesh
Test feed weight (700 cm^3):		1160.5	g
Percent minus 100 mesh in feed:		7.3	%
Target product weights:	+ 100 mesh	828.9	g
	- 100 mesh	331.6	g

Test Ball Charge

Ball Size in.	Number of Balls	Weight g
1.45	43	8,806
1.17	67	7,209
1.00	10	672
0.75	71	2,009
0.61	94	1,428
Totals:	285	20,125

			Undersize			Undersize Product		
Stage No.	Revolutions	New Feed g	In Feed g	To Be Ground g	U'Size In Product g	Total g	Per Mill Revolutions g	Circ. Load %
1	175	1160.5	85.1	246.5	320.2	235.1	1.34	262.4
2	229	320.2	23.5	308.1	343.5	320.0	1.40	237.8
3	219	343.5	25.2	306.4	335.6	310.4	1.42	245.8
4	217	335.6	24.6	307.0	330.2	305.6	1.41	251.5
5	218	330.2	24.2	307.4	326.7	302.5	1.39	255.2
6	222	326.7	24.0	307.6	330.0	306.0	1.38	251.7
7	223	330.0	24.2	307.4	327.9	303.7	1.36	253.9
8	226	327.9	24.0	307.5	328.2	304.2	1.35	253.6
9	229	328.2	24.1	307.5	331.2	307.1	1.34	250.4
10	229	331.2	24.3	307.3	330.7	306.4	1.34	250.9
11	230	330.7	24.3	307.3	331.7	307.4	1.34	249.9
12	230	331.7	24.3	307.2	331.8	307.5	1.34	249.8
				Average of Last Three Stages:			1.34	250.2

Standard Bond Ball Mill Grindability Test

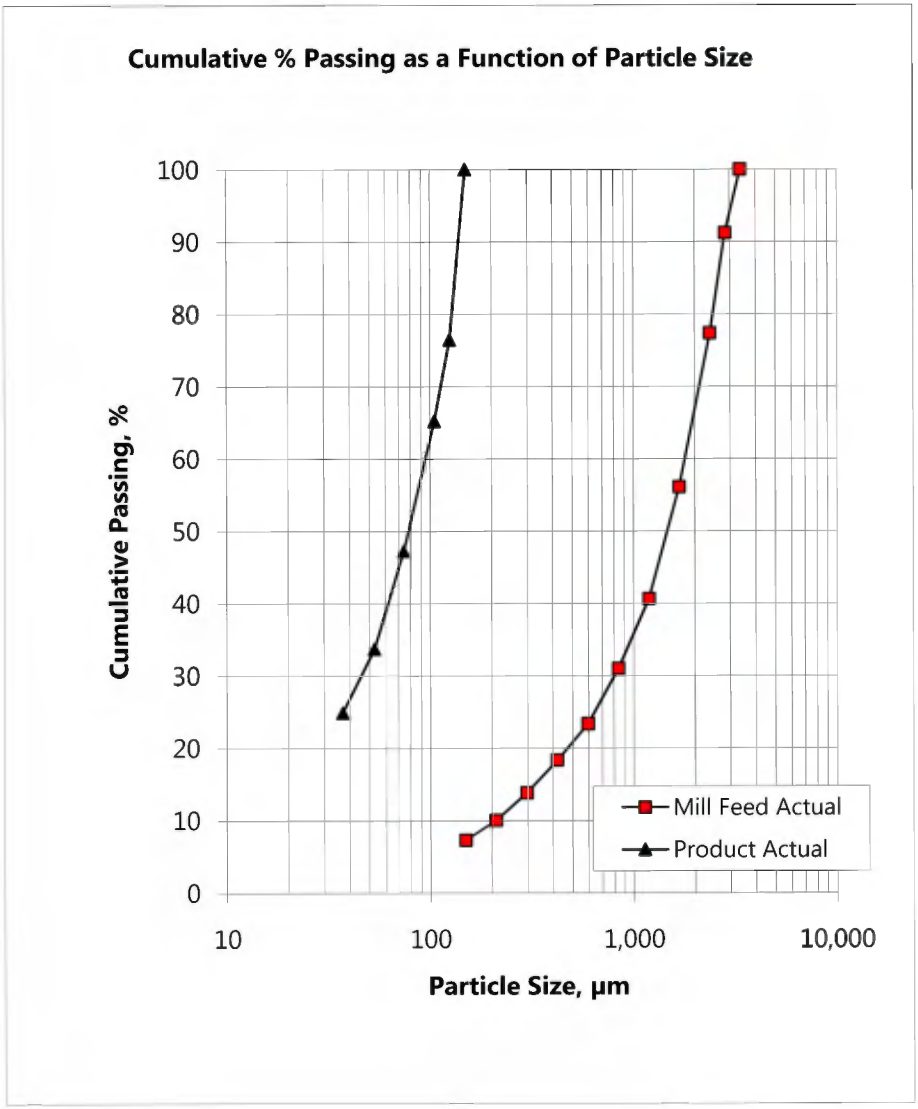
Date: **Aug-10**
Project No: **11066-01**
Test No: **2743**

Detailed Particle Analyses

Screen Size (Retained)		Mill Feed, -6 mesh				Circ. Load, +100 mesh			Test Product, -100 mesh			
Tyler Mesh	µm	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain	g	Wt. %	Pass	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain
6	3,360	0.0	0.0	100.0	0.0	0.0	0.0	100.0				
7	2,830	102.0	8.8	91.2	8.8	19.7	2.4	97.6				
8	2,380	160.4	13.8	77.4	22.6	39.7	4.8	92.8				
10	1,680	247.0	21.3	56.1	43.9	74.8	9.0	83.8				
14	1,190	179.1	15.4	40.7	59.3	59.3	7.2	76.6				
20	841	112.1	9.7	31.0	69.0	43.8	5.3	71.4				
28	595	88.4	7.6	23.4	76.6	49.6	6.0	65.4				
35	420	58.5	5.0	18.4	81.6	58.5	7.1	58.3				
48	297	52.4	4.5	13.8	86.2	102.8	12.4	45.9				
65	210	43.8	3.8	10.1	89.9	183.0	22.1	23.8				
100	149	31.7	2.7	7.3	92.7	197.4	23.8	0.0				
-100	-149	85.1	7.3	0.0	100.0	0.0	0.0	0.0				
	Totals:	1160.5	100.0			828.6	100.0					
100	149								0.0	0.0	100.0	0.0
115	125								47.6	23.6	76.4	23.6
150	105								22.7	11.2	65.2	34.8
200	74								36.3	18.0	47.2	52.8
270	53								27.3	13.5	33.7	66.3
400	37								17.8	8.8	24.9	75.1
-400	-37								50.3	24.9	0.0	100.0
								Totals:	202.0	100.0		

Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2743**



Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2744**

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number, BW_i.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices. The sample was stage crushed to minus 6 mesh. This material was used as feed for the grindability test.

Sample:
Client Identified: Dike 12.2
Sample No: 52461-8

Results:

P_1 = 100% passing size of product	149	µm
P_{80} = 80% passing size of product	130	µm
F_{80} = 80% passing size of feed	2,454	µm
Gpr = Grams per revolution	1.45	g

Calculation of a Bond Ball Mill Work Index:

$$BW_i = \frac{44.5}{P_1^{0.23} \times Gpr^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)}$$

BW_i = 15.4 kWh/st

BW_i = 17.0 kWh/t

Standard Bond Ball Mill Grindability Test

Date: **Aug-10**
Project No: **11066-01**
Test No: **2744**

Test Conditions and Notes

100% passing size of product:		100	mesh
Test feed weight (700 cm^3):		1245.0	g
Percent minus 100 mesh in feed:		7.9	%
Target product weights:	+ 100 mesh	889.3	g
	- 100 mesh	355.7	g

Test Ball Charge

Ball Size in.	Number of Balls	Weight g
1.45	43	8,806
1.17	67	7,209
1.00	10	672
0.75	71	2,009
0.61	94	1,428
Totals:	285	20,125

			Undersize			Undersize Product		
Stage No.	Revolutions	New Feed g	In Feed g	To Be Ground g	U'Size In Product g	Total g	Per Mill Revolutions g	Circ. Load %
1	183	1245.0	97.8	257.9	357.7	259.9	1.42	248.1
2	231	357.7	28.1	327.6	372.4	344.3	1.49	234.3
3	219	372.4	29.3	326.5	361.1	331.8	1.52	244.8
4	216	361.1	28.4	327.3	354.7	326.3	1.51	251.0
5	217	354.7	27.9	327.9	353.1	325.2	1.50	252.6
6	219	353.1	27.7	328.0	351.6	323.9	1.48	254.1
7	222	351.6	27.6	328.1	353.7	326.1	1.47	252.0
8	223	353.7	27.8	327.9	351.4	323.6	1.45	254.3
9	226	351.4	27.6	328.1	353.6	326.0	1.44	252.1
10	227	353.6	27.8	327.9	357.5	329.7	1.45	248.3
				Average of Last Three Stages:			1.45	251.5

Standard Bond Ball Mill Grindability Test

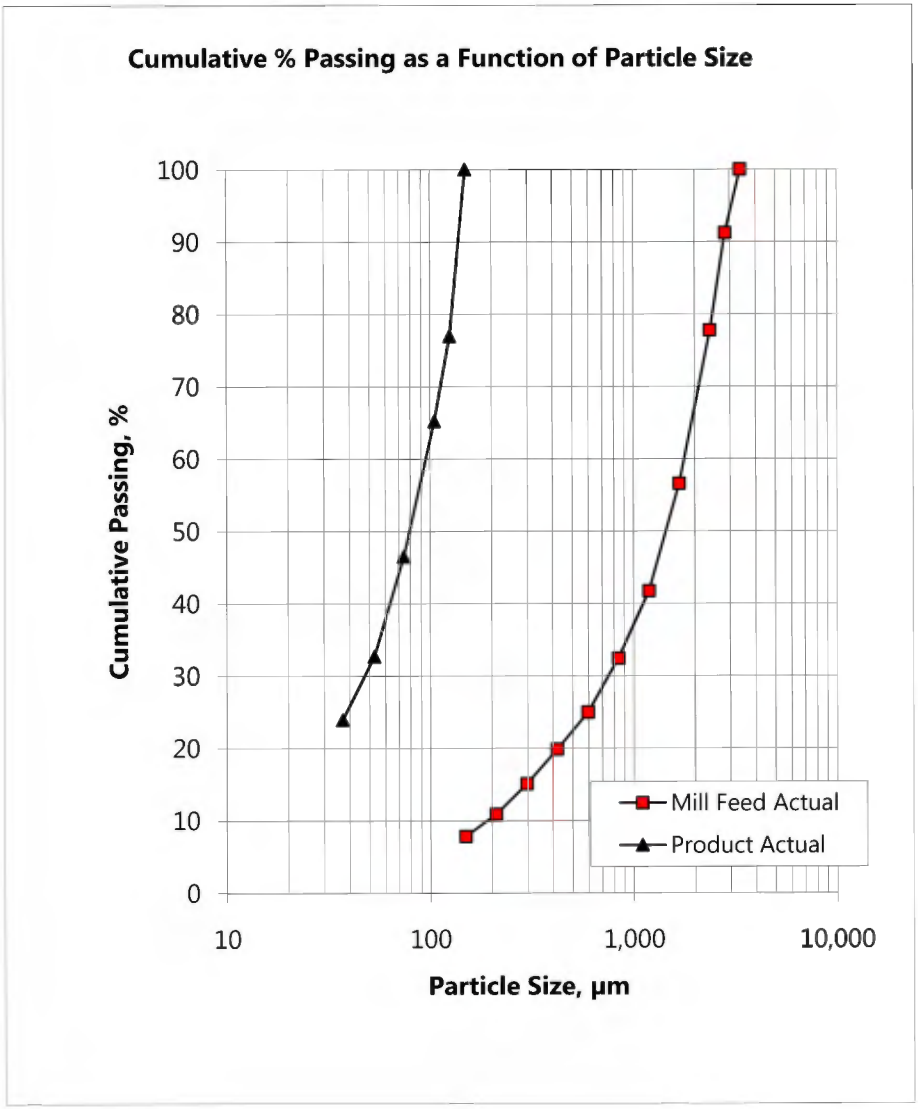
Date: **Aug-10**
Project No: **11066-01**
Test No: **2744**

Detailed Particle Analyses

Screen Size (Retained)		Mill Feed, -6 mesh				Circ. Load, +100 mesh			Test Product, -100 mesh			
Tyler Mesh	µm	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain	g	Wt. %	Pass	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain
6	3,360	0.0	0.0	100.0	0.0	0.0	0.0	100.0				
7	2,830	109.3	8.8	91.2	8.8	14.4	1.6	98.4				
8	2,380	167.4	13.4	77.8	22.2	32.7	3.7	94.7				
10	1,680	263.9	21.2	56.6	43.4	59.3	6.7	88.0				
14	1,190	184.7	14.8	41.7	58.3	48.0	5.4	82.6				
20	841	115.7	9.3	32.4	67.6	37.4	4.2	78.4				
28	595	92.9	7.5	25.0	75.0	46.8	5.3	73.1				
35	420	63.5	5.1	19.9	80.1	60.9	6.9	66.2				
48	297	59.8	4.8	15.1	84.9	123.8	14.0	52.3				
65	210	52.0	4.2	10.9	89.1	249.4	28.1	24.2				
100	149	38.0	3.1	7.9	92.1	214.5	24.2	0.0				
-100	-149	97.8	7.9	0.0	100.0	0.0	0.0	0.0				
	Totals:	1245.0	100.0			887.2	100.0					
100	149								0.0	0.0	100.0	0.0
115	125								49.2	23.1	76.9	23.1
150	105								25.0	11.7	65.2	34.8
200	74								39.8	18.7	46.5	53.5
270	53								29.4	13.8	32.7	67.3
400	37								18.7	8.8	23.9	76.1
-400	-37								51.0	23.9	0.0	100.0
								Totals:	213.1	100.0		

Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2744**



Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2745**

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number, BW_i.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices. The sample was stage crushed to minus 6 mesh. This material was used as feed for the grindability test.

Sample:
Client Identified: Dike 13.2
Sample No: 52461-9

Results:

P_1 = 100% passing size of product	149	µm
P_{80} = 80% passing size of product	131	µm
F_{80} = 80% passing size of feed	2,410	µm
Gpr = Grams per revolution	1.48	g

Calculation of a Bond Ball Mill Work Index:

$$BW_i = \frac{44.5}{P_1^{0.23} \times Gpr^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)}$$

BW_i = 15.3 kWh/st

BW_i = 16.9 kWh/t

Standard Bond Ball Mill Grindability Test

Date: **Aug-10**
Project No: **11066-01**
Test No: **2745**

Test Conditions and Notes

100% passing size of product:		100	mesh
Test feed weight (700 cm^3):		1260.7	g
Percent minus 100 mesh in feed:		8.6	%
Target product weights:	+ 100 mesh	900.5	g
	- 100 mesh	360.2	g

Test Ball Charge

Ball Size in.	Number of Balls	Weight g
1.45	43	8,806
1.17	67	7,209
1.00	10	672
0.75	71	2,009
0.61	94	1,428
Totals:	285	20,125

			Undersize			Undersize Product		
Stage No.	Revolutions	New Feed g	In Feed g	To Be Ground g	U'Size In Product g	Total g	Per Mill Revolutions g	Circ. Load %
1	100	1260.7	108.5	251.7	260.7	152.2	1.52	383.6
2	222	260.7	22.4	337.8	351.9	329.5	1.48	258.3
3	222	351.9	30.3	329.9	368.0	337.7	1.52	242.6
4	216	368.0	31.7	328.5	362.4	330.7	1.53	247.9
5	215	362.4	31.2	329.0	357.5	326.3	1.52	252.6
6	217	357.5	30.8	329.4	358.5	327.7	1.51	251.7
7	218	358.5	30.9	329.3	358.7	327.8	1.50	251.5
8	219	358.7	30.9	329.3	358.5	327.6	1.50	251.7
9	220	358.5	30.9	329.3	353.5	322.6	1.47	256.6
10	225	353.5	30.4	329.8	361.0	330.6	1.47	249.2
					Average of Last Three Stages:		1.48	252.5

Standard Bond Ball Mill Grindability Test

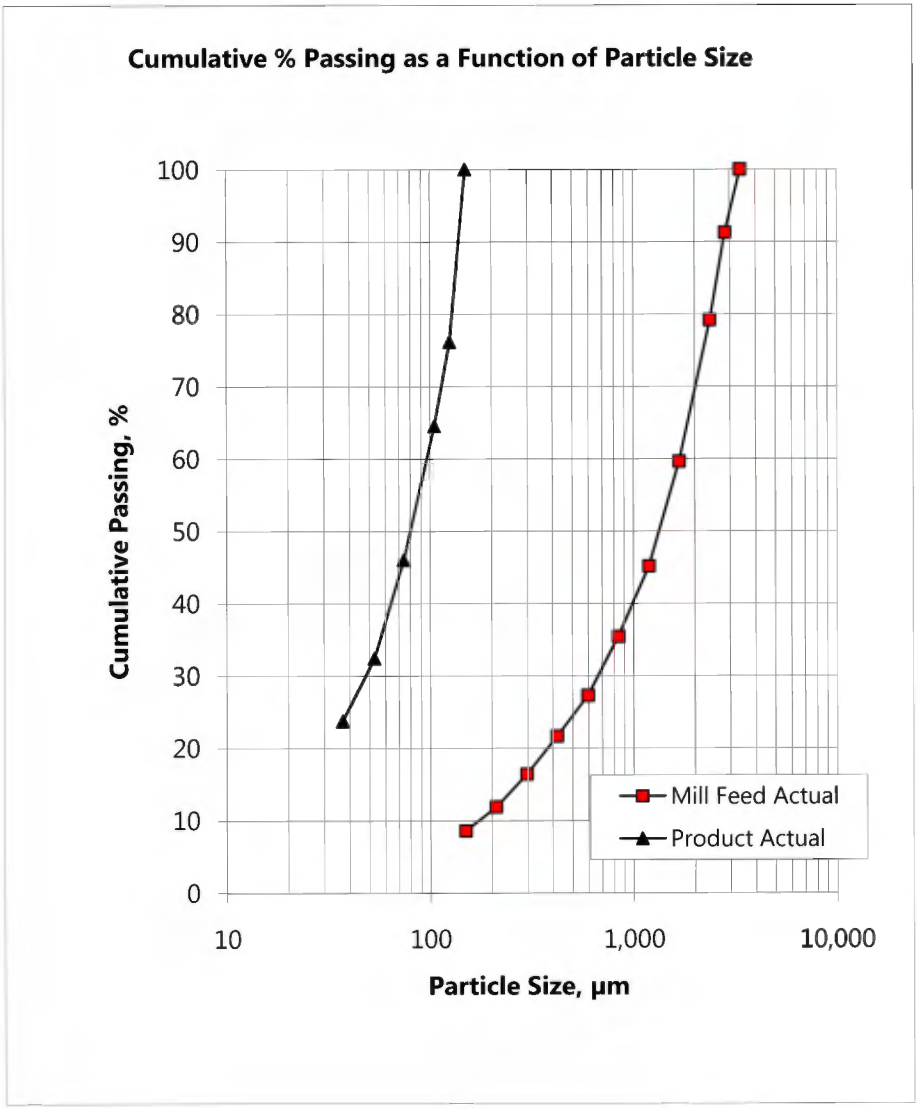
Date: **Aug-10**
Project No: **11066-01**
Test No: **2745**

Detailed Particle Analyses

Screen Size (Retained)		Mill Feed, -6 mesh				Circ. Load, +100 mesh			Test Product, -100 mesh			
Tyler Mesh	µm	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain	g	Wt. %	Pass	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain
6	3,360	0.0	0.0	100.0	0.0	0.0	0.0	100.0				
7	2,830	110.0	8.7	91.3	8.7	18.0	2.0	98.0				
8	2,380	152.3	12.1	79.2	20.8	31.8	3.5	94.5				
10	1,680	246.4	19.5	59.6	40.4	54.9	6.1	88.4				
14	1,190	182.7	14.5	45.2	54.8	45.8	5.1	83.3				
20	841	122.4	9.7	35.4	64.6	36.6	4.1	79.2				
28	595	102.4	8.1	27.3	72.7	47.3	5.3	73.9				
35	420	71.0	5.6	21.7	78.3	61.9	6.9	67.1				
48	297	66.5	5.3	16.4	83.6	121.3	13.5	53.6				
65	210	56.9	4.5	11.9	88.1	235.4	26.2	27.4				
100	149	41.6	3.3	8.6	91.4	246.3	27.4	0.0				
-100	-149	108.5	8.6	0.0	100.0	0.0	0.0	0.0				
	Totals:	1260.7	100.0			899.3	100.0					
100	149								0.0	0.0	100.0	0.0
115	125								48.7	23.9	76.1	23.9
150	105								23.7	11.6	64.5	35.5
200	74								37.8	18.5	46.0	54.0
270	53								27.7	13.6	32.4	67.6
400	37								17.7	8.7	23.8	76.2
-400	-37								48.5	23.8	0.0	100.0
								Totals:	204.1	100.0		

Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2745**



Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2746**

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number, BW_i.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices. The sample was stage crushed to minus 6 mesh. This material was used as feed for the grindability test.

Sample:
Client Identified: Dike 14.2
Sample No: 52461-10

Results:

P_1 = 100% passing size of product	149	µm
P_{80} = 80% passing size of product	132	µm
F_{80} = 80% passing size of feed	2,434	µm
Gpr = Grams per revolution	1.42	g

Calculation of a Bond Ball Mill Work Index:

$$BW_i = \frac{44.5}{P_1^{0.23} \times Gpr^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)}$$

BW_i = 15.8 kWh/st

BW_i = 17.4 kWh/t

Standard Bond Ball Mill Grindability Test

Date: **Aug-10**
Project No: **11066-01**
Test No: **2746**

Test Conditions and Notes

100% passing size of product:		100	mesh
Test feed weight (700 cm^3):		1233.7	g
Percent minus 100 mesh in feed:		8.3	%
Target product weights:	+ 100 mesh	881.2	g
	- 100 mesh	352.5	g

Test Ball Charge

Ball Size in.	Number of Balls	Weight g
1.45	43	8,806
1.17	67	7,209
1.00	10	672
0.75	71	2,009
0.61	94	1,428
Totals:	285	20,125

			Undersize			Undersize Product		
Stage No.	Revolutions	New Feed g	In Feed g	To Be Ground g	U'Size In Product g	Total g	Per Mill Revolutions g	Circ. Load %
1	100	1233.7	102.9	249.6	251.2	148.3	1.48	391.1
2	224	251.2	21.0	331.5	342.6	321.6	1.44	260.1
3	226	342.6	28.6	323.9	361.3	332.7	1.47	241.5
4	219	361.3	30.1	322.4	353.5	323.4	1.48	249.0
5	219	353.5	29.5	323.0	350.9	321.4	1.47	251.6
6	220	350.9	29.3	323.2	349.1	319.8	1.45	253.4
7	222	349.1	29.1	323.4	350.5	321.4	1.45	252.0
8	223	350.5	29.2	323.3	348.6	319.4	1.43	253.9
9	226	348.6	29.1	323.4	349.0	319.9	1.42	253.5
10	228	349.0	29.1	323.4	351.6	322.5	1.41	250.9
11	228	351.6	29.3	323.2	355.1	325.8	1.43	247.4
					Average of Last Three Stages:		1.42	250.6

Standard Bond Ball Mill Grindability Test

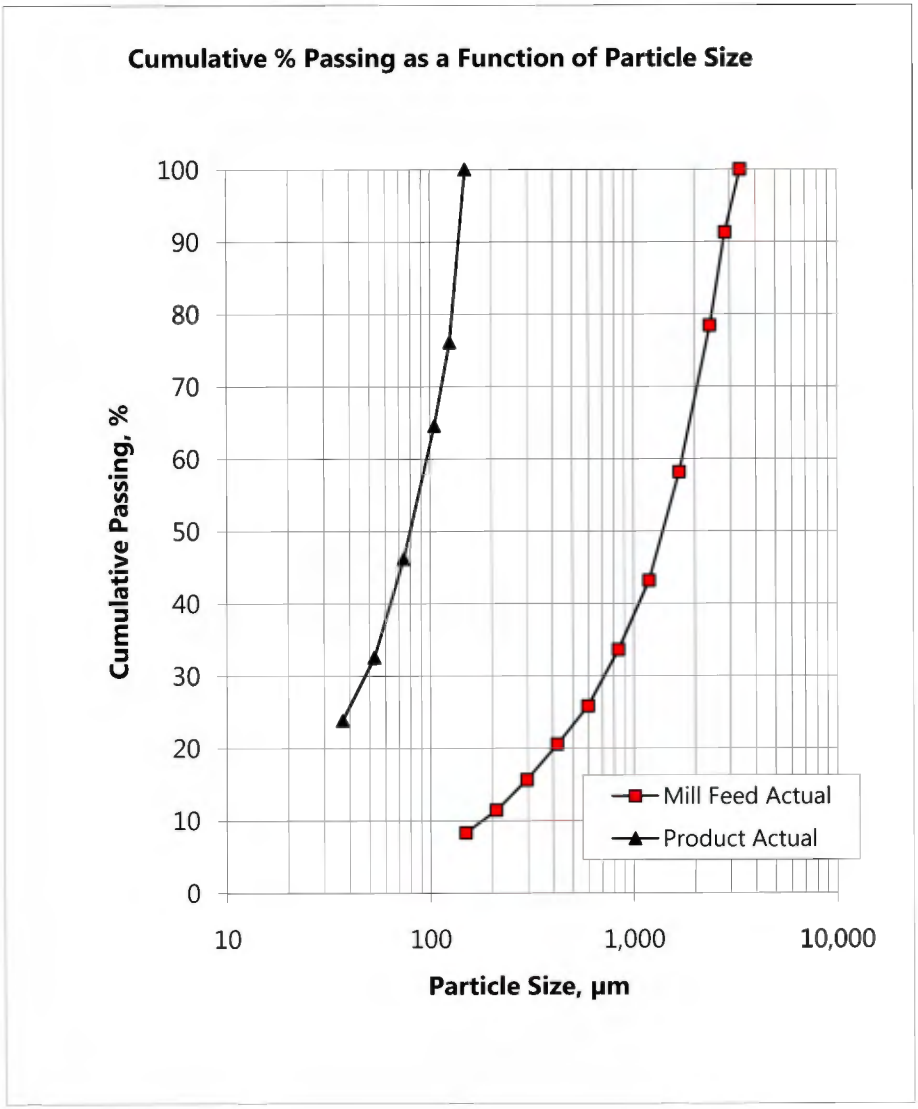
Date: **Aug-10**
Project No: **11066-01**
Test No: **2746**

Detailed Particle Analyses

Screen Size (Retained)		Mill Feed, -6 mesh				Circ. Load, +100 mesh			Test Product, -100 mesh			
Tyler Mesh	µm	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain	g	Wt. %	Pass	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain
6	3,360	0.0	0.0	100.0	0.0	0.0	0.0	100.0				
7	2,830	107.1	8.7	91.3	8.7	18.4	2.1	97.9				
8	2,380	158.7	12.9	78.5	21.5	30.1	3.4	94.5				
10	1,680	250.3	20.3	58.2	41.8	56.1	6.4	88.1				
14	1,190	185.0	15.0	43.2	56.8	46.4	5.3	82.8				
20	841	117.7	9.5	33.6	66.4	38.1	4.3	78.5				
28	595	95.8	7.8	25.9	74.1	47.5	5.4	73.1				
35	420	65.2	5.3	20.6	79.4	60.6	6.9	66.2				
48	297	60.8	4.9	15.7	84.3	115.7	13.2	53.0				
65	210	51.7	4.2	11.5	88.5	221.1	25.2	27.8				
100	149	38.5	3.1	8.3	91.7	244.5	27.8	0.0				
-100	-149	102.9	8.3	0.0	100.0	0.0	0.0	0.0				
	Totals:	1233.7	100.0			878.5	100.0					
100	149								0.0	0.0	100.0	0.0
115	125								48.6	23.9	76.1	23.9
150	105								23.4	11.5	64.6	35.4
200	74								37.3	18.4	46.2	53.8
270	53								27.7	13.6	32.6	67.4
400	37								17.7	8.7	23.9	76.1
-400	-37								48.5	23.9	0.0	100.0
								Totals:	203.2	100.0		

Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2746**



Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2747**

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number, BW_i.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices. The sample was stage crushed to minus 6 mesh. This material was used as feed for the grindability test.

Sample:
Client Identified: Dike 15.1
Sample No: 52461-11

Results:

P_1 = 100% passing size of product	149	µm
P_{80} = 80% passing size of product	132	µm
F_{80} = 80% passing size of feed	2,336	µm
Gpr = Grams per revolution	1.38	g

Calculation of a Bond Ball Mill Work Index:

$$BW_i = \frac{44.5}{P_1^{0.23} \times Gpr^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)}$$

BW_i = 16.3 kWh/st

BW_i = 17.9 kWh/t

Standard Bond Ball Mill Grindability Test

Date: **Aug-10**
Project No: **11066-01**
Test No: **2747**

Test Conditions and Notes

100% passing size of product:		100	mesh
Test feed weight (700 cm^3):		1145.5	g
Percent minus 100 mesh in feed:		8.2	%
Target product weights:	+ 100 mesh	818.2	g
	- 100 mesh	327.3	g

Test Ball Charge

Ball Size in.	Number of Balls	Weight g
1.45	43	8,806
1.17	67	7,209
1.00	10	672
0.75	71	2,009
0.61	94	1,428
Totals:	285	20,125

			Undersize			Undersize Product		
Stage No.	Revolutions	New Feed g	In Feed g	To Be Ground g	U'Size In Product g	Total g	Per Mill Revolutions g	Circ. Load %
1	100	1145.5	94.5	232.8	238.3	143.8	1.44	380.7
2	214	238.3	19.7	307.6	320.5	300.8	1.41	257.4
3	214	320.5	26.4	300.8	333.0	306.6	1.43	244.0
4	209	333.0	27.5	299.8	327.5	300.0	1.44	249.8
5	209	327.5	27.0	300.3	325.3	298.3	1.43	252.1
6	211	325.3	26.8	300.4	324.3	297.5	1.41	253.2
7	213	324.3	26.8	300.5	324.4	297.6	1.40	253.1
8	215	324.4	26.8	300.5	325.3	298.5	1.39	252.1
9	216	325.3	26.8	300.4	326.3	299.5	1.39	251.1
10	217	326.3	26.9	300.4	324.6	297.7	1.37	252.9
11	219	324.6	26.8	300.5	328.5	301.7	1.38	248.7
					Average of Last Three Stages:		1.38	250.9

Standard Bond Ball Mill Grindability Test

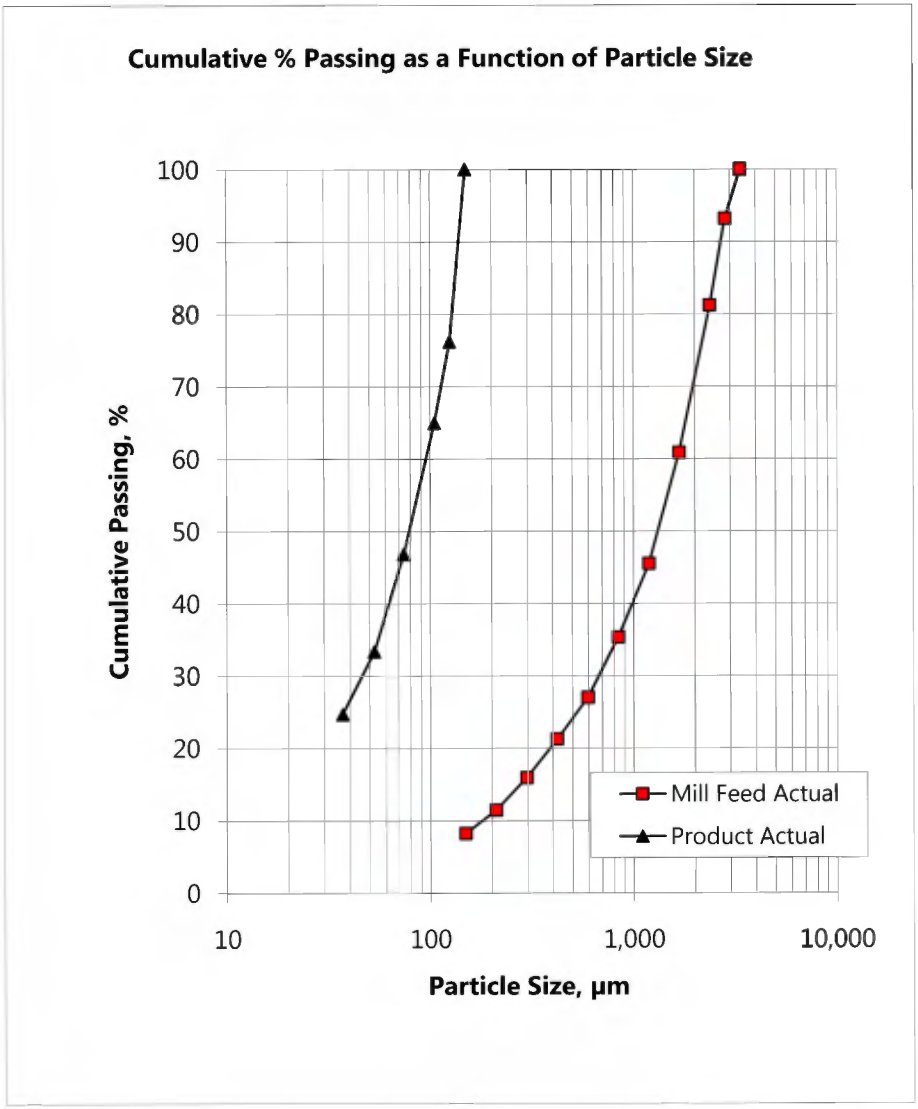
Date: **Aug-10**
Project No: **11066-01**
Test No: **2747**

Detailed Particle Analyses

Screen Size (Retained)		Mill Feed, -6 mesh				Circ. Load, +100 mesh			Test Product, -100 mesh			
Tyler Mesh	µm	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain	g	Wt. %	Pass	g	Wt. %	Cum. Wt. % Pass	Cum. Wt. % Retain
6	3,360	0.0	0.0	100.0	0.0	0.0	0.0	100.0				
7	2,830	77.9	6.8	93.2	6.8	13.6	1.7	98.3				
8	2,380	137.1	12.0	81.2	18.8	30.8	3.8	94.6				
10	1,680	233.0	20.3	60.9	39.1	58.2	7.1	87.4				
14	1,190	176.0	15.4	45.5	54.5	48.2	5.9	81.5				
20	841	116.4	10.2	35.4	64.6	38.2	4.7	76.9				
28	595	95.2	8.3	27.1	72.9	47.2	5.8	71.1				
35	420	66.0	5.8	21.3	78.7	58.3	7.1	63.9				
48	297	61.2	5.3	15.9	84.1	108.2	13.2	50.7				
65	210	51.2	4.5	11.5	88.5	199.1	24.4	26.3				
100	149	37.0	3.2	8.2	91.8	215.0	26.3	0.0				
-100	-149	94.5	8.2	0.0	100.0	0.0	0.0	0.0				
	Totals:	1145.5	100.0			816.8	100.0					
100	149								0.0	0.0	100.0	0.0
115	125								49.5	23.8	76.2	23.8
150	105								23.3	11.2	64.9	35.1
200	74								37.6	18.1	46.8	53.2
270	53								27.9	13.4	33.4	66.6
400	37								18.0	8.7	24.7	75.3
-400	-37								51.3	24.7	0.0	100.0
								Totals:	207.6	100.0		

Standard Bond Ball Mill Grindability Test

Date:	**Aug-10**
Project No:	**11066-01**
Test No:	**2747**



Abrasion Test

Date: **Aug-10**
Project No: **11066**
Test No: **8420**

Purpose: To determine the abrasion index which can be used to determine steel media and liner wear in crushers, rod mills, and ball mills.

Procedure: The equipment and procedure duplicate the Pennsylvania Crusher method for determining an abrasion index.

Sample: Client Identified: Dike 7.2
Sample Number: 52461-1

Results:

Original Coupon Weight	=	94.6079 g
Final Coupon Weight	=	94.142 g
Abrasion Index (A_i)	=	**0.4659 g**

Equipment		Equations[1] (A_i > 0.021)	Wear Rate (lb/kWh)
Wet Rod Mill	Rods =	$0.35(A_i - 0.020)^{0.2}$	= 0.2978
	Liner =	$0.035(A_i - 0.015)^{0.3}$	= 0.0276
Wet Ball Mill	Balls =	$0.35(A_i - 0.015)^{0.33}$	= 0.2691
(overflow and grate discharge)	Liner =	$0.026(A_i - 0.015)^{0.3}$	= 0.0205
Dry Ball Mill	Balls =	$0.05A_i^{0.5}$	= 0.0341
(grate discharge, A_i < 0.22)	Liner =	$0.005A_i^{0.5}$	= 0.0034
Crushers	Liner =	$(A_i + 0.22)/11$	= 0.0624
(gyratory, jaw, and cone)			
Roller Crushers	Roll Shell =	$(0.1A_i)^{0.67}$	= 0.1293

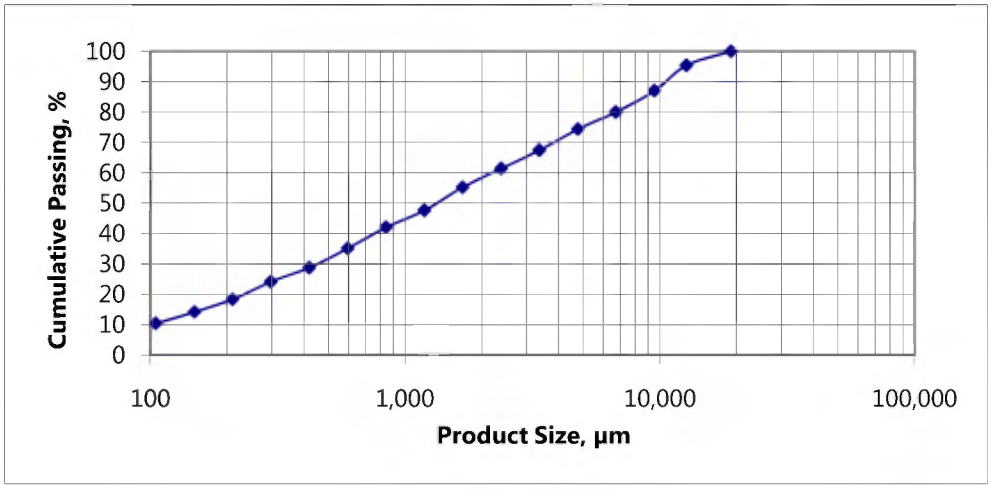
[1] Bond, FC, "Metal Wear in Crushing and Grinding," Allis-Chalmers Publication 07P1701, Dec. 1963.

Abrasion Test

Date: Aug-10
Project No: 11066
Test No: 8420

Tyler Mesh		Direct Wt.		Cumulative %	
in. or mesh	µm	g	%	Passing	Retained
3/4 in.	19,000	0.0	0.0	100.0	0.0
1/2	12,700	74.5	4.7	95.3	4.7
3/8	9,510	133.5	8.3	87.0	13.0
3 mesh	6,700	113.2	7.1	79.9	20.1
4	4,760	89.6	5.6	74.3	25.7
6	3,360	110.6	6.9	67.4	32.6
8	2,380	96.7	6.0	61.4	38.6
10	1,680	99.3	6.2	55.2	44.8
14	1,190	121.3	7.6	47.6	52.4
20	841	89.5	5.6	42.0	58.0
28	595	110.8	6.9	35.1	64.9
35	420	101.7	6.4	28.7	71.3
48	297	74.1	4.6	24.1	75.9
65	210	92.5	5.8	18.3	81.7
100	149	66.5	4.2	14.2	85.8
150	105	60.3	3.8	10.4	89.6
200	74	45.0	2.8	7.6	92.4
-200	-74	121.4	7.6	0.0	100.0
	Total	**1,600.5**	**100.0**		

Product Size Passing 80% (P_{80}) **6,729 µm**



Abrasion Test

Date: **Aug-10**
Project No: **11066**
Test No: **8421**

Purpose: To determine the abrasion index which can be used to determine steel media and liner wear in crushers, rod mills, and ball mills.

Procedure: The equipment and procedure duplicate the Pennsylvania Crusher method for determining an abrasion index.

Sample: Client Identified: Dike 7.6
Sample Number: 52461-2

Results:

Original Coupon Weight	=	94.5872 g
Final Coupon Weight	=	94.1628 g
Abrasion Index (A_i)	=	**0.4244 g**

Equipment		Equations[1] (A_i > 0.021)	Wear Rate (lb/kWh)
Wet Rod Mill	Rods =	$0.35(A_i - 0.020)^{0.2}$	= 0.2920
	Liner =	$0.035(A_i - 0.015)^{0.3}$	= 0.0268
Wet Ball Mill	Balls =	$0.35(A_i - 0.015)^{0.33}$	= 0.2607
(overflow and grate discharge)	Liner =	$0.026(A_i - 0.015)^{0.3}$	= 0.0199
Dry Ball Mill	Balls =	$0.05A_i^{0.5}$	= 0.0326
(grate discharge, A_i < 0.22)	Liner =	$0.005A_i^{0.5}$	= 0.0033
Crushers	Liner =	$(A_i + 0.22)/11$	= 0.0586
(gyratory, jaw, and cone)			
Roller Crushers	Roll Shell =	$(0.1A_i)^{0.67}$	= 0.1215

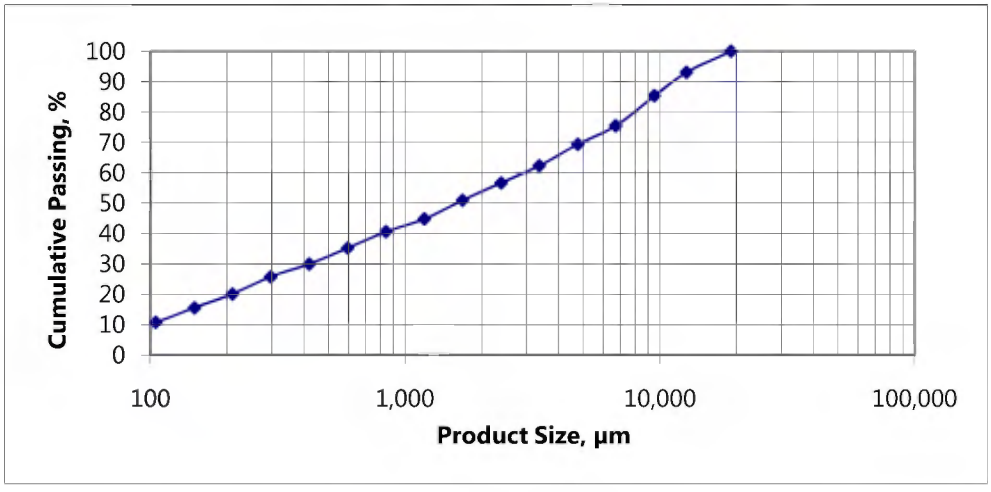
[1] Bond, FC, "Metal Wear in Crushing and Grinding," Allis-Chalmers Publication 07P1701, Dec. 1963.

Abrasion Test

Date: Aug-10
Project No: 11066
Test No: 8421

Tyler Mesh		Direct Wt.		Cumulative %	
in. or mesh	µm	g	%	Passing	Retained
3/4 in.	19,000	0.0	0.0	100.0	0.0
1/2	12,700	110.9	6.9	93.1	6.9
3/8	9,510	123.9	7.8	85.3	14.7
3 mesh	6,700	158.1	9.9	75.4	24.6
4	4,760	97.4	6.1	69.3	30.7
6	3,360	111.8	7.0	62.3	37.7
8	2,380	90.8	5.7	56.6	43.4
10	1,680	92.1	5.8	50.9	49.1
14	1,190	97.9	6.1	44.8	55.2
20	841	67.4	4.2	40.5	59.5
28	595	85.5	5.3	35.2	64.8
35	420	84.2	5.3	29.9	70.1
48	297	66.7	4.2	25.7	74.3
65	210	90.5	5.7	20.1	79.9
100	149	72.7	4.5	15.5	84.5
150	105	76.4	4.8	10.8	89.2
200	74	41.9	2.6	8.1	91.9
-200	-74	130.0	8.1	0.0	100.0
	Total	**1,598.2**	**100.0**		

Product Size Passing 80% (P_{80}) **8,099 µm**



Abrasion Test

Date: **Aug-10**
Project No: **11066**
Test No: **8422**

Purpose: To determine the abrasion index which can be used to determine steel media and liner wear in crushers, rod mills, and ball mills.

Procedure: The equipment and procedure duplicate the Pennsylvania Crusher method for determining an abrasion index.

Sample: Client Identified: Dike 8.3
Sample Number: 52461-3

Results:

Original Coupon Weight	=	94.142 g
Final Coupon Weight	=	93.6977 g
Abrasion Index (A_i)	=	**0.4443 g**

Equipment		Equations[1] (A_i > 0.021)	Wear Rate (lb/kWh)
Wet Rod Mill	Rods =	$0.35(A_i - 0.020)^{0.2}$	= 0.2949
	Liner =	$0.035(A_i - 0.015)^{0.3}$	= 0.0272
Wet Ball Mill	Balls =	$0.35(A_i - 0.015)^{0.33}$	= 0.2648
(overflow and grate discharge)	Liner =	$0.026(A_i - 0.015)^{0.3}$	= 0.0202
Dry Ball Mill	Balls =	$0.05A_i^{0.5}$	= 0.0333
(grate discharge, A_i < 0.22)	Liner =	$0.005A_i^{0.5}$	= 0.0033
Crushers	Liner =	$(A_i + 0.22)/11$	= 0.0604
(gyratory, jaw, and cone)			
Roller Crushers	Roll Shell =	$(0.1A_i)^{0.67}$	= 0.1253

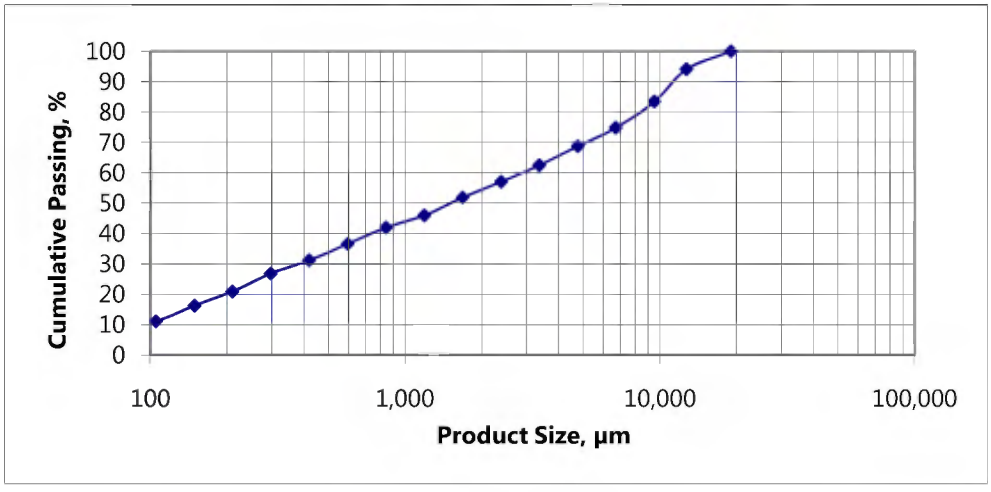
[1] Bond, FC, "Metal Wear in Crushing and Grinding," Allis-Chalmers Publication 07P1701, Dec. 1963.

Abrasion Test

Date: Aug-10
Project No: 11066
Test No: 8422

Tyler Mesh		Direct Wt.		Cumulative %	
in. or mesh	µm	g	%	Passing	Retained
3/4 in.	19,000	0.0	0.0	100.0	0.0
1/2	12,700	95.7	6.0	94.0	6.0
3/8	9,510	170.9	10.6	83.4	16.6
3 mesh	6,700	138.6	8.6	74.8	25.2
4	4,760	97.7	6.1	68.7	31.3
6	3,360	99.9	6.2	62.5	37.5
8	2,380	87.4	5.4	57.0	43.0
10	1,680	84.3	5.2	51.8	48.2
14	1,190	93.9	5.8	46.0	54.0
20	841	65.1	4.1	41.9	58.1
28	595	85.8	5.3	36.6	63.4
35	420	86.4	5.4	31.2	68.8
48	297	70.0	4.4	26.8	73.2
65	210	94.8	5.9	20.9	79.1
100	149	75.1	4.7	16.3	83.7
150	105	83.5	5.2	11.1	88.9
200	74	39.5	2.5	8.6	91.4
-200	-74	138.2	8.6	0.0	100.0
	Total	**1,606.8**	**100.0**		

Product Size Passing 80% (P_{80}) **8,447 µm**



Abrasion Test

Date: **Aug-10**
Project No: **11066**
Test No: **8423**

Purpose: To determine the abrasion index which can be used to determine steel media and liner wear in crushers, rod mills, and ball mills.

Procedure: The equipment and procedure duplicate the Pennsylvania Crusher method for determining an abrasion index.

Sample: Client Identified: Dike 8.7
Sample Number: 52461-4

Results:

Original Coupon Weight	=	94.1628 g
Final Coupon Weight	=	93.7692 g
Abrasion Index (A_i)	=	**0.3936 g**

Equipment		Equations[1] (A_i > 0.021)	Wear Rate (lb/kWh)
Wet Rod Mill	Rods =	$0.35(A_i - 0.020)^{0.2}$	= 0.2874
	Liner =	$0.035(A_i - 0.015)^{0.3}$	= 0.0262
Wet Ball Mill	Balls =	$0.35(A_i - 0.015)^{0.33}$	= 0.2540
(overflow and grate discharge)	Liner =	$0.026(A_i - 0.015)^{0.3}$	= 0.0194
Dry Ball Mill	Balls =	$0.05A_i^{0.5}$	= 0.0314
(grate discharge, A_i < 0.22)	Liner =	$0.005A_i^{0.5}$	= 0.0031
Crushers	Liner =	$(A_i + 0.22)/11$	= 0.0558
(gyratory, jaw, and cone)			
Roller Crushers	Roll Shell =	$(0.1A_i)^{0.67}$	= 0.1156

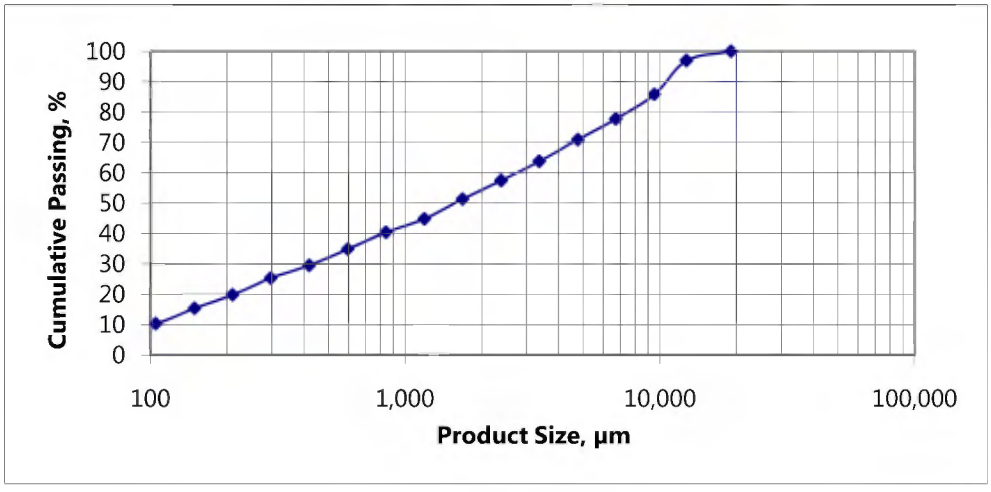
[1] Bond, FC, "Metal Wear in Crushing and Grinding," Allis-Chalmers Publication 07P1701, Dec. 1963.

Abrasion Test

Date: Aug-10
Project No: 11066
Test No: 8423

Tyler Mesh		Direct Wt.		Cumulative %	
in. or mesh	µm	g	%	Passing	Retained
3/4 in.	19,000	0.0	0.0	100.0	0.0
1/2	12,700	49.6	3.1	96.9	3.1
3/8	9,510	177.9	11.1	85.8	14.2
3 mesh	6,700	129.7	8.1	77.7	22.3
4	4,760	108.9	6.8	70.9	29.1
6	3,360	114.3	7.1	63.8	36.2
8	2,380	101.7	6.3	57.4	42.6
10	1,680	98.3	6.1	51.3	48.7
14	1,190	104.0	6.5	44.8	55.2
20	841	70.8	4.4	40.4	59.6
28	595	88.8	5.5	34.8	65.2
35	420	85.4	5.3	29.5	70.5
48	297	67.0	4.2	25.3	74.7
65	210	88.9	5.5	19.8	80.2
100	149	71.0	4.4	15.4	84.6
150	105	81.0	5.1	10.3	89.7
200	74	33.8	2.1	8.2	91.8
-200	-74	131.3	8.2	0.0	100.0
	Total	**1,602.4**	**100.0**		

Product Size Passing 80% (P_{80}) **7,472 µm**



Abrasion Test

Date: Aug-10
Project No: 11066
Test No: 8424

Purpose: To determine the abrasion index which can be used to determine steel media and liner wear in crushers, rod mills, and ball mills.

Procedure: The equipment and procedure duplicate the Pennsylvania Crusher method for determining an abrasion index.

Sample: Client Identified: Dike 9.2
Sample Number: 52461-5

Results:

Original Coupon Weight	=	94.6069 g
Final Coupon Weight	=	94.1918 g
Abrasion Index (A_i)	=	**0.4151 g**

Equipment		Equations[1] (A_i > 0.021)	Wear Rate (lb/kWh)
Wet Rod Mill	Rods =	$0.35(A_i - 0.020)^{0.2}$	= 0.2907
	Liner =	$0.035(A_i - 0.015)^{0.3}$	= 0.0266
Wet Ball Mill	Balls =	$0.35(A_i - 0.015)^{0.33}$	= 0.2587
(overflow and grate discharge)	Liner =	$0.026(A_i - 0.015)^{0.3}$	= 0.0198
Dry Ball Mill	Balls =	$0.05A_i^{0.5}$	= 0.0322
(grate discharge, A_i < 0.22)	Liner =	$0.005A_i^{0.5}$	= 0.0032
Crushers	Liner =	$(A_i + 0.22)/11$	= 0.0577
(gyratory, jaw, and cone)			
Roller Crushers	Roll Shell =	$(0.1A_i)^{0.67}$	= 0.1198

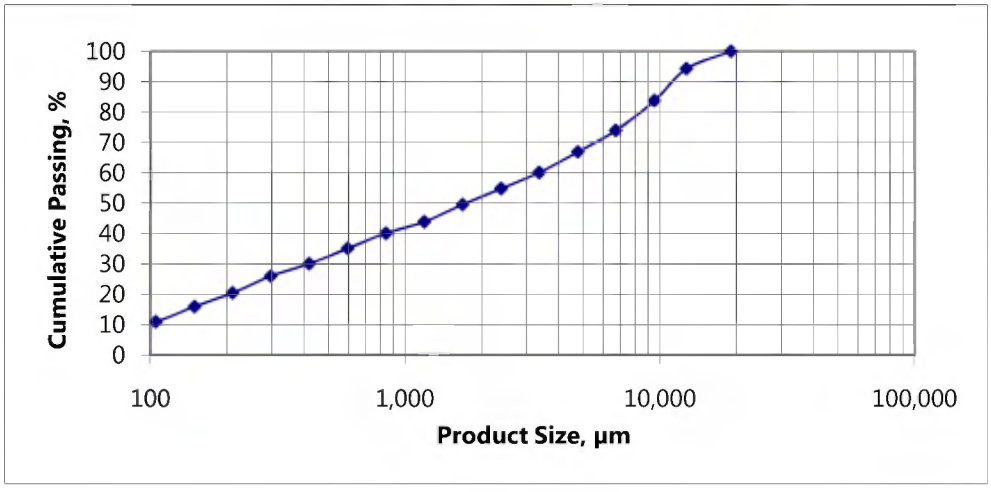
[1] Bond, FC, "Metal Wear in Crushing and Grinding," Allis-Chalmers Publication 07P1701, Dec. 1963.

Abrasion Test

Date: Aug-10
Project No: 11066
Test No: 8424

Tyler Mesh		Direct Wt.		Cumulative %	
in. or mesh	µm	g	%	Passing	Retained
3/4 in.	19,000	0.0	0.0	100.0	0.0
1/2	12,700	92.8	5.8	94.2	5.8
3/8	9,510	168.9	10.5	83.7	16.3
3 mesh	6,700	157.8	9.8	73.9	26.1
4	4,760	113.4	7.1	66.8	33.2
6	3,360	108.7	6.8	60.1	39.9
8	2,380	84.8	5.3	54.8	45.2
10	1,680	85.5	5.3	49.5	50.5
14	1,190	90.2	5.6	43.8	56.2
20	841	61.2	3.8	40.0	60.0
28	595	80.1	5.0	35.1	64.9
35	420	80.7	5.0	30.0	70.0
48	297	65.4	4.1	26.0	74.0
65	210	89.5	5.6	20.4	79.6
100	149	72.0	4.5	15.9	84.1
150	105	80.3	5.0	10.9	89.1
200	74	40.8	2.5	8.4	91.6
-200	-74	134.4	8.4	0.0	100.0
	Total	**1,606.5**	**100.0**		

Product Size Passing 80% (P_{80}) **8,474 µm**



Abrasion Test

Date: Aug-10
Project No: 11066
Test No: 8425

Purpose: To determine the abrasion index which can be used to determine steel media and liner wear in crushers, rod mills, and ball mills.

Procedure: The equipment and procedure duplicate the Pennsylvania Crusher method for determining an abrasion index.

Sample: Client Identified: Dike 10.4
Sample Number: 52461-6

Results:

Original Coupon Weight	=	94.5236 g
Final Coupon Weight	=	94.1012 g
Abrasion Index (A_i)	=	**0.4224 g**

Equipment		Equations[1] (A_i > 0.021)	Wear Rate (lb/kWh)
Wet Rod Mill	Rods =	$0.35(A_i - 0.020)^{0.2}$	= 0.2917
	Liner =	$0.035(A_i - 0.015)^{0.3}$	= 0.0267
Wet Ball Mill	Balls =	$0.35(A_i - 0.015)^{0.33}$	= 0.2602
(overflow and grate discharge)	Liner =	$0.026(A_i - 0.015)^{0.3}$	= 0.0199
Dry Ball Mill	Balls =	$0.05A_i^{0.5}$	= 0.0325
(grate discharge, A_i < 0.22)	Liner =	$0.005A_i^{0.5}$	= 0.0032
Crushers	Liner =	$(A_i + 0.22)/11$	= 0.0584
(gyratory, jaw, and cone)			
Roller Crushers	Roll Shell =	$(0.1A_i)^{0.67}$	= 0.1212

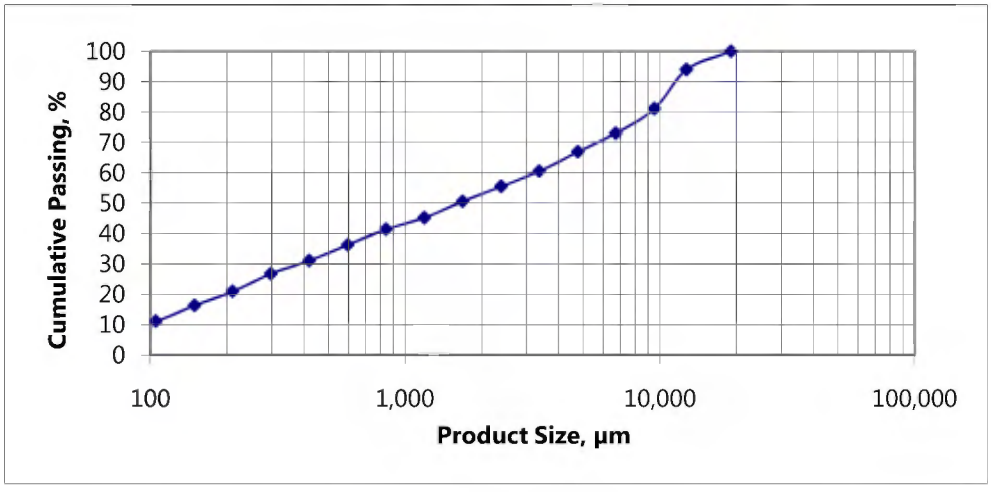
[1] Bond, FC, "Metal Wear in Crushing and Grinding," Allis-Chalmers Publication 07P1701, Dec. 1963.

Abrasion Test

Date: Aug-10
Project No: 11066
Test No: 8425

Tyler Mesh		Direct Wt.		Cumulative %	
in. or mesh	µm	g	%	Passing	Retained
3/4 in.	19,000	0.0	0.0	100.0	0.0
1/2	12,700	97.5	6.1	93.9	6.1
3/8	9,510	206.8	12.8	81.1	18.9
3 mesh	6,700	130.0	8.1	73.0	27.0
4	4,760	98.8	6.1	66.9	33.1
6	3,360	102.5	6.4	60.5	39.5
8	2,380	81.0	5.0	55.5	44.5
10	1,680	79.2	4.9	50.6	49.4
14	1,190	86.3	5.4	45.2	54.8
20	841	61.9	3.8	41.4	58.6
28	595	82.2	5.1	36.3	63.7
35	420	83.6	5.2	31.1	68.9
48	297	69.5	4.3	26.7	73.3
65	210	93.4	5.8	20.9	79.1
100	149	74.8	4.6	16.3	83.7
150	105	83.2	5.2	11.1	88.9
200	74	40.0	2.5	8.6	91.4
-200	-74	139.2	8.6	0.0	100.0
	Total	**1,609.9**	**100.0**		

Product Size Passing 80% (P_{80}) **9,149 µm**



Abrasion Test

Date: **Aug-10**
Project No: **11066**
Test No: **8426**

Purpose: To determine the abrasion index which can be used to determine steel media and liner wear in crushers, rod mills, and ball mills.

Procedure: The equipment and procedure duplicate the Pennsylvania Crusher method for determining an abrasion index.

Sample: Client Identified: Dike 11.2
Sample Number: 52461-7

Results:

Original Coupon Weight	=	94.1918 g
Final Coupon Weight	=	93.7538 g
Abrasion Index (A_i)	=	**0.4380 g**

Equipment		Equations[1] (A_i > 0.021)	Wear Rate (lb/kWh)
Wet Rod Mill	Rods =	$0.35(A_i - 0.020)^{0.2}$	= 0.2940
	Liner =	$0.035(A_i - 0.015)^{0.3}$	= 0.0270
Wet Ball Mill	Balls =	$0.35(A_i - 0.015)^{0.33}$	= 0.2635
(overflow and grate discharge)	Liner =	$0.026(A_i - 0.015)^{0.3}$	= 0.0201
Dry Ball Mill	Balls =	$0.05A_i^{0.5}$	= 0.0331
(grate discharge, A_i < 0.22)	Liner =	$0.005A_i^{0.5}$	= 0.0033
Crushers	Liner =	$(A_i + 0.22)/11$	= 0.0598
(gyratory, jaw, and cone)			
Roller Crushers	Roll Shell =	$(0.1A_i)^{0.67}$	= 0.1241

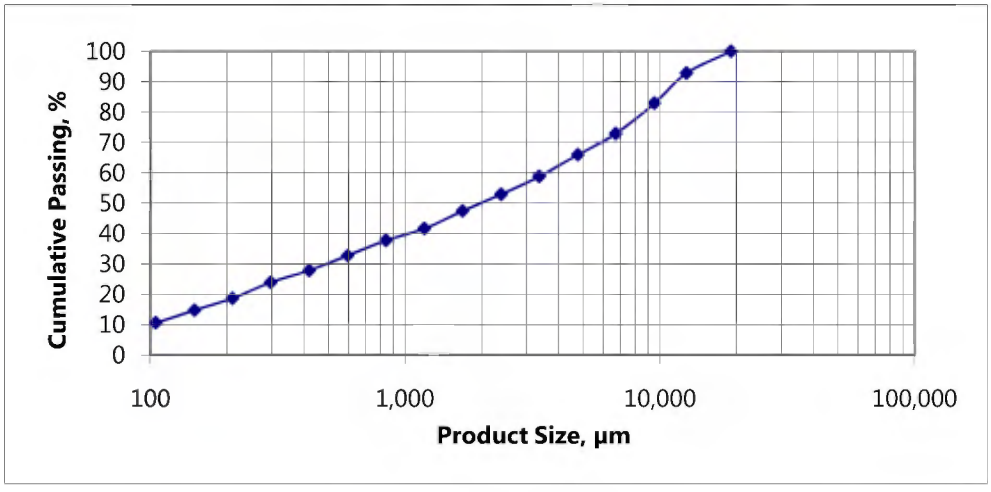
[1] Bond, FC, "Metal Wear in Crushing and Grinding," Allis-Chalmers Publication 07P1701, Dec. 1963.

Abrasion Test

Date: Aug-10
Project No: 11066
Test No: 8426

Tyler Mesh		Direct Wt.		Cumulative %	
in. or mesh	µm	g	%	Passing	Retained
3/4 in.	19,000	0.0	0.0	100.0	0.0
1/2	12,700	116.1	7.2	92.8	7.2
3/8	9,510	158.8	9.9	82.9	17.1
3 mesh	6,700	160.9	10.0	72.9	27.1
4	4,760	111.6	6.9	65.9	34.1
6	3,360	114.7	7.1	58.8	41.2
8	2,380	94.2	5.9	52.9	47.1
10	1,680	88.2	5.5	47.4	52.6
14	1,190	93.2	5.8	41.6	58.4
20	841	62.7	3.9	37.7	62.3
28	595	79.8	5.0	32.7	67.3
35	420	78.6	4.9	27.8	72.2
48	297	63.3	3.9	23.9	76.1
65	210	83.7	5.2	18.7	81.3
100	149	63.5	4.0	14.7	85.3
150	105	65.8	4.1	10.6	89.4
200	74	42.5	2.6	8.0	92.0
-200	-74	128.3	8.0	0.0	100.0
	Total	**1,605.9**	**100.0**		

Product Size Passing 80% (P_{80}) **8,726 µm**



Abrasion Test

Date: **Aug-10**
Project No: **11066**
Test No: **8427**

Purpose: To determine the abrasion index which can be used to determine steel media and liner wear in crushers, rod mills, and ball mills.

Procedure: The equipment and procedure duplicate the Pennsylvania Crusher method for determining an abrasion index.

Sample: Client Identified: Dike 12.2
Sample Number: 52461-8

Results:

Original Coupon Weight	=	94.1012 g
Final Coupon Weight	=	93.6838 g
Abrasion Index (A_i)	=	**0.4174 g**

Equipment		Equations[1] (A_i > 0.021)	Wear Rate (lb/kWh)
Wet Rod Mill	Rods =	$0.35(A_i - 0.020)^{0.2}$	= 0.2910
	Liner =	$0.035(A_i - 0.015)^{0.3}$	= 0.0266
Wet Ball Mill	Balls =	$0.35(A_i - 0.015)^{0.33}$	= 0.2592
(overflow and grate discharge)	Liner =	$0.026(A_i - 0.015)^{0.3}$	= 0.0198
Dry Ball Mill	Balls =	$0.05A_i^{0.5}$	= 0.0323
(grate discharge, A_i < 0.22)	Liner =	$0.005A_i^{0.5}$	= 0.0032
Crushers	Liner =	$(A_i + 0.22)/11$	= 0.0579
(gyratory, jaw, and cone)			
Roller Crushers	Roll Shell =	$(0.1A_i)^{0.67}$	= 0.1202

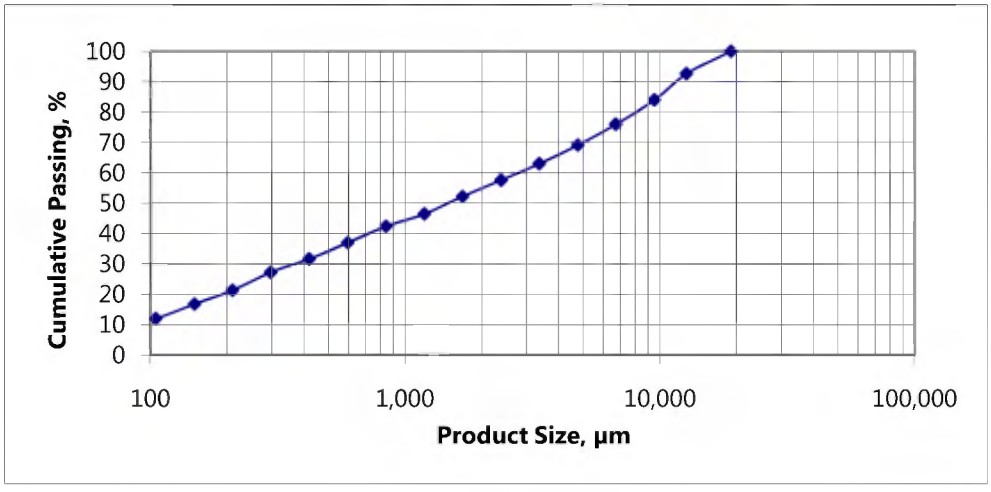
[1] Bond, FC, "Metal Wear in Crushing and Grinding," Allis-Chalmers Publication 07P1701, Dec. 1963.

Abrasion Test

Date: Aug-10
Project No: 11066
Test No: 8427

Tyler Mesh		Direct Wt.		Cumulative %	
in. or mesh	µm	g	%	Passing	Retained
3/4 in.	19,000	0.0	0.0	100.0	0.0
1/2	12,700	119.1	7.4	92.6	7.4
3/8	9,510	139.0	8.6	84.0	16.0
3 mesh	6,700	128.9	8.0	76.0	24.0
4	4,760	110.7	6.9	69.1	30.9
6	3,360	97.5	6.1	63.0	37.0
8	2,380	88.0	5.5	57.6	42.4
10	1,680	86.3	5.4	52.2	47.8
14	1,190	92.6	5.8	46.4	53.6
20	841	66.1	4.1	42.3	57.7
28	595	86.0	5.3	37.0	63.0
35	420	86.4	5.4	31.6	68.4
48	297	70.5	4.4	27.2	72.8
65	210	94.9	5.9	21.3	78.7
100	149	73.7	4.6	16.8	83.2
150	105	76.4	4.7	12.0	88.0
200	74	47.8	3.0	9.0	91.0
-200	-74	145.6	9.0	0.0	100.0
	Total	**1,609.5**	**100.0**		

Product Size Passing 80% (P_{80}) **8,059 µm**



Abrasion Test

Date: **Aug-10**
Project No: **11066**
Test No: **8428**

Purpose: To determine the abrasion index which can be used to determine steel media and liner wear in crushers, rod mills, and ball mills.

Procedure: The equipment and procedure duplicate the Pennsylvania Crusher method for determining an abrasion index.

Sample: Client Identified: Dike 13.2
Sample Number: 52461-9

Results:

Original Coupon Weight	=	94.5238 g
Final Coupon Weight	=	94.1102 g
Abrasion Index (A_i)	=	**0.4136 g**

Equipment		Equations[1] (A_i > 0.021)	Wear Rate (lb/kWh)
Wet Rod Mill	Rods =	$0.35(A_i - 0.020)^{0.2}$	= 0.2905
	Liner =	$0.035(A_i - 0.015)^{0.3}$	= 0.0266
Wet Ball Mill	Balls =	$0.35(A_i - 0.015)^{0.33}$	= 0.2584
(overflow and grate discharge)	Liner =	$0.026(A_i - 0.015)^{0.3}$	= 0.0197
Dry Ball Mill	Balls =	$0.05A_i^{0.5}$	= 0.0322
(grate discharge, A_i < 0.22)	Liner =	$0.005A_i^{0.5}$	= 0.0032
Crushers	Liner =	$(A_i + 0.22)/11$	= 0.0576
(gyratory, jaw, and cone)			
Roller Crushers	Roll Shell =	$(0.1A_i)^{0.67}$	= 0.1195

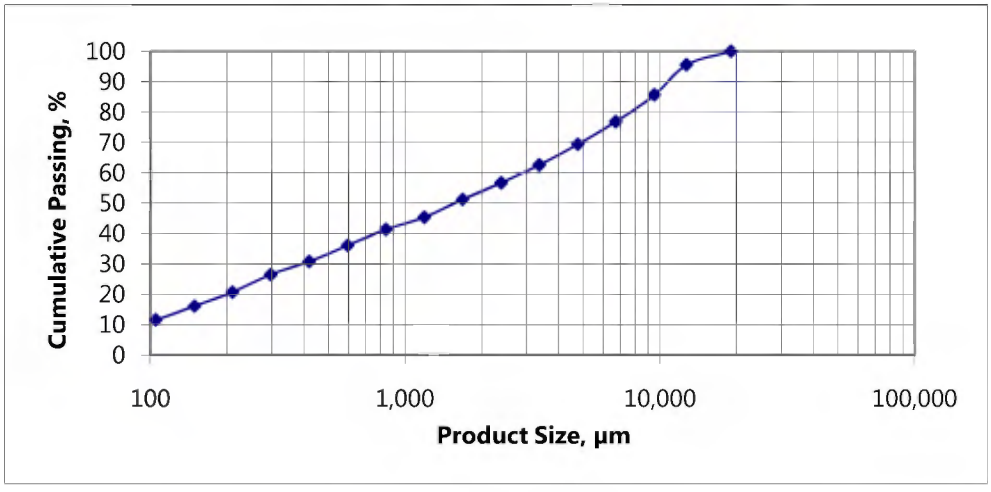
[1] Bond, FC, "Metal Wear in Crushing and Grinding," Allis-Chalmers Publication 07P1701, Dec. 1963.

Abrasion Test

Date: Aug-10
Project No: 11066
Test No: 8428

Tyler Mesh		Direct Wt.		Cumulative %	
in. or mesh	µm	g	%	Passing	Retained
3/4 in.	19,000	0.0	0.0	100.0	0.0
1/2	12,700	71.9	4.5	95.5	4.5
3/8	9,510	158.3	9.9	85.6	14.4
3 mesh	6,700	140.8	8.8	76.8	23.2
4	4,760	119.5	7.5	69.4	30.6
6	3,360	108.8	6.8	62.6	37.4
8	2,380	94.2	5.9	56.7	43.3
10	1,680	88.0	5.5	51.2	48.8
14	1,190	93.2	5.8	45.4	54.6
20	841	64.6	4.0	41.3	58.7
28	595	84.3	5.3	36.1	63.9
35	420	84.7	5.3	30.8	69.2
48	297	68.5	4.3	26.5	73.5
65	210	92.4	5.8	20.7	79.3
100	149	74.1	4.6	16.1	83.9
150	105	72.1	4.5	11.6	88.4
200	74	46.1	2.9	8.7	91.3
-200	-74	139.5	8.7	0.0	100.0
	Total	**1,601.0**	**100.0**		

Product Size Passing 80% (P_{80}) **7,665 µm**



Abrasion Test

Date: **Aug-10**
Project No: **11066**
Test No: **8429**

Purpose: To determine the abrasion index which can be used to determine steel media and liner wear in crushers, rod mills, and ball mills.

Procedure: The equipment and procedure duplicate the Pennsylvania Crusher method for determining an abrasion index.

Sample: Client Identified: Dike 14.2
Sample Number: 52461-10

Results:

Original Coupon Weight	=	94.5647 g
Final Coupon Weight	=	94.1367 g
Abrasion Index (A_i)	=	**0.4280 g**

Equipment		Equations[1] (A_i > 0.021)	Wear Rate (lb/kWh)
Wet Rod Mill	Rods =	$0.35(A_i - 0.020)^{0.2}$	= 0.2925
	Liner =	$0.035(A_i - 0.015)^{0.3}$	= 0.0268
Wet Ball Mill	Balls =	$0.35(A_i - 0.015)^{0.33}$	= 0.2614
(overflow and grate discharge)	Liner =	$0.026(A_i - 0.015)^{0.3}$	= 0.0199
Dry Ball Mill	Balls =	$0.05A_i^{0.5}$	= 0.0327
(grate discharge, A_i < 0.22)	Liner =	$0.005A_i^{0.5}$	= 0.0033
Crushers	Liner =	$(A_i + 0.22)/11$	= 0.0589
(gyratory, jaw, and cone)			
Roller Crushers	Roll Shell =	$(0.1A_i)^{0.67}$	= 0.1222

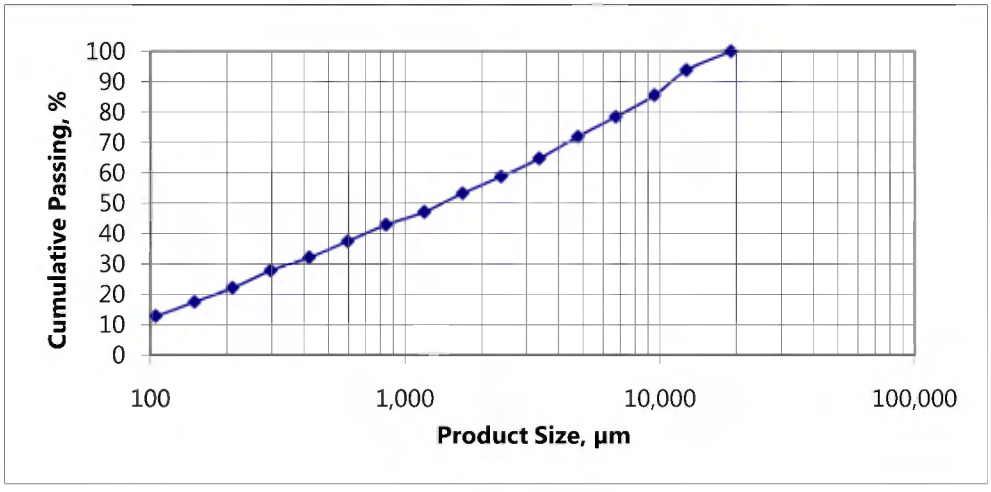
[1] Bond, FC, "Metal Wear in Crushing and Grinding," Allis-Chalmers Publication 07P1701, Dec. 1963.

Abrasion Test

Date: Aug-10
Project No: 11066
Test No: 8429

Tyler Mesh		Direct Wt.		Cumulative %	
in. or mesh	µm	g	%	Passing	Retained
3/4 in.	19,000	0.0	0.0	100.0	0.0
1/2	12,700	101.4	6.3	93.7	6.3
3/8	9,510	133.7	8.2	85.5	14.5
3 mesh	6,700	114.8	7.1	78.4	21.6
4	4,760	106.5	6.6	71.9	28.1
6	3,360	116.1	7.2	64.7	35.3
8	2,380	96.9	6.0	58.7	41.3
10	1,680	89.2	5.5	53.2	46.8
14	1,190	99.2	6.1	47.1	52.9
20	841	69.1	4.3	42.9	57.1
28	595	87.7	5.4	37.4	62.6
35	420	86.7	5.3	32.1	67.9
48	297	69.6	4.3	27.8	72.2
65	210	92.8	5.7	22.1	77.9
100	149	75.1	4.6	17.5	82.5
150	105	75.1	4.6	12.8	87.2
200	74	54.1	3.3	9.5	90.5
-200	-74	153.9	9.5	0.0	100.0
	Total	**1,621.9**	**100.0**		

Product Size Passing 80% (P_{80}) **7,278 µm**



Abrasion Test

Date: **Aug-10**
Project No: **11066**
Test No: **8430**

Purpose: To determine the abrasion index which can be used to determine steel media and liner wear in crushers, rod mills, and ball mills.

Procedure: The equipment and procedure duplicate the Pennsylvania Crusher method for determining an abrasion index.

Sample: Client Identified: Dike 15.1
Sample Number: 52461-11

Results:

Original Coupon Weight	=	94.1102 g
Final Coupon Weight	=	93.7352 g
Abrasion Index (A_i)	=	**0.3750 g**

Equipment		Equations[1] (A_i > 0.021)	Wear Rate (lb/kWh)
Wet Rod Mill	Rods =	$0.35(A_i - 0.020)^{0.2}$	= 0.2845
	Liner =	$0.035(A_i - 0.015)^{0.3}$	= 0.0258
Wet Ball Mill	Balls =	$0.35(A_i - 0.015)^{0.33}$	= 0.2498
(overflow and grate discharge)	Liner =	$0.026(A_i - 0.015)^{0.3}$	= 0.0191
Dry Ball Mill	Balls =	$0.05A_i^{0.5}$	= 0.0306
(grate discharge, A_i < 0.22)	Liner =	$0.005A_i^{0.5}$	= 0.0031
Crushers	Liner =	$(A_i + 0.22)/11$	= 0.0541
(gyratory, jaw, and cone)			
Roller Crushers	Roll Shell =	$(0.1A_i)^{0.67}$	= 0.1119

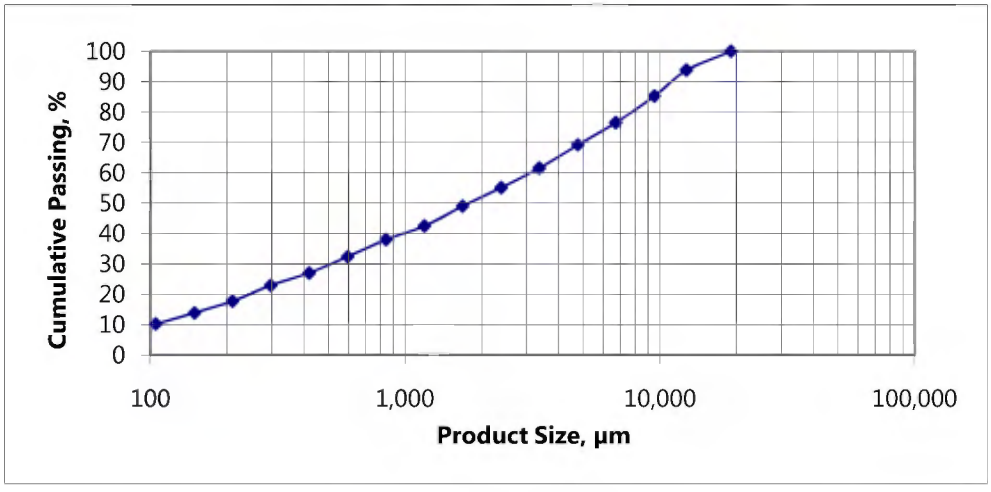
[1] Bond, FC, "Metal Wear in Crushing and Grinding," Allis-Chalmers Publication 07P1701, Dec. 1963.

Abrasion Test

Date: Aug-10
Project No: 11066
Test No: 8430

Tyler Mesh		Direct Wt.		Cumulative %	
in. or mesh	µm	g	%	Passing	Retained
3/4 in.	19,000	0.0	0.0	100.0	0.0
1/2	12,700	100.7	6.3	93.7	6.3
3/8	9,510	135.8	8.5	85.3	14.7
3 mesh	6,700	140.8	8.8	76.5	23.5
4	4,760	117.7	7.3	69.2	30.8
6	3,360	122.1	7.6	61.5	38.5
8	2,380	104.0	6.5	55.1	44.9
10	1,680	98.1	6.1	48.9	51.1
14	1,190	104.7	6.5	42.4	57.6
20	841	71.6	4.5	38.0	62.0
28	595	90.2	5.6	32.3	67.7
35	420	85.9	5.4	27.0	73.0
48	297	65.7	4.1	22.9	77.1
65	210	83.7	5.2	17.7	82.3
100	149	61.6	3.8	13.8	86.2
150	105	58.6	3.7	10.2	89.8
200	74	41.3	2.6	7.6	92.4
-200	-74	122.2	7.6	0.0	100.0
	Total	**1,604.7**	**100.0**		

Product Size Passing 80% (P_{80}) **7,784 µm**



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SMC TEST REPORT

on

Eleven Samples

from

LI1 Project

Tested at

Hazen Research Inc., Golden, Colorado, USA

for

LI1

JKTech Job No. 10012/P12 - August 2010

JKTech Pty Ltd

SMC Test Report

on

Eleven Samples

from

LI1 Project

JKTech Job No. 10012/P12 - August 2010

Submitted to

LI1

Tested at Hazen Research Inc., Golden, Colorado, USA

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1 INTRODUCTION

SMC test data for eleven samples from LI1 Project were received from Hazen Research Inc. on August 24, 2010, by JKTech for data analysis. The samples were identified as 52641-1, 52461-2, 52461-3, 52461-4, 52461-5, 52461-6, 52461-7, 52461-8, 52461-9, 52461-10 and 52461-11. The test results were forwarded to SMC Testing Pty Ltd for analysis. Analysis and reporting were completed on August 25, 2010.

2 THE SMC TEST®

2.1 INTRODUCTION

The standard JKTech drop-weight test provides ore specific parameters for use in the JKSimMet Mineral Processing Simulator software. In JKSimMet, these parameters are combined with equipment details and operating conditions to analyse and/or predict SAG/autogenous mill performance. The same test procedure also provides ore type characterisation for the JKSimMet crusher model.

The SMC (SAG Mill Comminution) test was developed by Steve Morrell of SMC Testing Pty Ltd (SMCT) to provide a cost effective means of obtaining these parameters from drill core or in situations where limited quantities of material are available. The ore specific parameters have been calculated from the test results and are supplied to LI1 in this report as part of the standard procedure.

2.2 GENERAL DESCRIPTION AND TEST BACKGROUND

The SMC Test® was originally designed for the breakage characterisation of drill core and it generates a relationship between input energy (kWh/t) and the percent of broken product passing a specified sieve size. The results are used to determine the so-called drop-weight index (DWi), which is a measure of the strength of the rock when broken under impact conditions and has the units kWh/m^3. The DWi is directly related to the JK rock breakage parameters A and b and hence can be used to estimate the values of these parameters as well as being correlated with the JK abrasion parameter - t_a. For crusher modelling the t_{10}-E_{cs} matrix can also be derived. This is done by using the size-by-size A*b values that are used in the SMC Test® data analysis (see below) to estimate the t_{10}-E_{cs} values for each of the relevant size fractions in the crusher model matrix.

For power-based calculations, (see APPENDIX B), the SMC Test® provides the comminution parameters M_{ia}, M_{ih} and M_{ic}. M_{ia} is the work index for the grinding of coarser particles (> 750 µm) in tumbling mills such as autogenous (AG), semi-autogenous (SAG), rod and ball mills. M_{ih} is the work index for the grinding in High Pressure Grinding Rolls (HPGR) and M_{ic} for size reduction in conventional crushers.

The SMC Test® is a precision test, which uses particles that are either cut from drill core using a diamond saw to achieve close size replication or else selected from crushed material so that particle mass variation is controlled within a prescribed range. The particles are then broken at a number of prescribed impact energies. The high degree of control imposed on both the size of particles and the breakage energies used, means that the test is largely free of the repeatability problems associated with tumbling-mill based tests. Such tests usually suffer from variations in feed size (which is not closely controlled) and energy input, often assumed to be constant when in reality it can be highly variable (Levin, 1989).

The relationship between the DWi and the JK rock breakage parameters makes use of the size-by-size nature of rock strength that is often apparent from the results of full JK Drop-Weight tests. The effect is illustrated in Figure 1, which plots the normalized values of A*b against particle size. This figure also shows how the gradient of these plots varies across the full range of rock types tested. In the case of a conventional Drop-Weight test, these values are effectively averaged and a

mean value of A and b is reported. The SMC Test® uses a single size and makes use of relationships such as that shown in Figure 1 to predict the A and b of the particle size that has the same value as the mean for a full drop-weight test.

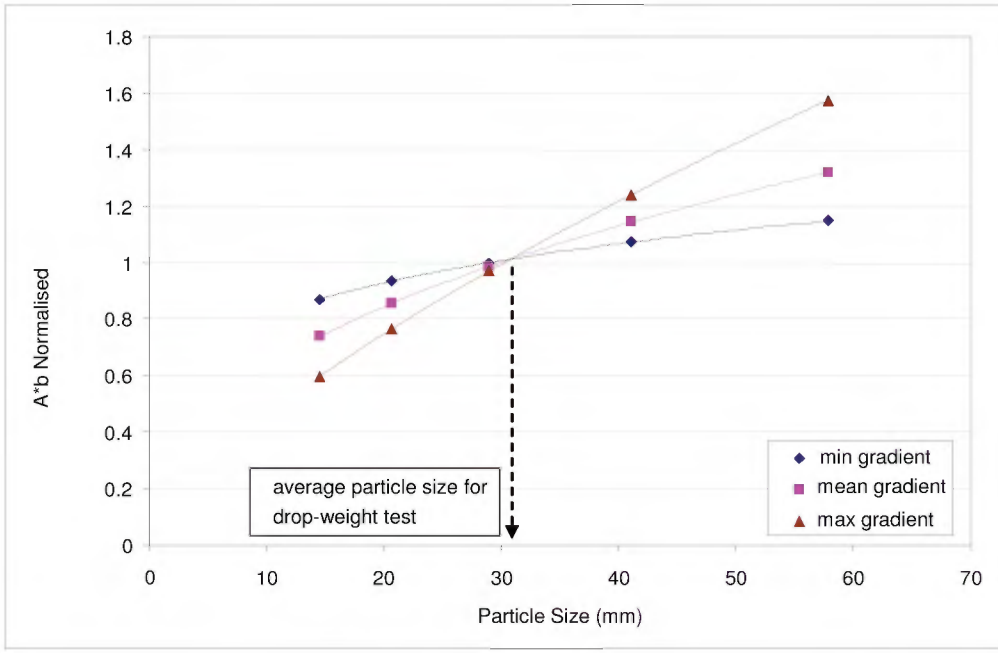

Figure 1 – Relationship between Particle Size and A*b

2.3 THE TEST PROCEDURE

In the SMC Test®, five sets of 20 particles are broken, each set at a different specific energy level, using a JK Drop-Weight tester. The breakage products are screened at a sieve size selected to provide a direct measurement of the t_{10} value.

The test calls for a prescribed target average volume for the particles, with the target being chosen to be equivalent to the mean volume of particles in one of the standard drop-weight test size fractions.

The rest height of the drop-head (gap) is recorded after breakage of each particle to allow for a correction to the drop energy. After breaking all 20 particles in a set, the broken product is sieved at an aperture size, one tenth of the original particle size. Thus, the percent passing mass gives a direct reading of the t_{10} value for breakage at that energy level.

There are two alternative methods of preparing the particle sets for breakage testing: the particle selection method and the cut core method. The particle selection method is the most commonly used as it is generally less time consuming. The cut core method requires less material, so tends to be used as a fallback method, only when necessary to cope with restricted sample availability.

2.3.1 Particle Selection Method

For the particle selection method, the test is carried out on material in one of three alternative size fractions: -31.5+26.5, -22.4+19 or -16+12.4 mm. The largest size fraction is preferred but requires more material. The middle size fraction tends to be

most commonly used, whilst the finest size fraction tends only to be used if there is an issue with starting material size distribution or quantity.

In the particle selection method, particles are chosen so that their individual masses lie within ±30% of the target mass and the mean mass for each set of 20 lies within ±10% of the target mass. A typical set of particles is shown in Figure 2.

Figure 2 – A Typical Set of Particles for Breakage (Particle Selection Method)

Before commencing breakage tests on the particles, the ore density is determined by first weighing a representative sample of particles in air and then in water.

2.3.2 Cut Core Method

The cut core method uses cut pieces of quartered (slivered) drill core. Whole core or half core can be used, but when received in this form it needs to be first quartered as a preliminary step in the procedure. Once quartered, any broken or tapered ends of the quartered lengths are cut, to square them off. Before the lengths of quartered core are cut to produce the pieces for the drop-weight testing, each one is weighed in air and then in water, to obtain a density measurement and a measure of its mass per unit length.

The size fraction targeted when the cut core method is used depends on the original core diameter. The target size fraction is selected to ensure that pieces of the correct volume will have “chunky” rather than “slabby” proportions.

Having measured the density of the core, the target volume can be translated into a target mass and with the average mass per unit length also known, an average cutting interval can be determined for the core.

Sufficient pieces of the quartered core are cut to generate 100 particles. These are then divided into the five sets of 20 and broken in the drop-weight tester at the five different energy levels. Within each set, the three possible orientations of the particles are equally represented (as far as possible, given that there are 20 particles). The orientations prescribed for testing are shown in Figure 3.

Figure 3 – Orientations of Pieces for Breakage (Cut Core Method)

The cut core method cannot be used for cores with diameters exceeding 70 mm, where the particle masses would be too large to achieve the highest prescribed energy level.

2.4 SMC TEST® RESULTS

The SMC Test® results for the eleven samples from LI1 Project are given in Table 1. This table includes the average rock density and the drop-weight index that is the direct result of the test procedure, plus the derived estimates of parameters A, b and t_a that are required for JKSimMet comminution modelling. The values determined for the M_{ia}, M_{ih} and M_{ic} parameters developed by SMCT are also presented in this table. The M_{ia} parameter represents the coarse particle component (down to 750 µm), of the overall comminution energy and can be used together with the M_{ib} (fine particle component) to estimate the total energy requirements of a conventional comminution circuit. The use of these parameters is explained further in APPENDIX B.

In the case of the eleven samples from LI1 Project, the A and b estimates are based on a correlation using the database of all results so far accumulated by SMCT.

Table 1 - SMC Test® Results

Sample Designation	DWi	DWi	Mia	Mih	Mic	A	b	SG	t_a
	kWh/m^3	%	kWh/t	kWh/t	kWh/t				
52641-1	3.23	19	11.6	7.4	3.8	59.8	1.31	2.53	0.80
52461-2	3.77	24	12.3	8.1	4.2	65.2	1.10	2.69	0.69
52461-3	3.79	25	12.3	8.1	4.2	64.6	1.11	2.71	0.68
52461-4	3.38	20	11.0	7.1	3.7	64.3	1.27	2.76	0.77
52461-5	4.14	29	12.9	8.7	4.5	66.8	1.00	2.76	0.63
52461-6	4.03	27	13.0	8.7	4.5	63.0	1.06	2.69	0.64
52461-7	3.22	19	10.7	6.8	3.5	60.5	1.40	2.74	0.80
52461-8	3.84	25	12.3	8.2	4.2	62.7	1.13	2.73	0.67
52461-9	3.54	22	11.6	7.5	3.9	63.5	1.21	2.72	0.73
52461-10	3.39	20	11.2	7.2	3.7	63.4	1.27	2.71	0.76
52461-11	3.54	22	11.5	7.5	3.9	63.0	1.23	2.74	0.73

Note: For more details on how the M_{ia}, M_{ih} and M_{ic} parameters are derived and used, see APPENDIX B or go to the JKTech website at http://www.jktech.com.au/Products_Services/Laboratory-Services/Comminution-Testing/SMC-Test/index.htm and click on the link to download Steve Morrell's paper on this subject.

The influence of particle size on the specific comminution energy needed to achieve a particular t_{10} value can also be inferred from the SMC Test® results. The energy requirements for three particle sizes, each crushed to three different t_{10} values, are presented in Table 2.

Table 2 – Energy Requirements Related to Particle Size

Sample Designation	Particle Size (mm)								
	14.5			28.9			57.8		
	t_{10} Values for Given Specific Energies (%)								
	10 kWh/t	20 kWh/t	30 kWh/t	10 kWh/t	20 kWh/t	30 kWh/t	10 kWh/t	20 kWh/t	30 kWh/t
52641-1	0.18	0.40	0.66	0.14	0.29	0.47	0.10	0.21	0.34
52461-2	0.20	0.44	0.72	0.15	0.32	0.51	0.11	0.24	0.37
52461-3	0.20	0.44	0.72	0.15	0.32	0.51	0.11	0.24	0.37
52461-4	0.18	0.38	0.63	0.13	0.28	0.45	0.10	0.21	0.32
52461-5	0.22	0.47	0.78	0.16	0.34	0.55	0.12	0.25	0.40
52461-6	0.22	0.47	0.77	0.16	0.34	0.55	0.12	0.25	0.40
52461-7	0.17	0.37	0.61	0.13	0.27	0.43	0.09	0.20	0.31
52461-8	0.20	0.44	0.73	0.15	0.32	0.52	0.11	0.24	0.37
52461-9	0.19	0.41	0.67	0.14	0.30	0.48	0.10	0.22	0.34
52461-10	0.18	0.39	0.65	0.13	0.29	0.46	0.10	0.21	0.33
52461-11	0.19	0.41	0.67	0.14	0.30	0.47	0.10	0.22	0.34

The SMC Test® database now contains over 11,000 test results on samples representing more than 600 different deposits worldwide.

Around 99% of the DWi values lie in the range 0.5 to14.0 kWh/m^3, with soft ores being at the low end of this range and hard ores at the high end.

A cumulative graph of DWi values from the SMC Test® Database is shown in Figure 4 below. This graph can be used to compare the DWi of the material from LI1 Project, with the entire population of ores in the SMCT database. The figures on the y-axis of the graph represent the percentages of all ores tested that are softer than the x-axis (DWi) value selected.

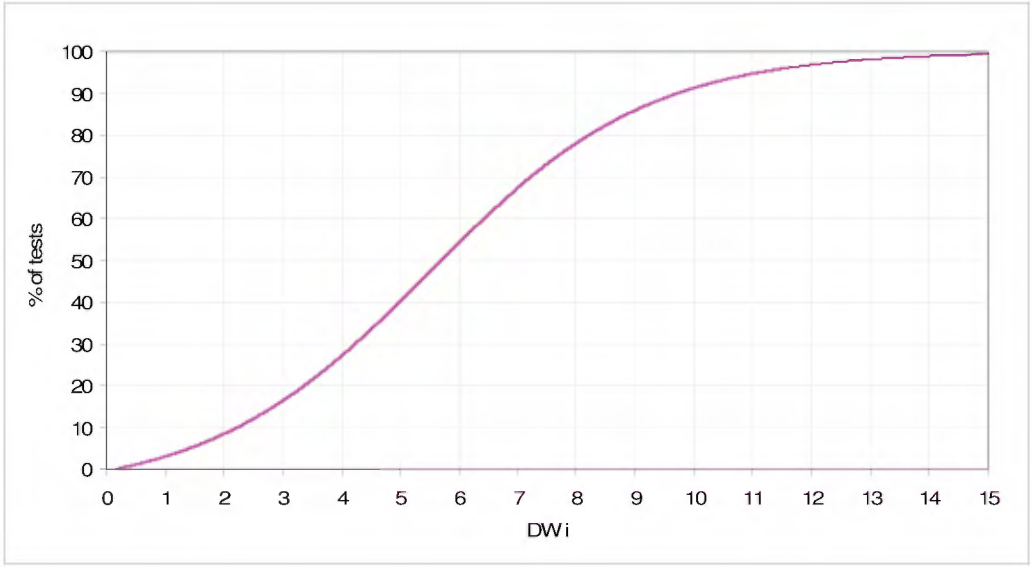

Figure 4 – Cumulative Distribution of DWi Values in SMC Test® Database

A further cumulative distribution graph is provided in Figure 5 to allow a comparison of the M_{ia}, M_{ih} and M_{ic} values obtained for the LI1 Project material, with the entire population of values for these parameters contained in the SMCT database.

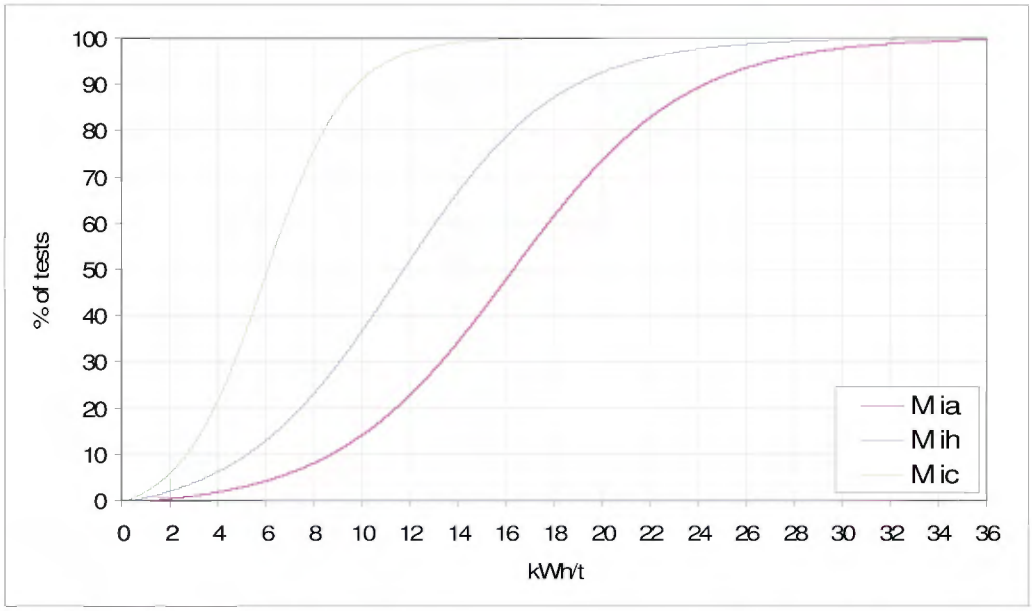

Figure 5 - Cumulative Distribution of M_{ia}, M_{ih} and M_{ic} Values in the SMCT Database

The value of A*b, which is also a measure of resistance to impact breakage, is calculated and presented in Table 3 along with indicators of how each A*b value compares with the accumulated values in the JKTech DW database (from full drop-weight testing). These indicators are the Category (eg "soft" etc), the Rank (how many out of 3,065 recordings in database are harder) and the percent of database values that are harder. Note that in contrast to the DWi, a high value of A*b means that an ore is soft whilst a low value means that it is hard.

Table 3 – Derived Values for A*b and t10 at 1 kWh/t

Sample Designation	A*b				t10 @ 1 kWh/t			
	Value	Category	Rank	%	Value	Category	Rank	%
52641-1	78.3	soft	2399	78.2%	43.7	soft	2436	79.5%
52461-2	71.7	soft	2304	75.1%	43.5	soft	2427	79.2%
52461-3	71.7	soft	2304	75.1%	43.3	soft	2423	79.0%
52461-4	81.7	soft	2444	79.7%	46.2	soft	2544	83.0%
52461-5	66.8	soft	2202	71.8%	42.2	soft	2365	77.1%
52461-6	66.8	soft	2200	71.8%	41.2	soft	2306	75.2%
52461-7	84.7	soft	2474	80.7%	45.6	soft	2517	82.1%
52461-8	70.9	soft	2282	74.4%	42.4	soft	2378	77.6%
52461-9	76.8	soft	2381	77.7%	44.6	soft	2476	80.8%
52461-10	80.5	soft	2430	79.3%	45.6	soft	2518	82.1%
52461-11	77.5	soft	2386	77.8%	44.6	soft	2476	80.8%

The calculated value of t_{10} at an E_{cs} of 1 kWh/t is also shown in Table 3. This is again accompanied by Category, Rank and the % of values in the database that are

harder, so each can be seen against the yard-stick of all other samples in the JKTech database.

The derived A*b values range from 66.8 to 84.7 giving an average of 75.2, while the t_{10} at 1 kWh/t values ranged from 41.2 to 46.2 giving an average of 43.9.

In Figure 6 and Figure 7 below, histogram style frequency distributions for the A*b values and for the t_{10} at 1 kWh/t values in the JKTech DW database are shown respectively.

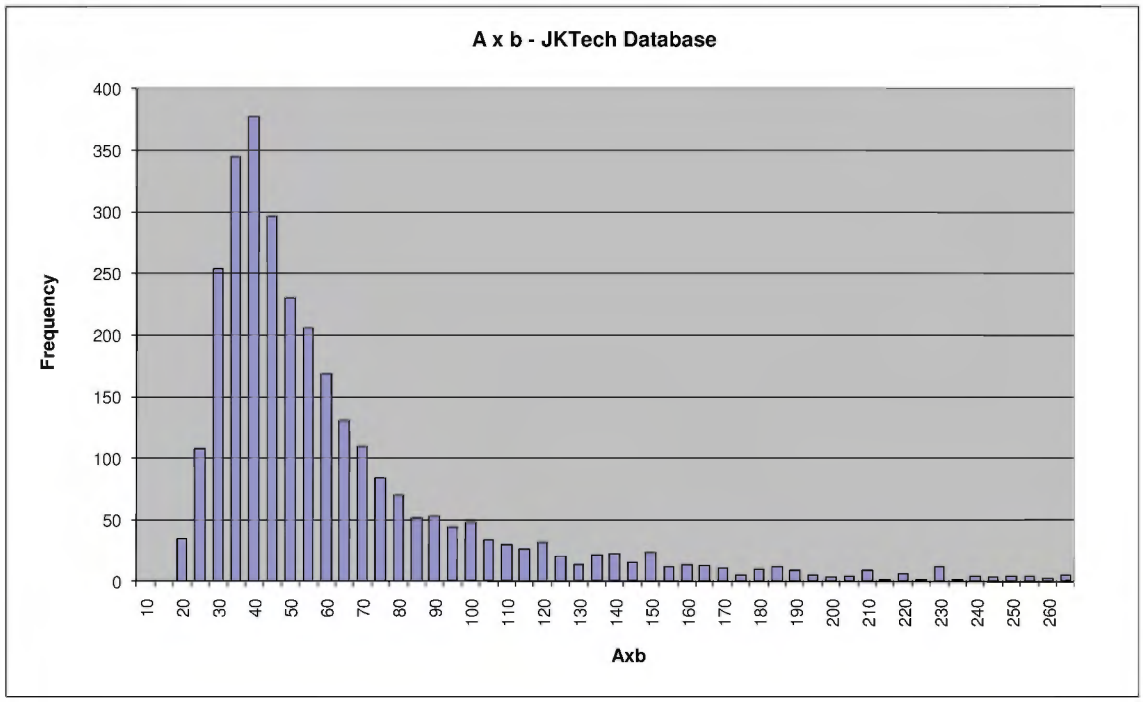

Figure 6 - Frequency Distribution of A*b in the JKTech Database

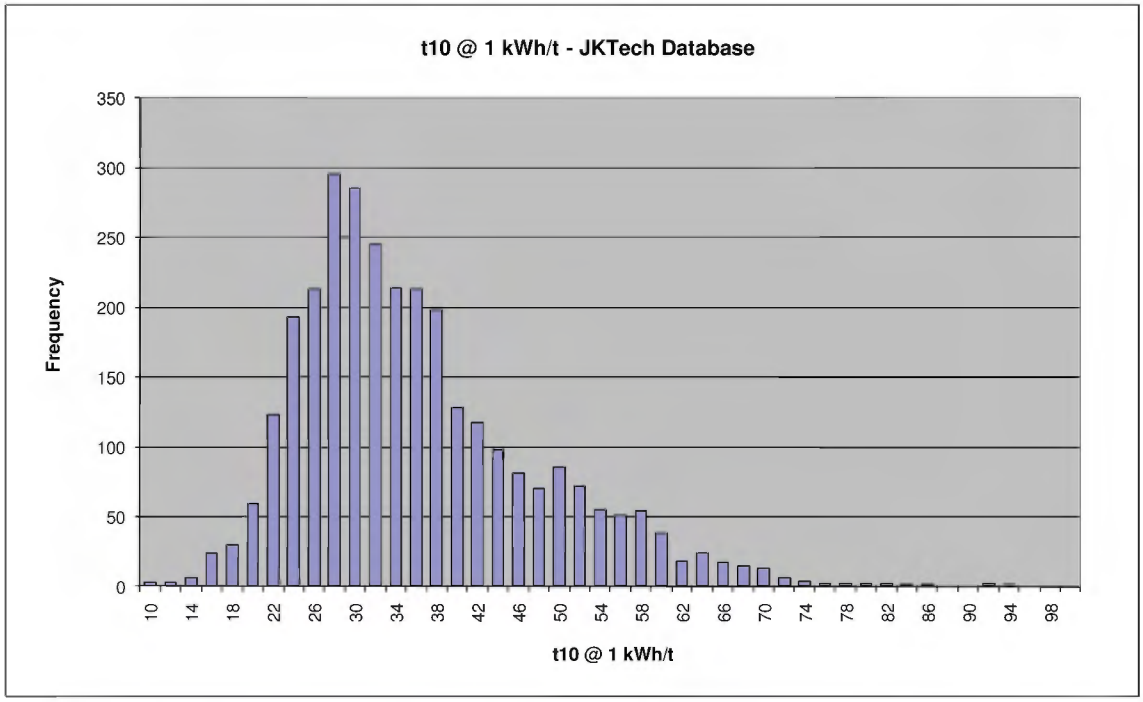

Figure 7 - Frequency Distribution of t_{10}@1kWh/t in the JKTech Database

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APPENDICES

APPENDIX A. BACKGROUND TO THE SMC TEST®

A 1 HOW THE SMC TEST® RESULTS ARE USED

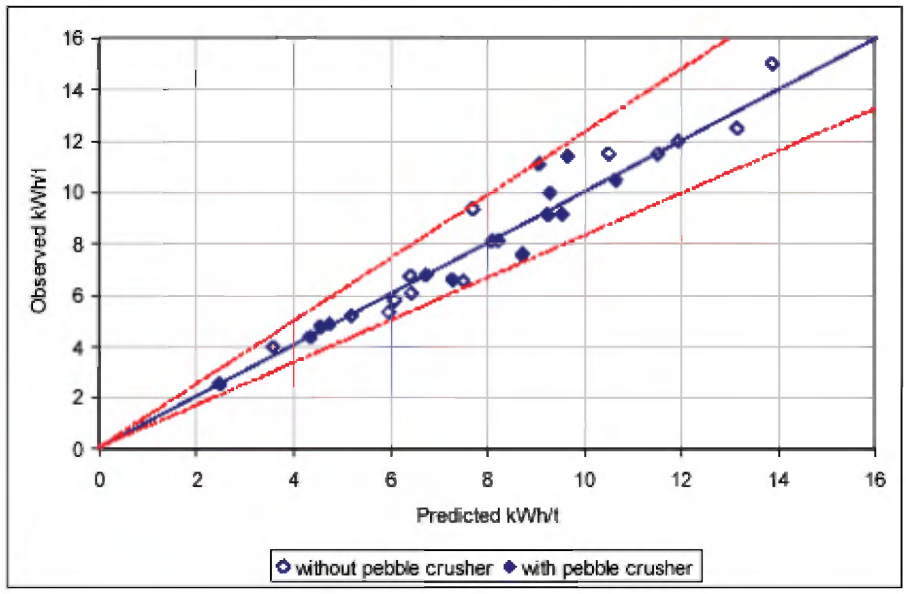
The SMC Test® generates a relationship between specific input energy (kWh/t) and the percent of broken product passing a specified sieve size. The results are used to determine the drop-weight index (DWi), which is a measure of the strength of the rock when broken under impact conditions. The DWi is directly related to the JK rock breakage parameters A and b and hence can be used to estimate the values of these parameters.

Provision of a relatively low cost method of estimating the A and b parameters opens the possibility of incorporating these data into mine and mill planning operations. However a number of full drop-weight tests is still recommended for any particular orebody, to ensure that an accurate correlation between the DWi and the A and b parameters is available. The number of full drop-weight tests required for a given orebody will depend on its variability and should at least cover the major recognised ore types.

The A and b parameters are used in AG/SAG mill models, such as those in JKSimMet, for predicting how the rock will break inside the mill. From this description the models can predict what the throughput, power draw and product size distribution will be (Napier-Munn et al (1996)). Modelling also enables a detailed flowsheet to be built up of the comminution circuit response to changes in ore type. It also allows optimisation strategies to be developed to overcome any deleterious changes in circuit performance predicted from differences in ore type when such changes are indicated by the SMC Test®. These strategies can include both changes to how mills are operated (eg ball load, speed etc) and changes to feed size distribution through modification of blasting practices and primary crusher operation (mine-to-mill).

The mine to mill models require information on rock mass competence such as provided by the point load index. The DWi is correlated with the point load index and hence can also be used in blast fragmentation modelling where direct measurements of point load index are not available.

The DWi is related to the resistance of a rock to breakage under impact. SMCT has developed a series of equations that relate the DWi to the specific energy (kWh/t) requirements of complete AG and SAG mill circuits. These equations take into consideration factors such as ball charge, feed size, aspect ratio, whether the mill is operated with or without a pebble crusher and whether it is closed with a fine classifier such as a cyclone. The ability of these equations to predict AG/SAG mill circuit specific energy is illustrated in App Figure 1. The data shown cover 19 different operations and include Cu, Au, Ni and Pb/Zn ores.



App Figure 1 - Mill Power Prediction Based on DWi

It should be noted that the parameter ta, which is the parameter representing the low energy abrasion component of breakage, is not yielded by the SMC Test®. This parameter is derived from a tumbling test that is carried out as part of the full drop-weight test. The fact that it is also required as an input to the JKSimMet SAG/AG models provides a further reason for ensuring that some full drop-weight tests are also performed to represent at least the main rock types of an orebody.

A 2 IMPACT COMMINUTION THEORY

When a rock fragment is broken, the degree of breakage can be characterised by the "t10" parameter. The t10 value is the percentage of the original rock mass that passes a screen aperture one tenth of the original rock fragment size. This parameter allows the degree of breakage to be compared across different starting sizes.

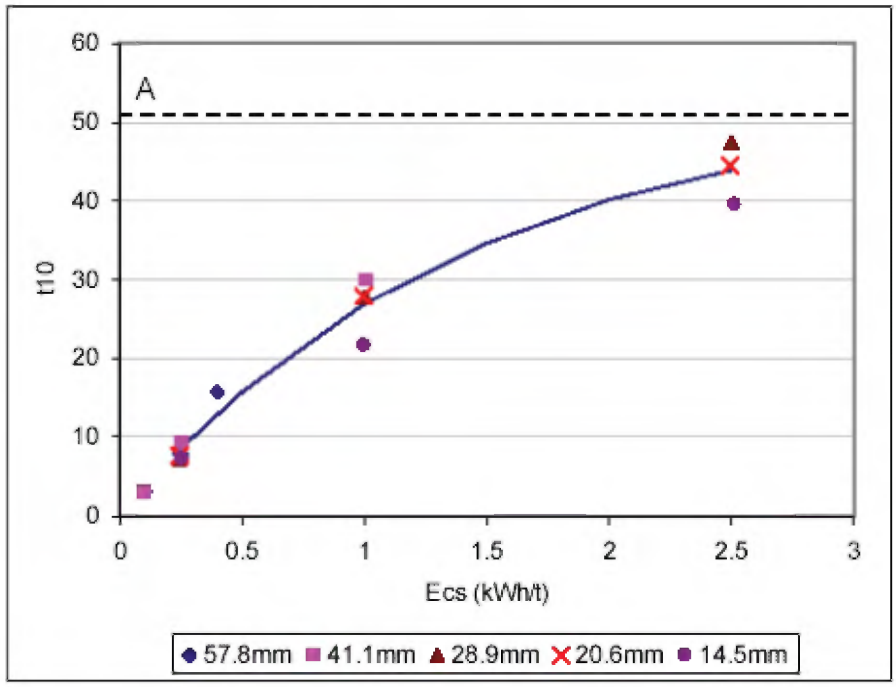
The specific comminution energy (Ecs) has the units kWh/t and is the energy applied during impact breakage. As the impact energy is varied, so does the t10 value vary in response. Higher impact energies produce higher values of t10, which of course means products with finer size distributions.

The equation describing the relationship between the t10 and Ecs is given below.

$$t_{10} = A (1 - e^{-b.Ecs})$$

As can be seen from this equation, there are two rock breakage parameters A and b that relate the t10 (size distribution index) to the applied specific energy (Ecs). These parameters are ore specific and are normally determined from a full drop-weight test.

A typical plot of t10 vs Ecs from a drop-weight test is shown in App Figure 2. The relationship is characterised by the two-parameter equation above, where t10 is the dependent variable.

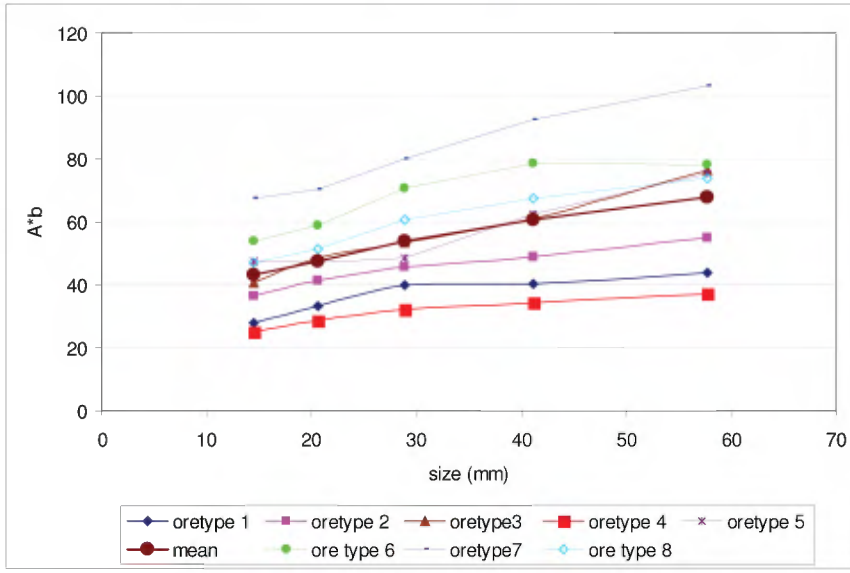


App Figure 2 - Typical t_{10} v Ecs Plot

The t10 can be thought of as a "fineness index" with larger values of t10 indicating a finer product size distribution. The value of parameter A is the limiting value of t10. This limit indicates that at higher energies, little additional size reduction occurs as the Ecs is increased beyond a certain value. A*b is the slope of the curve at 'zero' input energy and is generally regarded as an indication of the strength of the rock, lower values indicating a higher strength.

The A and b parameters can also be used with equation 1 to generate a table of Ecs values, given a range of t10 values. Such a table is used in crusher modelling to predict the power requirement of the crusher given a feed and a product size specification.

The DWi can be used to estimate the JK rock breakage parameters A and b by utilizing the fact that there is usually a pronounced (and ore specific) trend to decreasing rock strength with increasing particle size. This trend is illustrated in App Figure 3 which shows a plot of A*b versus particle size for a number of different rock types.

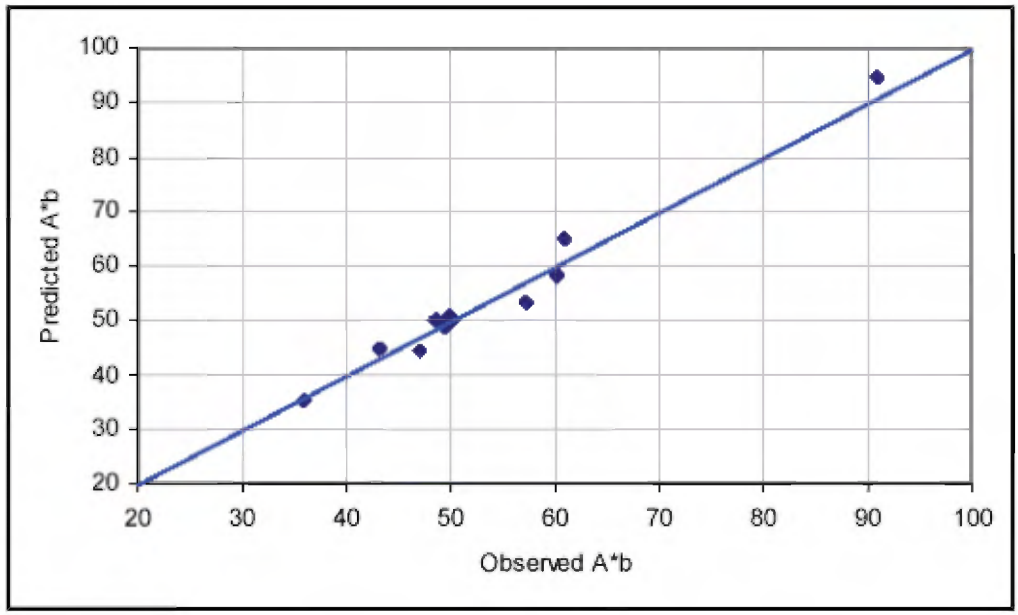


App Figure 3 - Size Dependence of A*b for a Range of Ore Types

In the case of a conventional drop-weight test these values are effectively averaged and a mean value of A and b is reported. The SMC Test® uses a single size and makes use of relationships such as that shown in App Figure 3 to predict the A and b of the particle size that has the same value as the mean for a full drop-weight test.

An example of this is illustrated in App Figure 4 - Predicted v Observed A*b

where the observed values of the product A*b are plotted against those predicted using the DWi. Each of the data points in App Figure 4 is a result from a different ore type within an orebody.



App Figure 4 - Predicted v Observed A*b

APPENDIX B. USE OF THE Mia, Mib, Mih, Mic IN PREDICTING COMMINUTION CIRCUIT SPECIFIC ENERGY

B 1 INTRODUCTION

The following technical note describes the recently extended use of the SMC Test® to include crushers and HPGRs in determining the overall specific energy demand of comminution circuits. It builds on previous work which included tumbling mills only and should be read in conjunction with an earlier technical note dated September 2007 entitled "*Use of the SMC Test® in Predicting Total Comminution Circuit Specific Energy"* as well as published papers: Morrell, 2008a, 2008b. This enhancement now enables the SMC Test® to be used in conjunction with the Bond ball work index test to predict the specific energy of comminution circuits where such circuits include combinations of any of the following equipment:

- AG and SAG mills
- Ball mills
- Rod mills
- Crushers
- High pressure Grinding Rolls (HPGR)

B 2 EQUATIONS

B 2.1 General

The approach divides comminution equipment into three categories:

- Tumbling mills, eg AG, SAG, rod and ball mills
- Conventional reciprocating crushers, eg jaw, gyratory and cone
- HPGRs

Tumbling mills are described using 2 indices: M_{ia} and M_{ib}

Crushers have one index: M_{ic}

HPGRs have one index: M_{ih}

For tumbling mills the 2 indices relate to "coarse" and "fine" ore properties plus an efficiency factor which represents the influence of a pebble crusher in AG/SAG mill circuits. "Coarse" in this case is defined as spanning the size range from a P_{80} of 750 µm up to the P_{80} of the product of the last stage of crushing or HPGR size reduction prior to grinding. "Fine" covers the size range from a P_{80} of 750 µm down to P_{80} sizes typically reached by conventional ball milling, ie about 45 µm. The choice of 750 µm as the division between "coarse" and "fine" particle sizes was determined during the

development of the technique and was found to give the best overall results across the range of plants in SMCT's database. Implicit in the approach is that distributions are parallel and linear in log-log space.

The work index covering grinding in tumbling mills of coarse sizes is labelled M_{ia}. The work index covering grinding of fine particles is labelled M_{ib}. M_{ia} values are provided as a standard output from a SMC Test® (Morrell, 2004[a]) whilst M_{ib} values can be determined using the data generated by a conventional Bond ball mill work index test (M_{ib} is NOT the Bond ball work index). M_{ic} and M_{ih} values are also provided as a standard output from a SMC Test®.

The general size reduction equation is as follows (Morrell, 2004[b]):

$$W_i = M_i 4\left(x_2^{\,f(x_2)} - x_1^{\,f(x_1)}\right) \qquad (1)$$

Where:

M_i = Work index related to the breakage property of an ore (kWh/tonne). For grinding from the product of the final stage of crushing to a P_{80} of 750 µm (coarse particles) the index is labelled M_{ia} and for size reduction from 750 µm to the final product P_{80} normally reached by conventional ball mills (fine particles) it is labelled M_{ib}. For conventional crushing M_{ic} is used and for HPGRs M_{ih} is used.

W_i = Specific comminution (kWh/tonne)

x_2 = 80% passing size for the product (µm)

x_1 = 80% passing size for the feed (µm)

$f(x_j)$ = $-(0.295 + x_j/1000000)$ (Morrell, 2006) (2)

For tumbling mills the specific comminution energy (Wi) relates to the power at the pinion or for gearless drives - the motor output. For HPGRs it is the energy inputted to the rolls, whilst for conventional crushers Wi relates to the specific energy as determined using the motor input power less the no-load power.

B 2.2 Specific Energy Determination for Comminution Circuits

The total specific energy (W_T) to reduce in size primary crusher product to final product is given by:

$$W_T = W_a + W_b + W_c + W_h + W_s \qquad (3)$$

Where:

W_a = specific energy to grind coarser particles in tumbling mills

W_b = specific energy to grind finer particles in tumbling mills

W_c = specific energy for conventional crushing

W_h = specific energy for HPGRs

W_s = specific energy correction for size distribution

Clearly only the *W* values associated with the relevant equipment in the circuit being studied are included in equation 3.

B 2.2.1 Tumbling mills

For coarse particle grinding in tumbling mills equation 1 is written as:

$$W_a = K_1 M_{ia} 4\left(x_2^{\,f(x_2)} - x_1^{\,f(x_1)}\right) \quad (4)$$

Where:

K_1 = 1.0 for all circuits that do not contain a recycle pebble crusher and 0.95 where circuits do have a pebble crusher

x_1 = P_{80} (µm) of the product of the last stage of crushing before grinding

x_2 = 750 µm

M_{ia} = Coarse ore work index and is provided directly by SMC Test®

For fine particle grinding equation 1 is written as:

$$W_b = M_{ib} 4\left(x_3^{\,f(x_3)} - x_2^{\,f(x_2)}\right) \quad (5)$$

Where:

x_2 = 750 µm

x_3 = P_{80} (µm) of final grind

M_{ib} = Provided by data from the standard Bond ball work index test using the following equation (Morrell, 2006):

$$M_{ib} = \frac{18.18}{P_1^{0.295}(Gbp)\left(p_{80}^{\,f(p_{80})} - f_{80}^{\,f(f_{80})}\right)} \quad (6)$$

Where:

M_{ib} = fine ore work index (kWh/tonne)

P_1 = closing screen size (µm)

Gbp = net grams of screen undersize per mill revolution

p_{80} = 80% passing size of the product (µm)

f_{80} = 80% passing size of the feed (µm)

Note that the Bond ball work index test should be carried out with a closing screen size chosen to give a final product P_{80} similar to that intended for the full-scale circuit.

B 2.2.2 Conventional Crushers

Equation 1 for conventional crushers is written as:

$$W_c = K_2 M_{ic} 4\left(x_2^{f(x_2)} - x_1^{f(x_1)}\right) \quad (7)$$

K_2 = 1.0 for all crushers operating in closed circuit with a classifying screen. If the crusher is in open circuit, eg pebble crusher in a AG/SAG circuit, K2 takes the value of 1.19.

x_1 = P_{80} (µm) of the circuit feed

x_2 = P_{80} (µm) of the circuit product

M_{ic} = Crushing ore work index and is provided directly by SMC Test®

B 2.2.3 HPGR

Equation 1 for conventional crushers is written as:

$$W_h = K_3 M_{ih} 4\left(x_2^{f(x_2)} - x_1^{f(x_1)}\right) \quad (8)$$

K_3 = 1.0 for all HPGRs operating in closed circuit with a classifying screen. If the HPGR is in open circuit, K3 takes the value of 1.19.

x_1 = P_{80} (µm) of the circuit feed

x_2 = P_{80} (µm) of the circuit product

M_{ih} = HPGR ore work index and is provided directly by SMC Test®

B 2.2.4 Specific Energy Correction for Size Distribution (Ws)

Implicit in the approach described in this paper is that the feed and product size distributions are parallel and linear in log-log space. Where they are not, allowances (corrections) need to be made. By and large, such corrections are most likely to be necessary (or are large enough to be warranted) when evaluating circuits in which closed circuit secondary/tertiary crushing is followed by ball milling. This is because such crushing circuits tend to produce a product size distribution which is relatively steep when compared to the ball mill circuit cyclone overflow. This is illustrated in App Figure 5, which shows measured distributions from an open and closed crusher circuit as well as a ball mill cyclone overflow. The closed circuit crusher distribution can be seen to be relatively steep compared with the open circuit crusher distribution and ball mill cyclone overflow. Also the open circuit distribution more closely follows the gradient of the cyclone overflow. If a ball mill circuit were to be fed 2 distributions, each with same P_{80} but with the open and closed circuit gradients in App Figure 5, the closed circuit distribution would require more energy to grind to the final P_{80}. How much more energy is required is difficult to determine. However, for the purposes of this approach it has been assumed that the additional specific energy for ball milling is the same as the difference in specific energy between open and closed crushing to reach the nominated ball mill feed size. This assumes that a

crusher would provide this energy. However, in this situation the ball mill has to supply this energy and it has a different (higher) work index than the crusher (ie the ball mill is less energy efficient than a crusher and has to input more energy to do the same amount of size reduction). Hence from equation 7, to crush to the ball mill circuit feed size (x_2) in open circuit requires specific energy equivalent to:

$$W_c = 1.19 * M_{ic} 4\left(x_2{}^{f(x_2)} - x_1{}^{f(x_1)}\right) \quad (9)$$

For closed circuit crushing the specific energy is:

$$W_c = 1 * M_{ic} 4\left(x_2{}^{f(x_2)} - x_1{}^{f(x_1)}\right) \quad (10)$$

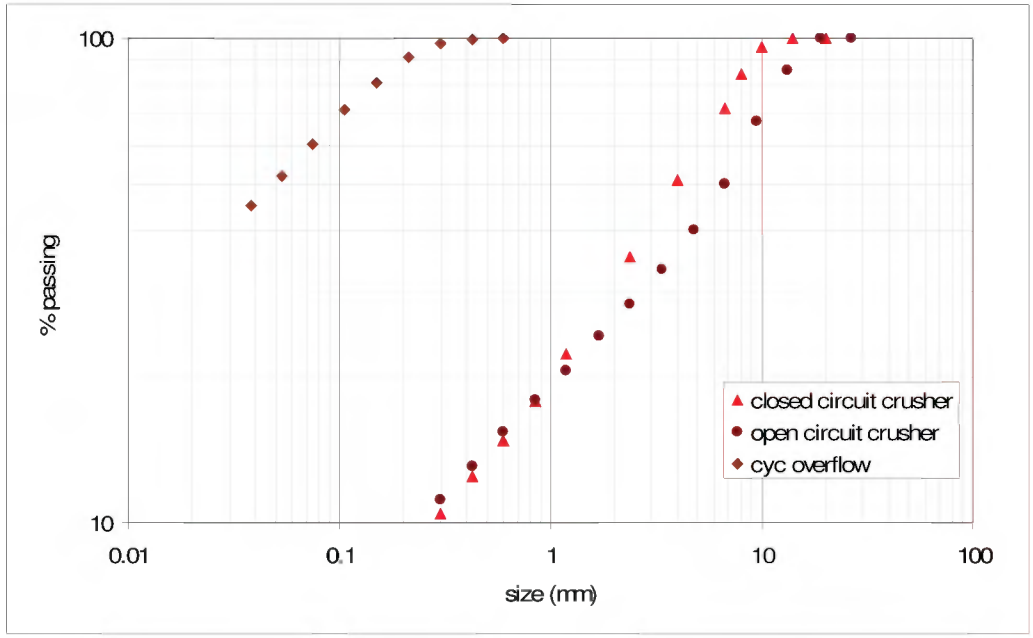
The difference between the two (eq 9 – eq 10) has to be provided by the milling circuit with an allowance for the fact that the ball mill, with its lower energy efficiency, has to provide it and not the crusher. This is what is referred to in equation 3 as W_s and for the above example is therefore represented by:

$$W_s = 0.19 * M_{ia} 4\left(x_2{}^{f(x_2)} - x_1{}^{f(x_1)}\right) \quad (11)$$

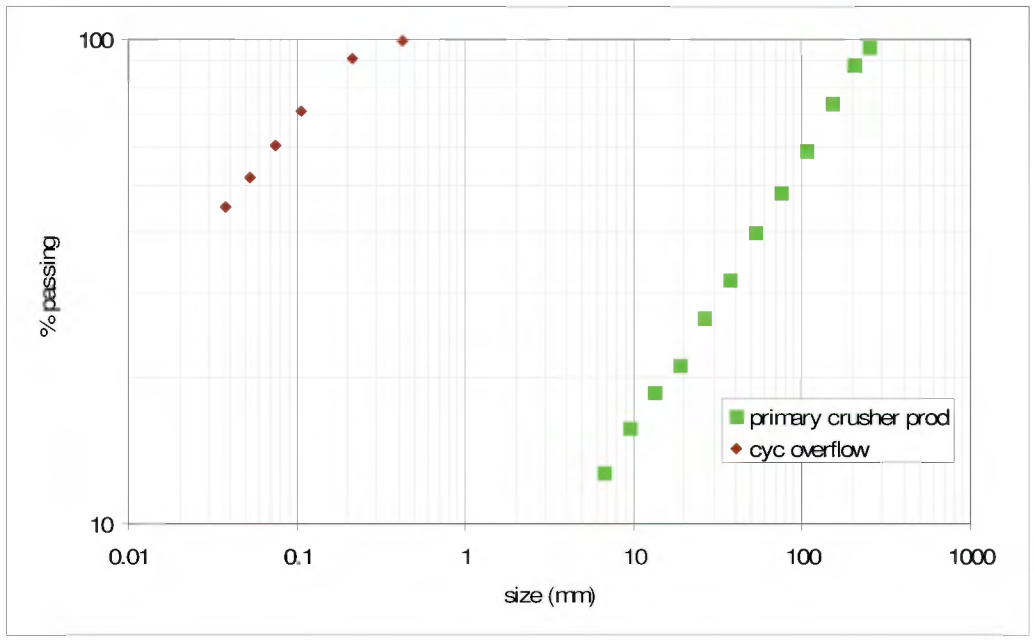
Note that in equation 11 M_{ic} has been replaced with M_{ia}, the coarse particle tumbling mill grinding work index.

In AG/SAG based circuits the need for W_s appears to be unnecessary as App Figure 6 illustrates. Primary crusher feeds often have the shape shown in App Figure 6 and this has a very similar gradient to typical ball mill cyclone overflows. A similar situation appears to apply with HPGR product size distributions, as illustrated in App Figure 7. Interestingly SMCT's data show that for HPGRs, closed circuit operation appears to require a lower specific energy to reach the same P_{80} as in open circuit, even though the distributions for open and closed circuit look to have almost identical gradients. Closer examination of the distributions in fact shows that in closed circuit the final product tends to have slightly less very fine material, which may account for the different energy requirements between the two modes of operation. It is also possible that recycled material in closed circuit is inherently weaker than new feed, as it has already passed through the HPGR previously and may have sustained micro-cracking. A reduction in the Bond ball mill work index as measured by testing HPGR products compared it to the Bond ball mill work index of HPGR feed has been noticed in many cases in the laboratory (see next section) and hence there is no reason to expect the same phenomenon would not affect the recycled HPGR screen oversize.

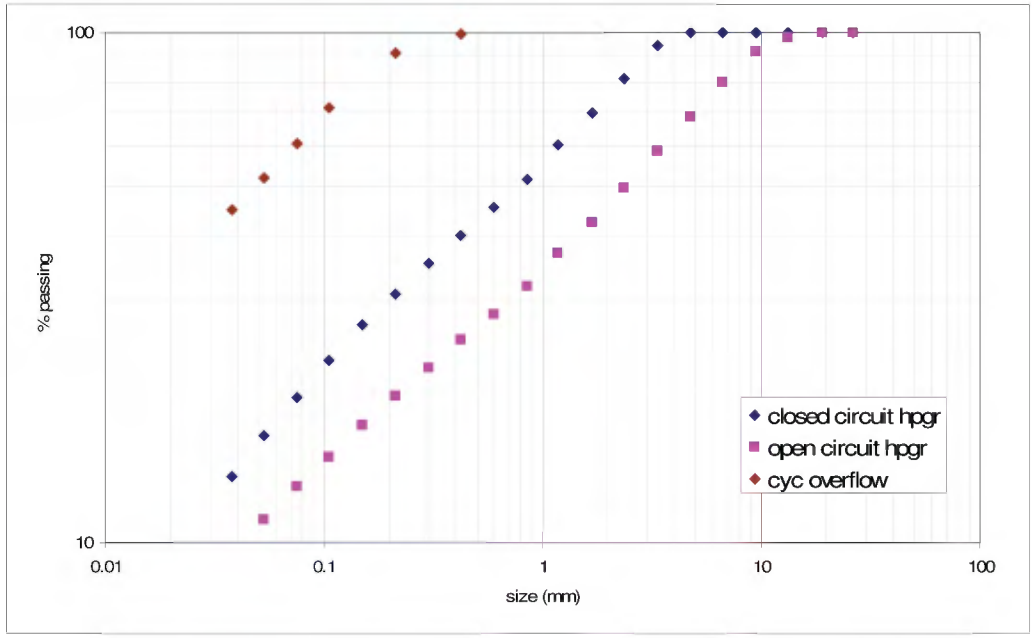
It follows from the above arguments that in HPGR circuits, which are typically fed with material from closed circuit secondary crushers, a similar feed size distribution correction should also be applied. However, as the secondary crushing circuit uses such a relatively small amount of energy compared to the rest of the circuit (as it crushes to a relatively coarse size) the magnitude of size distribution correction is very small indeed – much smaller than the error associated with the technique - and hence may be omitted in calculations.



App Figure 5 – Examples of Open and Closed Circuit Crushing Distributions Compared with a Typical Ball Mill Cyclone Overflow Distribution



App Figure 6 – Example of a Typical Primary Crusher (Open and Circuit) Product Distribution Compared with a Typical Ball Mill Cyclone Overflow Distribution



App Figure 7 – Examples of Open and Closed Circuit HPGR Distributions Compared with a Typical Ball Mill Cyclone Overflow Distribution

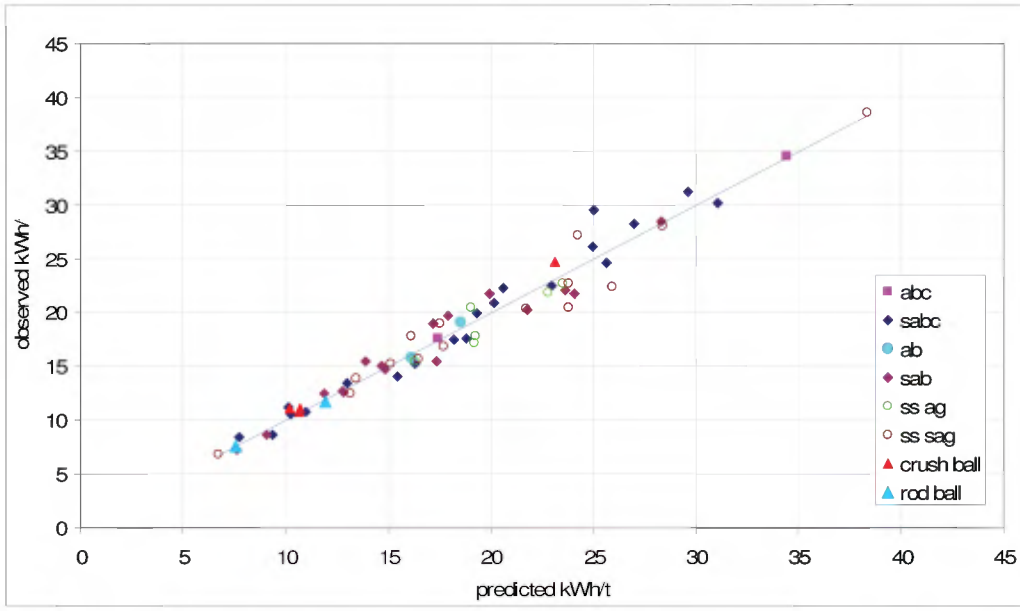
B 2.2.5 Weakening of HPGR Products

As mentioned in the previous section, laboratory experiments have been reported by various researchers in which the Bond ball work index of HPGR products is less than that of the feed. The amount of this reduction appears to vary with both material type and the pressing force used. Observed reductions in the Bond ball work index have typically been in the range 0-10%. In the approach described in this paper no allowance has been made for such weakening. However, if HPGR products are available which can be used to conduct Bond ball work index tests on then M_{ib} values obtained from such tests can be used in equation 5. Alternatively the M_{ib} values from Bond ball mill work index tests on HPGR feed material can be reduced by an amount that the user thinks is appropriate. Until more data become available from full scale HPGR/ball mill circuits it is suggested that, in the absence of Bond ball mill work index data on HPGR products, the M_{ib} results from HPGR feed material are reduced by no more than 5% to allow for the effects of micro-cracking.

B 3 VALIDATION

B 3.1 Tumbling Mill Circuits

The approach described in the previous section was applied to 65 industrial data sets. The results are shown in App Figure 8. In all cases, the specific energy relates to the tumbling mills contributing to size reduction from the product of the final stage of crushing to the final grind. Data are presented in terms of equivalent specific energy at the pinion. In determining what these values were on each of the plants in the data base it was assumed that power at the pinion was 93.5% of the measured gross (motor input) power, this figure being typical of what is normally accepted as being reasonable to represent losses across the motor and gearbox. For gearless drives (so-called wrap-around motors) a figure of 97% was used.



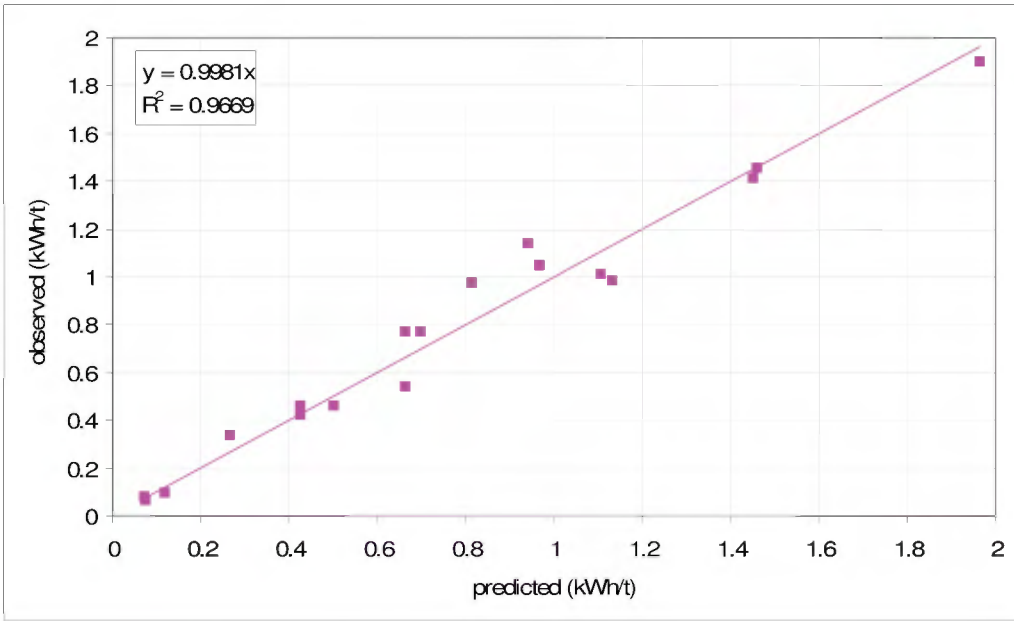
App Figure 8 – Observed vs Predicted Tumbling Mill Specific Energy

B 3.2 Conventional Crushers

Validation of equation 1 used 10 different crushing circuits (18 data sets), including secondary, tertiary and pebble crushers in AG/SAG circuits. Observed vs predicted specific energies are given in App Figure 9. The observed specific energies were calculated from the crusher throughput and the net power draw of the crusher as defined by:

Net Power = Motor Input Power – No Load Power (12)

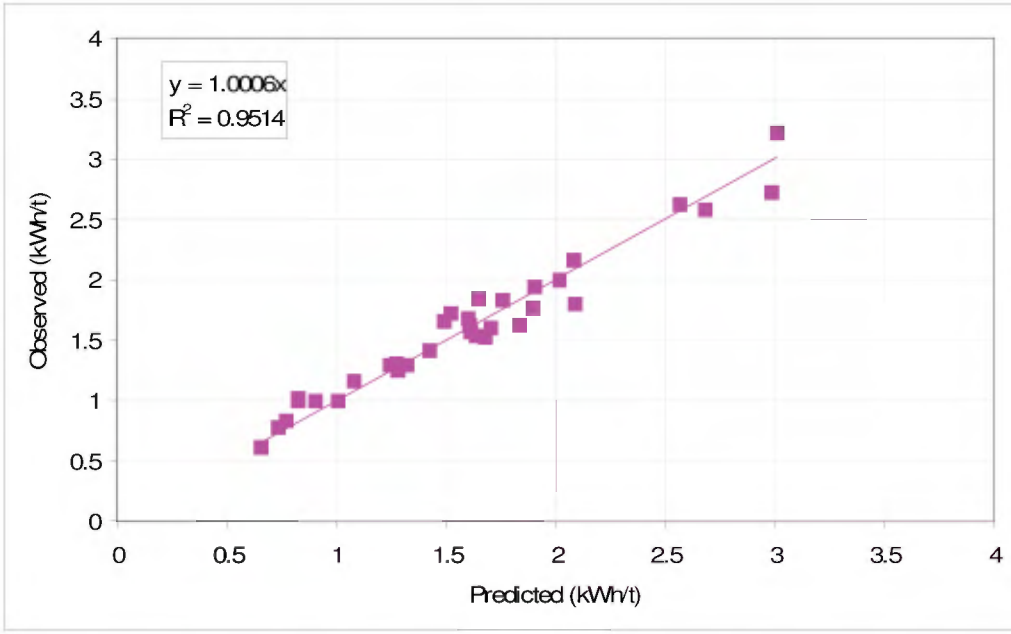
No-load power tends to be relatively high in conventional crushers and hence net power is significantly lower than the motor input power. From examination of the 18 crusher data sets the motor input power was found to be on average 35% higher than the net power.



App Figure 9 – Observed vs Predicted Conventional Crusher Specific Energy

B 3.3 HPGRs

Validation of equation 1 for HPGRs used data from 18 different circuits (35 data sets) including laboratory, pilot and industrial scale equipment. Observed vs predicted specific energies are given in App Figure 10. The data relate to HPGRs operating with specific grinding forces typically in the range 2.5-3.5 N/mm^2. The observed specific energies relate to power delivered by the roll drive shafts. Motor input power for full scale machines is expected to be 8-10% higher.



App Figure 10 – Observed vs Predicted HPGR Specific Energy

B 4 WORKED EXAMPLES

A SMC Test® and Bond ball work index test were carried out on a representative ore sample. The following results were obtained:

SMC Test:

M_{ia} = 19.4 kWh/t

M_{ic} = 7.2 kWh/t

M_{ih} = 13.9 kWh/t

Bond test carried out with a 150 micron closing screen:

M_{ib} = 18.8 kWh/t

Three circuits are to be evaluated:

- SABC
- HPGR/ball mill
- Conventional crushing/ball mill

The overall specific grinding energy to reduce a primary crusher product with a P_{80} of 100 mm to a final product P_{80} of 106 µm needs to be estimated.

B 4.1 SABC Circuit

Coarse particle tumbling mill specific energy

Combining eq 2 and 4:

$$W_a = 0.95 * 19.4 * 4 * \left(750^{\ -(0.295+750/1000000)} - 100000^{\ -(0.295+100000/1000000)}\right)$$

= 9.6 kWh/t

Fine particle tumbling mill specific energy

Combining eq 2 and 5:

$$W_b = 18.8 * 4 * \left(106^{\ -(0.295+106/1000000)} - 750^{\ -(0.295+750/1000000)}\right)$$

= 8.4 kWh/t

Pebble crusher specific energy

In this circuit, it is assumed that the pebble crusher feed P_{80} is 52.5mm. As a rule of thumb this value can be estimated by assuming that it is 0.75 of the nominal pebble port aperture (in this case the pebble port aperture is 70mm). The pebble crusher is set to give a product P_{80} of 12mm. The pebble crusher feed rate is expected to be 25% of new feed tph.

Combining eq 2 and 7:

$$W_c = 1.19 * 7.2 * 4 * \left(12000^{-(0.295+12000/1000000)} - 52500^{-(0.295+52500/1000000)}\right)$$

= 1.12 kWh/t when expressed in terms of the crusher feed rate

= 1.12 * 0.25 kWh/t when expressed in terms of the SABC circuit new feed rate

= 0.3 kWh/t of SAG mill circuit new feed

Total net comminution specific energy:

From eq 3:

W_T = 9.6 + 8.4 + 0.3 kWh/t

= 18.3 kWh/t

B 4.2 HPGR/Ball Milling Circuit

In this circuit primary crusher product is reduced to a HPGR circuit feed P_{80} of 35 mm by closed circuit secondary crushing. The HPGR is also in closed circuit and reduces the 35 mm feed to a circuit product P_{80} of 4 mm. This is then fed to a closed circuit ball mill which takes the grind down to a P_{80} of 106 µm.

Secondary crushing specific energy

Combining eq 2 and 7:

$$W_c = 1 * 7.2 * 4 * \left(35000^{-(0.295+35000/1000000)} - 100000^{-(0.295+100000/1000000)}\right)$$

= 0.6 kWh/t

HPGR specific energy

Combining eq 2 and 8:

$$W_c = 1 * 13.9 * 4 * \left(4000^{-(0.295+4000/1000000)} - 35000^{-(0.295+35000/1000000)}\right)$$

= 2.9 kWh/t

Coarse particle tumbling mill specific energy

Combining eq 2 and 4:

$$W_a = 1 * 19.4 * 4 * \left(750^{-(0.295+750/1000000)} - 4000^{-(0.295+4000/1000000)}\right)$$

= 4.5 kWh/t

Fine particle tumbling mill specific energy

Combining eq 2 and 5:

$$W_b = 18.8 * 4 * \left(106^{-(0.295+106/1000000)} - 750^{-(0.295+750/1000000)}\right)$$

= 8.4 kWh/t

Total net comminution specific energy:

From eq 3:

W_T = 4.5 + 8.4 + 0.6 + 2.9 kWh/t

= 16.4 kWh/t

B 4.3 Conventional Crushing/Ball Milling Circuit

In this circuit primary crusher product is reduced in size to P_{80} of 6.5 mm via a secondary/tertiary crushing circuit (closed). This is then fed to a closed circuit ball mill which grinds to a P80 of 106 µm.

Secondary/tertiary crushing specific energy

Combining eq 2 and 7:

$$W_c = 1 * 7.2 * 4 * \left(6500^{-(0.295+6500/1000000)} - 100000^{-(0.295+100000/1000000)}\right)$$

= 1.7 kWh/t

Coarse particle tumbling mill specific energy

Combining eq 2 and 4:

$$W_a = 1 * 19.4 * 4 * \left(750^{-(0.295+750/1000000)} - 6500^{-(0.295+6500/1000000)}\right)$$

= 5.5 kWh/t

Fine particle tumbling mill specific energy

Combining eq 2 and 5:

$$W_b = 18.8 * 4 * \left(106^{-(0.295+106/1000000)} - 750^{-(0.295+750/1000000)}\right)$$

= 8.4 kWh/t

Size distribution correction

$$W_s = 0.19 * 19.4 * 4 * \left(6500^{-(0.295+6500/1000000)} - 100000^{-(0.295+100000/1000000)}\right)$$

= 0.9 kWh/t

Total net comminution specific energy:

From eq 3:

W_T = 5.5 + 8.4 + 1.7 + 0.9kWh/t

= 16.5 kWh/t

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APPENDIX E

Flotation Data Sheets

Laboratory Flotation Test Report

Test No.: 3230-102 **Date:** 3/26/10 **Technician:** C. Schultz, G. Hearn, and B. Chavez
Project: 11066
Objective: Direct spodumene flotation using Flotinor 1682 tall oil fatty acid as a collector
Conditions: 1,500 rpm in a 2-L stainless steel cell for the rougher and cleaner stages
Sample: 52346-7, minus 10 mesh (Dike 11.2) **Weight:** 1,000 g for rougher stage; 663.61 g for Cleaner 1 (22.0% solids); 601.55 g for Cleaner 2 (22.9% solids)
Grind: Sepor laboratory 9- by 10-in. rod mill at 50% solids for 5 min

Stage	Reagent, kg/t		Time, min			Pulp pH		Observations and Remarks
	200 g/L NaOH	Flotinor 1682	Grind	Cond	Froth	Start	End	
Rod Mill Grind 1	0.249		5					
Deslime								at 10–15 µm
Condition 1		0.754		10				55% solids
Rougher 1					5	7.5		Very highly mineralized grayish froth
Cleaner 1					4			Highly mineralized grayish froth
Cleaner 2					4			Highly mineralized grayish froth
Total	0.249	0.754		10	13			

Product	Dry Wt, g	Wt, %	Analysis, %						Weight, g						Distribution, %				
			Li						Li						Li				
Cleaner 2 concentrate	536.1	54.8	1.1						5.9						76.9				
Total concentrate	536.1	54.8	1.1						5.9						76.9				
Cleaner 2 tails	65.5	6.7	0.19						0.1						1.6				
Cleaner 1 tails	62.1	6.3	0.21						0.1						1.7				
Rougher tails	122.5	12.5	0.45						0.5						7.1				
Total tails	250.0	25.5	0.32						0.8						10.5				
Slimes	192.6	19.7	0.5						1.0						12.6				
Total slimes	192.6	19.7	0.5						1.0						12.6				
Calculated feed	978.7	80.3	0.78						7.7						100.0				
Feed assay			0.81																

Note: All reagent addition dosages are reported as kg/t of feed

Laboratory Flotation Test Report

Test No.: 3230-103 **Date:** 3/26/10 **Technician:** C. Schultz, G. Hearn, and B. Chavez
Project: 11066
Objective: Direct spodumene flotation using Sylfat FA-1 tall oil fatty acid as a collector and a polyglycol frother (Oreprep F-549)
Conditions: 1,500 rpm in a 2-L stainless steel cell for the rougher and cleaner stages
Sample: 52346-7, minus 10 mesh (Dike 11.2) **Weight:** 1,000 g for rougher stage; 334.86 g for Cleaner 1 (22.8% solids); 243.42 g for Cleaner 2 (17.7% solids)
Grind: Sepor laboratory 9- by 10-in. rod mill at 50% solids for 5 min

Stage	Reagent, kg/t			Time, min			Pulp pH		Observations and Remarks
	200 g/L NaOH	FA-1	F-549	Grind	Cond	Froth	Start	End	
Rod Mill Grind 1	0.249			5					
Deslime									at 10–15 µm
Condition 1		0.754			10				55% solids
Rougher 1			0.02			5	7.6		Very selective, dark gray froth, distinct endpoint
Cleaner 1						4			Very selective, dark gray froth, distinct endpoint
Cleaner 2			0.02			4			Very selective, dark gray froth, distinct endpoint
Total	0.249	0.754	0.04		10	13			

Product	Dry Wt, g	Wt, %	Analysis, %						Weight, g						Distribution, %				
			Li						Li						Li				
Cleaner 2 concentrate	224.7	22.6	2.3						5.3						61.5				
Total concentrate	224.7	22.6	2.3						5.3						61.5				
Cleaner 2 tails	18.7	1.9	1.71						0.3						3.7				
Cleaner 1 tails	91.4	9.2	0.61						0.6						6.5				
Rougher tails	457.5	46.0	0.31						1.4						16.4				
Total tails	567.6	57.1	0.40						2.3						26.6				
Slimes	201.5	20.3	0.5						1.0						11.9				
Total slimes	201.5	20.3	0.5						1.0						11.9				
Calculated feed	993.9	79.7	0.86						8.6						100.0				
Feed assay			0.81																

Note: All reagent addition dosages are reported as kg/t of feed

Laboratory Flotation Test Report

Test No.: 3230-107 **Date:** 4/21/10 **Technician:** C. Schultz and G. Hearn
Project: 11066
Objective: Direct spodumene flotation using Sylfat FA-1 tall oil fatty acid as a collector and a polyglycol frother (Oreprep F-549)
Conditions: 1,500 rpm in a 2-L stainless steel cell for the rougher stage and 1,200 rpm in a 1-L stainless steel cell for the cleaner stages
Sample: Composite 1 (minus 10 mesh) **Weight:** 1,000 g for rougher stage; 330.15 g for Cleaner 1 (18.2% solids); 256.39 g for Cleaner 2 (18.2% solids)
Grind: Sepor laboratory 9- by 10-in. rod mill at 50% solids for 8 min

Stage	Reagent, kg/t			Time, min			Pulp pH		Observations and Remarks
	200 g/L NaOH	FA-1	F-549	Grind	Cond	Froth	Start	End	
Rod Mill Grind 1	0.249			8					
Deslime-Cyclone									at 10–15 µm
Condition 1		0.754			10		7.7		55% solids
Rougher 1			0.02			5			Very selective, dark gray froth, distinct endpoint
Cleaner 1						4	6.5		Very selective, dark gray froth, distinct endpoint
Cleaner 2			0.02			4	6.6		Very selective, dark gray froth, distinct endpoint
Total	0.249	0.754	0.04		10	13			

Product	Dry Wt, g	Wt, %	Analysis, %					Weight, g					Distribution, %					
			Li	Estimated Spodumene				Li						Li				
Cleaner 2 concentrate	240.4	23.8	2.8					6.6						73.8				
Total concentrate	240.4	23.8	2.8	75.0				6.6						73.8				
Cleaner 2 tails	16.0	1.6	1.19					0.2						2.1				
Cleaner 1 tails	73.8	7.3	0.46	5.0				0.3						3.7				
Rougher tails	502.1	49.6	0.18	5.0				0.9						9.8				
Total tails	591.9	58.5	0.24					1.4						15.7				
Slimes	179.7	17.8	0.5					0.9						10.5				
Total slimes	179.7	17.8	0.5					0.9						10.5				
Calculated feed	1012.0	100.0	0.89					9.0						100.0				
Feed assay			0.78															

Note: All reagent addition dosages are reported as kg/t of feed

Laboratory Flotation Test Report

Test No.: 3230-108 **Date:** 4/21/10 **Technician:** C. Schultz and G. Hearn
Project: 11066
Objective: Direct spodumene flotation using Sylfat FA-1 tall oil fatty acid as a collector and a polyglycol frother (Oreprep F-549)
Conditions: 1,500 rpm in a 2-L stainless steel cell for the rougher stage and 1,200 rpm in a 1-L stainless steel cell for the cleaner stages
Sample: Composite 2 (minus 10 mesh) **Weight:** 1,000 g for rougher stage; 357.46 g for Cleaner 1 (24.1% solids); 288.97 g for Cleaner 2 (20.7% solids)
Grind: Sepor laboratory 9- by 10-in. rod mill at 50% solids for 8 min

Stage	Reagent, kg/t			Time, min			Pulp pH		Observations and Remarks
	200 g/L NaOH	FA-1	F-549	Grind	Cond	Froth	Start	End	
Rod Mill Grind 1	0.249			8					
Deslime-Cyclone									at 10–15 µm
Condition 1		0.754			10		7.9		55% solids
Rougher 1			0.02			5			Very selective, dark gray froth, distinct endpoint
Cleaner 1						4	6.7		Very selective, dark gray froth, distinct endpoint
Cleaner 2			0.02			4	6.9		Very selective, dark gray froth, distinct endpoint
Total	0.249	0.754	0.04		10	13			

Product	Dry Wt, g	Wt, %	Analysis, %					Weight, g					Distribution, %					
			Li	Estimated Spodumene				Li						Li				
Cleaner 2 concentrate	251.4	26.5	2.4					6.1						78.9				
Total concentrate	251.4	26.5	2.4	70.0				6.1						78.9				
Cleaner 2 tails	37.6	4.0	0.35					0.1						1.7				
Cleaner 1 tails	68.5	7.2	0.26					0.2						2.3				
Rougher tails	440.7	46.4	0.15	5.0				0.6						8.4				
Total tails	546.8	57.6	0.18					1.0						12.4				
Slimes	151.3	15.9	0.4					0.7						8.7				
Total slimes	151.3	15.9	0.4					0.7						8.7				
Calculated feed	949.5	100.0	0.81					7.7						100.0				
Feed assay			0.68															

Note: All reagent addition dosages are reported as kg/t of feed

Laboratory Flotation Test Report

Test No.: 3230-109 **Date:** 4/21/10 **Technician:** C. Schultz and G. Hearn
Project: 11066
Objective: Direct spodumene flotation using Sylfat FA-1 tall oil fatty acid as a collector and a polyglycol frother (Oreprep F-549)
Conditions: 1,500 rpm in a 2-L stainless steel cell for the rougher stage and 1,200 rpm in a 1-L stainless steel cell for the cleaner stages
Sample: Composite 3 (minus 10 mesh) **Weight:** 1,000 g for rougher stage; 314.66 g for Cleaner 1 (21.5% solids); 252.54 g for Cleaner 2 (16.9% solids)
Grind: Sepor laboratory 9- by 10-in. rod mill at 50% solids for 8 min

Stage	Reagent, kg/t			Time, min			Pulp pH		Observations and Remarks
	200 g/L NaOH	FA-1	F-549	Grind	Cond	Froth	Start	End	
Rod Mill Grind 1	0.249			8					
Deslime-Cyclone									at 10–15 µm
Condition 1		0.754			10		8.0		55% solids
Rougher 1			0.02			5			Very selective, dark gray froth, distinct endpoint
Cleaner 1						4	6.7		Very selective, dark gray froth, distinct endpoint
Cleaner 2			0.02			4	6.9		Very selective, dark gray froth, distinct endpoint
Total	0.249	0.754	0.04		10	13			

Product	Dry Wt, g	Wt, %	Analysis, %					Weight, g					Distribution, %					
			Li	Estimated Spodumene				Li						Li				
Cleaner 2 concentrate	239.0	25.2	2.5					5.9						76.7				
Total concentrate	239.0	25.2	2.5	70.0				5.9						76.7				
Cleaner 2 tails	13.6	1.4	1.25					0.2						2.2				
Cleaner 1 tails	62.1	6.6	0.29					0.2						2.4				
Rougher tails	491.6	51.9	0.15	5.0				0.7						9.5				
Total tails	567.3	59.9	0.19					1.1						14.1				
Slimes	141.6	14.9	0.5					0.7						9.2				
Total slimes	141.6	14.9	0.5					0.7						9.2				
Calculated feed	947.9	100.0	0.81					7.7						100.0				
Feed assay			0.71															

Note: All reagent addition dosages are reported as kg/t of feed

Laboratory Flotation Test Report

Test No.: 3230-110 **Date:** 4/21/10 **Technician:** C. Schultz and G. Hearn
Project: 11066
Objective: Direct spodumene flotation using Sylfat FA-1 tall oil fatty acid as a collector and a polyglycol frother (Oreprep F-549)
Conditions: 1,500 rpm in a 2-L stainless steel cell for the rougher stage and 1,200 rpm in a 1-L stainless steel cell for the cleaner stages
Sample: Composite 4 (minus 10 mesh) **Weight:** 1,000 g for rougher stage; 358.91 g for Cleaner 1 (22.7% solids); 283.41 g for Cleaner 2 (19.2% solids)
Grind: Sepor laboratory 9- by 10-in. rod mill at 50% solids for 8 min

Stage	Reagent, kg/t			Time, min			Pulp pH		Observations and Remarks
	200 g/L NaOH	FA-1	F-549	Grind	Cond	Froth	Start	End	
Rod Mill Grind 1	0.249			8					
Deslime-Cyclone									at 10–15 µm
Condition 1		0.754			10		8.0		55% solids
Rougher 1			0.02			5			Very selective, dark gray froth, distinct endpoint
Cleaner 1						4	6.8		Very selective, dark gray froth, distinct endpoint
Cleaner 2			0.02			4	6.8		Very selective, dark gray froth, distinct endpoint
Total	0.249	0.754	0.04		10	13			

Product	Dry Wt, g	Wt, %	Analysis, %					Weight, g					Distribution, %					
			Li	Estimated Spodumene				Li						Li				
Cleaner 2 concentrate	250.1	25.0	2.7					6.6						77.2				
Total concentrate	250.1	25.0	2.7	60.0				6.6						77.2				
Cleaner 2 tails	33.3	3.3	0.34					0.1						1.3				
Cleaner 1 tails	75.5	7.5	0.35					0.3						3.1				
Rougher tails	535.1	53.5	0.19	7.0				1.0						11.6				
Total tails	643.9	64.4	0.21					1.4						16.0				
Slimes	106.1	10.6	0.6					0.6						6.8				
Total slimes	106.1	10.6	0.6					0.6						6.8				
Calculated feed	1000.1	100.0	0.86					8.6						100.0				
Feed assay			0.71															

Note: All reagent addition dosages are reported as kg/t of feed

Laboratory Flotation Test Report

Test No.: 3230-111 **Date:** 4/28/10 **Technician:** C. Schultz and G. Hearn
Project: 11066
Objective: Direct spodumene flotation using Sylfat FA-1 tall oil fatty acid as a collector and a polyglycol frother (Oreprep F-549); Investigate higher NaOH dosage
Conditions: 1,500 rpm in a 2-L stainless steel cell for the rougher stage and 1,200 rpm in a 1-L stainless steel cell for the cleaner stages
Sample: Composite 1 (minus 10 mesh) **Weight:** 1,000 g for rougher stage; 315.37 g for Cleaner 1 (20.9% solids); 249.65 g for Cleaner 2 (16.9% solids)
Grind: Sepor laboratory 9- by 10-in. rod mill at 50% solids for 8 min

Stage	Reagent, kg/t		Time, min			Pulp pH		Observations and Remarks
	200 g/L NaOH	FA-1	Grind	Cond	Froth	Start	End	
Rod Mill Grind 1	1.356		8					
Deslime-Cyclone								at 10–15 µm
Condition 1		0.754		10		8.2		55% solids
Rougher 1					5			Very selective, dark gray froth, distinct endpoint
Cleaner 1					4	6.7		Very selective, dark gray froth, distinct endpoint
Cleaner 2					4	6.7		Very selective, dark gray froth, distinct endpoint
Total	1.356	0.754		10	13			

Product	Dry Wt, g	Wt, %	Analysis, %					Weight, g					Distribution, %					
			Li	Estimated Spodumene				Li						Li				
Cleaner 2 concentrate	222.4	22.4	3.0					6.6						76.7				
Total concentrate	222.4	22.4	3.0	90.0				6.6						76.7				
Cleaner 2 tails	16.0	1.6	0.55					0.1						1.0				
Cleaner 1 tails	73.8	7.4	0.41					0.3						3.5				
Rougher tails	502.1	50.5	0.13	2.0				0.7						7.7				
Total tails	591.9	59.5	0.18					1.1						12.2				
Slimes	179.7	18.1	0.5					1.0						11.1				
Total slimes	179.7	18.1	0.5					1.0						11.1				
Calculated feed	994.0	100.0	0.87					8.6						100.0				
Feed assay			0.78															

Note: All reagent addition dosages are reported as kg/t of feed

Laboratory Flotation Test Report

Test No.: 3230-112 **Date:** 4/28/10 **Technician:** C. Schultz and G. Hearn
Project: 11066
Objective: Direct spodumene flotation using Sylfat FA-1 tall oil fatty acid as a collector and a polyglycol frother (Oreprep F-549); Mica float on rougher tails using an amine (Aeromine 3000C) and fuel oil as collectors.
Conditions: 1,500 rpm in a 2-L stainless steel cell for the rougher stage and 1,200 rpm in a 1-L stainless steel cell for the cleaner stages
Sample: Composite 1 (minus 10 mesh) **Weight:** 1,000 g for rougher stage; ~307.28 g for Cleaner 1 (19.6% solids); 232.28 g for Cleaner 2 (15.0% solids); 210.01 g for Cleaner 3 (14.0% solids)
Grind: Sepor laboratory 9- by 10-in. rod mill at 50% solids for 8 min

Stage	Reagent, kg/t					Time, min			Pulp pH		Observations and Remarks
	200 g/L NaOH	FA-1	H_2SO_4	Aero 3000C	Fuel Oil #5	Grind	Cond	Froth	Start	End	
Rod Mill Grind 1	1.356					8					
Deslime-Cyclone											at 10–15 µm
Condition 1		0.754					10		8.2		55% solids
Rougher 1								5	6.8		Very selective, dark gray froth, distinct endpoint
Cleaner 1								4	6.9		Very selective, dark gray froth, distinct endpoint
Cleaner 2								3	7.0		Very selective, dark gray froth, distinct endpoint
Cleaner 3								3	7.1		
Mica condition 1			0.20				1				
Mica condition 2				0.35	2.55		3				
Mica rougher								2			non selective, heavy froth
Total	1.356	0.754	0.20	0.35	2.55		14	17			

Product	Dry Wt, g	Wt, %	Analysis, %					Weight, g					Distribution, %					
			Li	Estimated Spodumene				Li						Li				
Cleaner 3 concentrate	191.4	19.5	3.2					6.0						73.1				
Total concentrate	191.4	19.5	3.2	90.0				6.0						73.1				
Cleaner 3 tails	18.6	1.9	1.63					0.3						3.7				
Cleaner 2 tails	22.3	2.3	0.74					0.2						2.0				
Mica rougher 1 tails	6.8	0.7	0.60					0.0						0.5				
Mica rougher 1 concentrate	557.2	56.7	0.13	2.0				0.7						8.8				
Total tails	604.8	61.6	0.20					1.2						14.9				
Slimes	186.4	19.0	0.5					1.0						11.9				
Total slimes	186.4	19.0	0.5					1.0						11.9				
Calculated feed	982.6	100.0	0.84					8.3						100.0				
Feed assay			0.78															

Note: 1)All reagent addition dosages are reported as kg/t of feed 2) combined Cleaner 1 tails with rougher tails for Mica float

Laboratory Flotation Test Report

Test No.: 3230-113 **Date:** 5/4/10 **Technician:** C. Schultz and G. Hearn
Project: 11066
Objective: Direct spodumene flotation using Sylfat FA-1 tall oil fatty acid as a collector. Investigate coarse particle flotation after desliming and screening.
Conditions: 1,500 rpm in a 2-L stainless steel cell for the rougher stage and 1,200 rpm in a 1-L stainless steel cell for the cleaner stages
Sample: Composite 1, Cyclone underflow plus 100 mesh **Weight:** ~976 g for rougher stage; 421.29 g for Cleaner 1 (27.1% solids); 338.3 g for Cleaner 2 (22.2% solids)
Grind: Sepor laboratory 9- by 10-in. rod mill at 50% solids for 8 min

Stage	Reagent, kg/t		Time, min			Pulp pH		Observations and Remarks
	200 g/L NaOH	FA-1	Grind	Cond	Froth	Start	End	
Rod Mill Grind 1	1.356		8					
Deslime-Cyclone								at 10–15 µm
Screen underflow								at 100 mesh
Condition 1		0.754		10		8.0		55% solids
Rougher 1					5			Highly mineralized, dark gray heavy froth, distinct endpoint
Cleaner 1					4	6.9		Highly mineralized, dark gray heavy froth, distinct endpoint
Cleaner 2					3	7.1		Highly mineralized, dark gray heavy froth, distinct endpoint
Total	1.356	0.754		10	12			

Product	Dry Wt, g	Wt, %	Analysis, %					Weight, g					Distribution, %					
			Li	Estimated Spodumene				Li						Li				
Cleaner 2 concentrate	302.5	15.6	3.2	80–85				9.6						58.1				
Total concentrate	302.5	15.6	3.2					9.6						58.1				
Cleaner 2 tails	35.8	1.8	0.78	15–20				0.3						1.7				
Cleaner 1 tails	83.0	4.3	0.36	5				0.3						1.8				
Rougher tails	573.0	29.6	0.10	<1				0.6						3.4				
Total tails	691.8	35.7	0.16					1.1						6.9				
Minus 100 mesh	600.6	31.0	0.67					4.0						24.3				
Total minus 100 mesh	600.6	31.0	0.67					4.0						24.3				
Slimes	343.6	17.7	0.51					1.8						10.7				
Total slimes	343.6	17.7	0.51					1.8						10.7				
Calculated feed	1938.5	100.0	0.85					16.6						100.0				
Feed assay			0.78															

Note: All reagent addition dosages are reported as kg/t of feed

Laboratory Flotation Test Report

Test No.: 3230-114 **Date:** 5/4/10 **Technician:** C. Schultz and G. Hearn
Project: 11066
Objective: Direct spodumene flotation using Sylfat FA-1 tall oil fatty acid as a collector. Investigate fine particle flotation after desliming and screening.
Conditions: 1,500 rpm in a 2-L stainless steel cell for the rougher stage and 1,200 rpm in a 1-L stainless steel cell for the cleaner stages
Sample: Composite 1, Cyclone underflow minus 100 mesh **Weight:** ~760 g for rougher stage; 205.33 g for Cleaner 1 (13.6% solids); 164.2 g for Cleaner 2 (11.5% solids)
Grind: Sepor laboratory 9- by 10-in. rod mill at 50% solids for 8 min

Stage	Reagent, kg/t		Time, min			Pulp pH		Observations and Remarks
	200 g/L NaOH	FA-1	Grind	Cond	Froth	Start	End	
Rod Mill Grind 1	1.356		8					
Deslime-Cyclone								at 10–15 µm
Screen underflow								at 100 mesh
Condition 1		0.754		10		8.2		55% solids
Rougher 1					4			Highly mineralized, light greenish gray heavy froth, distinct endpoint
Cleaner 1					3	7.2		Highly mineralized, light greenish gray heavy froth, distinct endpoint
Cleaner 2					3	7.1		Highly mineralized, light greenish gray heavy froth, distinct endpoint
Total	1.356	0.754		10	10			

Product	Dry Wt, g	Wt, %	Analysis, %					Weight, g					Distribution, %					
			Li	Estimated Spodumene				Li						Li				
Cleaner 2 concentrate	146.7	7.6	2.5	90				3.7						22.1				
Total concentrate	146.7	7.6	2.5					3.7						22.1				
Cleaner 2 tails	17.5	0.9	0.61	10				0.1						0.6				
Cleaner 1 tails	41.1	2.1	0.26	5				0.1						0.6				
Rougher tails	395.3	20.4	0.05	<1				0.2						1.1				
Total tails	453.9	23.4	0.09					0.4						2.4				
Plus 100 mesh	994.3	51.3	1.08					10.7						64.9				
Total plus 100 mesh	994.3	51.3	1.08					10.7						64.9				
Slimes	343.6	17.7	0.51					1.8						10.7				
Total slimes	343.6	17.7	0.51					1.8						10.7				
Calculated feed	1938.5	100.0	0.85					16.6						100.0				
Feed assay			0.78															

Note: All reagent addition dosages are reported as kg/t of feed

Laboratory Flotation Test Report

Test No.: 3230-115 **Date:** 5/18/10 **Technician:** C. Schultz, J. Londono, and G. Hearn
Project: 11066
Objective: Direct spodumene flotation using Sylfat FA-1 tall oil fatty acid as a collector. Bulk coarse particle flotation after after desliming and screening to produce concentrate for roasting and leaching.
Conditions: 1,500 rpm in a 2-L stainless steel cell for the rougher stage and 1,200 rpm in a 1-L stainless steel cell for the cleaner stages
Sample: Master composite, cyclone underflow plus 100 mesh **Weight:** Approximately 35% solids for rougher, 27% solids for Cleaner 1, 22% solids for Cleaner 2
Grind: Sepor laboratory 9- by 10-in. rod mill at 50% solids for 8 min

Stage	Reagent, kg/t		Time, min			Pulp pH		Observations and Remarks
	200 g/L NaOH	FA-1	Grind	Cond	Froth	Start	End	
Rod Mill Grind 1	1.356		8					
Deslime-Cyclone								at 10–15 µm
Screen underflow								at 100 mesh
Condition 1		0.754		10		7.6		55% solids
Rougher 1					5			Highly mineralized, dark gray heavy froth, distinct endpoint
Cleaner 1					4	6.9		Highly mineralized, dark gray heavy froth, distinct endpoint
Cleaner 2					3	7.2		Highly mineralized, dark gray heavy froth, distinct endpoint
Total	1.356	0.754		10	12			

Product	Dry Wt, g	Wt, %	Analysis, %					Weight, g					Distribution, %					
			Li					Li						Li				
Cleaner 2 concentrate	1,768	11.4	3.11					55.0						43.8				
Total concentrate	1,768	11.4	3.11					55.0						43.8				
Cleaner 2 tails	254.7	1.6	1.45					3.7						2.9				
Cleaner 1 tails	579.5	3.7	0.83					4.8						3.8				
Rougher tails	4,871	31.5	0.17					8.5						6.7				
Total tails	5,705	36.8	0.30					17.0						13.5				
Minus 100 mesh	5,450	35.2	0.74					40.5						32.2				
Total minus 100 mesh	5,450	35.2	0.74					40.5						32.2				
Slimes	2,561	16.5	0.51					13.2						10.5				
Total slimes	2,561	16.5	0.51					13.2						10.5				
Calculated feed	15,484	100.0	0.81					125.6						100.0				
Feed assay			0.81															

Note: All reagent addition dosages are reported as kg/t of feed

Laboratory Flotation Test Report

Test No.: 3230-116 **Date:** 5/18/10 **Technician:** C. Schultz, J. Londono, and G. Hearn
Project: 11066
Objective: Direct spodumene flotation using Sylfat FA-1 tall oil fatty acid as a collector. Bulk fine particle flotation after after desliming and screening to produce concentrate for roasting and leaching.
Conditions: 1,500 rpm in a 2-L stainless steel cell for the rougher stage and 1,200 rpm in a 1-L stainless steel cell for the cleaner stages
Sample: Master composite, cyclone underflow minus 100 mesh **Weight:** Approximately 35% solids for rougher, 14% solids for Cleaner 1, 12% solids for Cleaner 2
Grind: Sepor laboratory 9- by 10-in. rod mill at 50% solids for 8 min

Stage	Reagent, kg/t		Time, min			Pulp pH		Observations and Remarks
	200 g/L NaOH	FA-1	Grind	Cond	Froth	Start	End	
Rod Mill Grind 1	1.356		8					
Deslime-Cyclone								at 10–15 µm
Screen underflow								at 100 mesh
Condition 1		0.754		10		8.1		55% solids
Rougher 1					5			Highly mineralized, light greenish gray heavy froth, distinct endpoint
Cleaner 1					4	7.0		Highly mineralized, light greenish gray heavy froth, distinct endpoint
Cleaner 2					3	7.2		Highly mineralized, light greenish gray heavy froth, distinct endpoint
Total	1.356	0.754		10	12			

Product	Dry Wt, g	Wt, %	Analysis, %					Weight, g					Distribution, %					
			Li					Li						Li				
Cleaner 2 concentrate	1,210	7.8	2.91					35.2						28.0				
Total concentrate	1,210	7.8	2.91					35.2						28.0				
Cleaner 2 tails	157.9	1.0	1.03					1.6						1.3				
Cleaner 1 tails	368.2	2.4	0.41					1.5						1.2				
Rougher tails	3,714	24.0	0.06					2.2						1.7				
Total tails	4,240	27.4	0.13					5.3						4.2				
Plus 100 mesh	7,473	48.3	0.96					72.0						57.3				
Total plus 100 mesh	7,473	48.3	0.96					72.0						57.3				
Slimes	2,561	16.5	0.51					13.2						10.5				
Total slimes	2,561	16.5	0.51					13.2						10.5				
Calculated feed	15,484	100.0	0.81					125.6						100.0				
Feed assay			0.81															

Note: All reagent addition dosages are reported as kg/t of feed

APPENDIX F

Calcining Data Sheets

4-INCH BATCH QUARTZ KILN
Operational Data Sheet

Test # 1 (~~SHAKEDOWN~~)
Sample ID Clz Conc.
Conditions 1000°C 1hr RT

Project # 11066
Date 5/24/10

Time	TEMPERATURES Kiln Shell °C	Kiln Burden °C	KILN Rotational Speed rpm	GAS FLOW Air to Kiln scfm	OFF GAS ANALYSES O_2 %	CO %	CO_2 %	SO_2 %		PRESSURE Kiln Inlet "H2O	Kiln Outlet "H2O
0656	20	20	1.0	0.1	20.9	0	0	0	"	2.0	
0715	446	390	1.0	0.1	20.9	0	0	0	OFF LINE	2.0	—
0730	677	653	1.0	0.1	19.9	0	0.2	0		2.0	
0745	843	826	1.0	0.1	20.1	0	0.1	0		2.0	
0800	974	957	1.0	0.1	20.0	0	0	0		2.0	
0815	1009	999	1.0	0.1	20.1	0	0	0		2.0	
0830	1011	1001	1.0	0.1	20.3	0	0	0		2.0	
0845	1011	1000	1.0	0.1	20.6	0	0	0		2.0	
0900	1011	1000	1.0	0.1	20.7	0	0	0		2.0	
0915	1011	1001	1.0	0.1	20.7	0	0	0		2.0	
			Switch To								
				0.1 scfm N_2 Flow							

MASS DATA

Mass of Sample Tested, g:
Gross Weight 2230.9
Kiln Tare 1927.5
Net Weight 303.4

Mass Treated, g:
Gross Weight 2229.0
Kiln Tare 1927.5
Net Weight 301.5

Condensate Data:
Mass, g 0
Volume, cc 0

Product Recovery Data:
Total Mass In, g 303.4
Total Mass Out, g 301.5
Recovery, % 99.4

Recovery % = Mass out X 100 / Mass In

Total Recovery, % = 99.4
(W/ Condensate)

Impinger Data:

Starting Wt., g	Ending Wt., g
#1 2035.5	#1 2031.0
#2 159.8	#2 160.0
Total 2195.3	Total 2191.0

Condensate Net Gain, g -4.3

Observations :
0656 Heat ON, TARGETING 1000°C BURDEN IN (NR)
0716 GAS ANALYSIS LINE NOW HOOKED UP FOR EMISSIONS —
0815 BURDEN @ 1000°C, START 60 MIN RT
0915 END TEST, SWITCH TO N_2 FLOW FOR COOLING
CALCINE IS A FREE FLOWING (PINK) MATERIAL, EASILY REMOVED.

4-INCH BATCH QUARTZ KILN
Operational Data Sheet

Test # 2
Sample ID Cl_2 Conc.
Conditions 1050 °C, 60 min

Project # 11064
Date 5/24/10

	TEMPERATURES		KILN	GAS FLOW	OFF GAS ANALYSES					PRESSURE		MASS DATA
	Kiln Shell	Kiln Burden	Rotational Speed	Air to Kiln	O_2	CO	CO_2	SO_2		Kiln Inlet	Kiln Outlet	
Time	°C	°C	rpm	scfm	%	%	%	%		"H2O	"H2O	
1053	319	34	1.0	0.1	20.3	0	0	0		2.0		Mass of Sample Tested, g:
1100	460	370	1.0	0.1	16.5	0.5	0.8	0		2.0		Gross Weight 2235.4
1115	729	706	1.0	0.1	20.0	0	0.2	0		2.0		Kiln Tare 1930.6
1130	905	888	1.0	0.1	20.2	0	0.1	0		2.0		Net Weight 304.8
1153	1058	1049	1.0	0.1	20.4	0	0	0		2.0		Mass Treated, g:
1215	1058	1051	1.0	0.1	20.5	0	0	0		2.0		Gross Weight 2228.2
1230	1057	1050	1.0	0.1	20.6	0	0	0		2.0		Kiln Tare 1930.6
1245	1057	1050	1.0	0.1	20.8	0	0	0		2.0		Net Weight 297.6
1253	1057	1050	1.0	0.1	20.8	0	0	0		2.0		
		End	0.1 N_2 Flow ON For cooling									Condensate Data:
												Mass, g 0
												Volume, cc 0
												Product Recovery Data:
												Total Mass In, g 304.8
												Total Mass Out, g 297.6
												Recovery, % 97.6
												Recover,y % = Mass out X 100 / Mass In
												Total Recovery, % = 97.6 (W/ Condensate)

Impinger Data:

Starting Wt., g		Ending Wt., g	
#1	2030.5	#1	2026.5
#2	159.9	#2	160.2
Total	2190.4	Total	2186.7

Condensate Net Gain, g -3.7

Observations :

1053 Heat ON For 1050°C Burden Temp.
W/ exhaust line To Gas Analyzers.
This time shell preheated

W/ Burden @ ~375°C, seeing Co & CO_2 IN off gas.
1102 changed out in line Filter to analyzers (spike)

1153 Burden @ 1050°C, start 1hr RT (Air Atmosphere)

1253 End Test 2, Heat OFF, N_2 ON For Calcine cooling.

4-INCH BATCH QUARTZ KILN
Operational Data Sheet

Project # 1100
Date 5/25/10

Test # 3
Sample ID Cl_2 Cont.
Conditions 950 °C, 60 min

Time	Kiln Shell °C	Kiln Burden °C	Kiln Rotational Speed rpm	Air to Kiln scfm	O_2 %	CO %	CO_2 %	SO_2 %		Kiln Inlet "H2O	Kiln Outlet "H2O
0636	20	19	1.0	0.1	20.7	0	0	0		2.0	
0645	230	114	1.0	0.1	20.8	0	0	0		2.0	
0700	539	497	1.0	0.1	19.3	0	0.8	0		2.0	
0715	740	720	1.0	0.1	20.2	0	0.1	0		2.0	
0730	890	875	1.0	0.1	20.4	0	0	0		2.0	
0745	957	950	1.0	0.1	20.5	0	0	0		2.0	
0800	956	950	1.0	0.1	20.5	0	0	0		2.0	
0815	956	950	1.0	0.1	20.6	0	0	0		2.0	
0830	956	950	1.0	0.1	20.8	0	0	0		2.0	
0845	955	950	1.0	0.1	20.9	0	0	0		2.0	
				N_2 Flow on for cooling							

MASS DATA

Mass of Sample Tested, g:
Gross Weight 2238.7
Kiln Tare 1935.9
Net Weight 302.8

Mass Treated, g:
Gross Weight 2237.2
Kiln Tare 1935.9
Net Weight 301.3

Condensate Data:
Mass, g 0
Volume, cc 0

Product Recovery Data:
Total Mass In, g 302.8
Total Mass Out, g 301.3
Recovery, % 99.5

Recovery % = Mass out X 100
Mass In

Total Recovery, % = 99.5
(W/ Condensate)

Impinger Data:
Starting Wt., g
#1 2026.0
#2 159.8
Total 2185.8

Ending Wt., g
#1 2021.2
#2 160.3
Total 2181.5

Condensate Net Gain, g -4.3

Observations :
0636 Heat on for 950 °C Burden Temp.
0655 Cl_2 peaked @ 1.9 % Burden Temp was ~ 395 °C
CO peaked @ 0.4 % also @ ~ 395 °C
0745 Burden @ 950 °C, start 1hr RT
0845 End Test 3, switch to N_2 flow for cooling.

4-INCH BATCH QUARTZ KILN
Operational Data Sheet

Test # 4
Sample ID Qtz conc.
Conditions 1000°C, 60 min

Project # 11066
Date 5-26-10

	TEMPERATURES		KILN	GAS FLOW	OFF GAS ANALYSES					PRESSURE	
	Kiln Shell	Kiln Burden	Rotational Speed	Air to Kiln	O_2	CO	CO_2			Kiln Inlet	Kiln Outlet
Time	°C	°C	rpm	scfm	%	%	%			"H2O	"H2O
0612	25	24	1.0	0.1	20.7	0	0				
0630	442	380	1.0	0.1	16.4	0.3	1.8				
0645	624	649	1.0	0.1	19.1	0	0.2				
0700	840	825	1.0	0.1	19.7	0	0				
0715	971	959	1.0	0.1	20.2	0	0				
0729	1008	1000	1.0	0.1	20.2	0	0				
0745	1005	1000	1.0	0.1	20.4	0	0				
0800	1006	1000	1.0	0.1	20.6	0	0				
0815	1006	1000	1.0	0.1	20.7	0	0				
0829	1005	1000	1.0	0.1	20.8	0	0				

MASS DATA

Mass of Sample Tested, g:
Gross Weight 2236.7
Kiln Tare 1934.0
Net Weight 302.7

Mass Treated, g:
Gross Weight 2235.1
Kiln Tare 1934.0
Net Weight 301.1

Condensate Data:
Mass, g 0
Volume, cc 0

Product Recovery Data:
Total Mass In, g 302.7
Total Mass Out, g 301.1
Recovery, % 99.5

Recovery % = Mass out X 100 / Mass In

Total Recovery, % = 99.5 (W/ Condensate)

Impinger Data:
Starting Wt., g: #1 2020.3, #2 160.1, Total 2180.4
Ending Wt., g: #1 2016.1, #2 160.7, Total 2176.8

Condensate Net Gain, g -3.6

Observations :
0612 Heat ON, Targeting 1000°C BURDEN
0630 CO spike ~0.3% & CO_2 max ~1.8% w/Burden ~370°C
0729 Burden @ 1000°C, start 60 min RT
0829 End Test 4, Switch to N_2 flow for cooling

4-INCH BATCH QUARTZ KILN
Operational Data Sheet

Test # 5
Sample ID Cl₂ conc.
Conditions 1050 °C, 30 min

Project # 11066
Date 5-26-10

Time	Kiln Shell °C	Kiln Burden °C	Kiln Rotational Speed rpm	Air to Kiln scfm	O_2 %	CO %	CO_2 %			Kiln Inlet "H2O	Kiln Outlet "H2O
1055	264	32	1.0	0.1	20.8	0	0			2.0	
1115	635	603	1.0	0.1	19.1	0	0.8			2.0	
1130	826	809	1.0	0.1	19.8	0	0			2.0	
1145	967	954	1.0	0.1	19.6	0	0			2.0	
1201	1058	1049	1.0	0.1	19.6	0	0			2.0	
1216	1054	1050	1.0	0.1	19.7	0	0			2.0	
1231	1055	1050	1.0	0.1	19.8	0	0			2.0	
				N_2 ON FOR Cooling							
					0	0	0				

MASS DATA

Mass of Sample Tested, g:
Gross Weight 2236.8
Kiln Tare 1936.9
Net Weight 299.9

Mass Treated, g:
Gross Weight 2235.2
Kiln Tare 1936.9
Net Weight 298.3

Condensate Data:
Mass, g —
Volume, cc —

Product Recovery Data:
Total Mass In, g 299.9
Total Mass Out, g
Recovery, %

Recovery % = Mass out X 100
Mass In

Total Recovery, % =
(W/ Condensate)

Impinger Data:
Starting Wt., g — Ending Wt., g —
#1 — #1 —
#2 — #2 —
Total — Total —

Condensate Net Gain, g —

Observations :
1055 Heat ON FOR 1050°C BURDEN
1104 Cl_2 PEAKED @ 0.5 %, BURDEN ~ 350 °C
1106 Cl_2 PEAKED @ 2.5 %, BURDEN ~ 420 °C
1201 BURDEN @ 1050 °C, Start 30 MIN RT
1231 END Test 5, N_2 ON FOR Cooling

4-INCH BATCH QUARTZ KILN
Operational Data Sheet

Test # 6
Sample ID Cl_2 conc.
Conditions 1050°C, 1hr

Project # 11066
Date 6-4-10

	TEMPERATURES		KILN	GAS FLOW	OFF GAS ANALYSES					PRESSURE		MASS DATA
Time	Kiln Shell °C	Kiln Burden °C	Rotational Speed rpm	Air to Kiln scfm	O_2 %	CO %	CO_2 %			Kiln Inlet "H2O	Kiln Outlet "H2O	
0600	25	25	1.0	0.1	—	0.1	0.1			2.0		Mass of Sample Tested, g:
0615	347	158	1.0	0.1	—	0.2	0.4			2.0		Gross Weight 3143.0
0630	590	462	1.0	0.1	—	1.4	8.3			2.0		Kiln Tare 1937.9
0645	770	690	1.0	0.1	—	0.2	1.4			2.0		Net Weight 1205.1
0700	910	847	1.0	0.1	—	0.1	0.5			2.0		
0715	1023	973	1.0	0.1	—	0.1	0.3			2.0		Mass Treated, g:
0730	1075	1019	1.0	0.1	—	0.1	0.1			2.0		Gross Weight 3136.6
												Kiln Tare 1937.9
0743	1074	1050	1.0	0.1	—	0	0			2.0		Net Weight 1198.7
0800	1051	1052	1.0	0.1	—	0	0			2.0		
0815	1054	1048	1.0	0.1	—	0	0			2.0		Condensate Data:
0830	1055	1050	1.0	0.1	—	0	0			2.0		Mass, g —
0843	1053	1051	1.0	0.1	—	0	0			2.0		Volume, cc —
												Product Recovery Data:
												Total Mass In, g 1205.1
												Total Mass Out, g 1198.7
												Recovery, % 99.5
												Recover,y % = Mass out X 100 / Mass In
												Total Recovery, % = 99.5 (W/ Condensate)

Impinger Data:
Starting Wt., g — Ending Wt., g —

#1 — #1 —
#2 — #2 —
Total — Total —

Condensate Net Gain, g —

Observations:
0600 Heat on for 1050°C Burden.
0620 Cl_2 peak of 2.9 %, Burden ~ 275 °C
0626 Cl_2 peak of 12.7 %, Burden ~ 390 °C
0743 Burden @ 1050 °C, start 1hr RT
0843 End Test 6, switch to Argon for cooling.

4-INCH BATCH QUARTZ KILN
Operational Data Sheet

Test # 7
Sample ID Cl_2 Conc.
Conditions 1050, 60 min

Project # 11066
Date 6-7-10

Time	TEMPERATURES: Kiln Shell °C	Kiln Burden °C	KILN: Rotational Speed rpm	GAS FLOW: Air to Kiln scfm	OFF GAS ANALYSES: O_2 %	CO %	CO_2 %			PRESSURE: Kiln Inlet "H2O	Kiln Outlet "H2O
0626	22	22	1.0	0.1	—	0	0			2.0	
0645	439	286	1.0	0.1	—	0.2	0.1			2.0	
0700	673	600	1.0	0.1	—	0.5	0.5			2.0	
0715	842	802	1.0	0.1	—	0.2	0.2			2.0	
0730	969	939	1.0	0.1	—	0.1	0.1			2.0	
0745	1050	1028	1.0	0.1	—	0	0			2.0	
0800	1060	1050	1.0	0.1	—	0	0			2.0	
0815	1056	1049	1.0	0.1	—	0	0			2.0	
0830	1058	1051	1.0	0.1	—	0	0			2.0	
0845	1058	1050	1.0	0.1	—	0	0			2.0	
0900	1057	1051	1.0	0.1	—	0	0			2.0	

MASS DATA

Mass of Sample Tested, g:
Gross Weight 2447.8
Kiln Tare 1951.3
Net Weight 496.5

Mass Treated, g:
Gross Weight 2446.7
Kiln Tare 1951.3
Net Weight 495.4

Condensate Data:
Mass, g —
Volume, cc —

Product Recovery Data:
Total Mass In, g 496.5
Total Mass Out, g 495.4
Recovery, % 100

Recovery % = Mass out X 100
Mass In

Total Recovery, % = 100
(W/ Condensate)

Impinger Data:

Starting Wt., g		Ending Wt., g	
#1	—	#1	—
#2	—	#2	—
Total	—	Total	—

Condensate Net Gain, g _____

Observations :
0626 Heat ON FOR 1050°C BURDEN
0800 BURDEN @ 1050°C, START 1 Wk RT
0900 Heat OFF, switch to ARGON FOR cooling

4-INCH BATCH QUARTZ KILN
Operational Data Sheet

Test # 8
Sample ID Ukwe, 1345
Conditions 1050 °C, Cousin

Project # 1106
Date 6/7/10

Time	TEMPERATURES: Kiln Shell °C	TEMPERATURES: Kiln Burden °C	KILN: Rotational Speed rpm	GAS FLOW: Air to Kiln scfm	OFF GAS ANALYSES:	OFF GAS ANALYSES: CO %	OFF GAS ANALYSES: CO_2 %	OFF GAS ANALYSES:	OFF GAS ANALYSES:	PRESSURE: Kiln Inlet "H2O	PRESSURE: Kiln Outlet "H2O
1105	301	33	1.0	0.1		0.1	0.3			2.4	
1115	467	217	1.0	0.1		0.1	0.2			2.4	
1130	705	602	1.0	0.1		0.1	0.2			2.4	
1145	881	835	1.0	0.1		0	0.1			2.4	
1200	1015	987	1.0	0.1		0	0			2.4	
1215	1061	1051	1.0	0.1		0	0			2.4	
1230	1053	1052	1.0	0.1		0	0			2.4	
1245	1053	1051	1.0	0.1		0	0			2.4	
1300	1052	1050	1.0	0.1		0	0			2.4	
1315	1052	1050	1.0	0.1		0	0			2.4	

MASS DATA

Mass of Sample Tested, g:
Gross Weight 2731.8
Kiln Tare 1940.1
Net Weight 791.7

Mass Treated, g:
Gross Weight 2682.5 (Kiln SWAS)
Kiln Tare 1940.1
Net Weight 791.3 48.9

Condensate Data:
Mass, g —
Volume, cc —

Product Recovery Data:
Total Mass In, g 791.7
Total Mass Out, g 791.3
Recovery, % 99.9

Recovery % = Mass out X 100
Mass In

Total Recovery, % = 99.9
(W/ Condensate)

Impinger Data:
Starting Wt., g
#1
#2
Total

Ending Wt., g
#1
#2
Total

Condensate Net Gain, g

Observations :
1105 HEAT ON FOR 1050°C BURDEN TEMP. (NEW KILN)
1215 BURDEN @ 1050 °C START 1hr RT
1315 END TEST B, ARGON GAS ON FOR COOLING

4-INCH BATCH QUARTZ KILN
Operational Data Sheet

Test # 9
Sample ID Calcine 1, 3 & 5
Conditions 1050°C 1hr RT

Project # 11064
Date 6-8-10

Time	Kiln Shell °C	Kiln Burden °C	Kiln Rotational Speed rpm	Air to Kiln scfm		CO %	CO_2 %			Kiln Inlet "H2O	Kiln Outlet "H2O
0600	35	24	1.0	0.1		0	0			2.0	
0615	362	296	1.0	0.1		0	0			2.0	
0630	635	607	1.0	0.1		0	0			2.0	
0645	818	802	1.0	0.1		0.1	0.1			2.0	
0700	948	934	1.0	0.1		0	0			2.0	
0715	1046	1036	1.0	0.1		0	0			2.0	
0720	1057	1048	1.0	0.1		0	0			2.0	
0735	1057	1049	1.0	0.1		0	0			2.0	
0750	1058	1051	1.0	0.1		0	0			2.0	
0805	1058	1050	1.0	0.1		0	0			2.0	
0820	1053	1050	1.0	0.1		0	0			2.0	
				0.1 Setting Argon Flow @ start Down							

MASS DATA

Mass of Sample Tested, g:
Gross Weight 2070.0
Kiln Tare 1947.2
Net Weight 122.8

Mass Treated, g:
Gross Weight 2069.8
Kiln Tare 1947.2
Net Weight 122.6

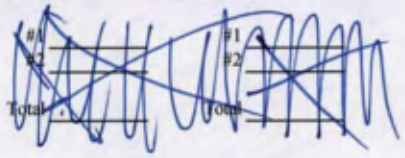
Condensate Data:
Mass, g —
Volume, cc —

Product Recovery Data:
Total Mass In, g 122.8
Total Mass Out, g 122.6
Recovery, % 100

Recovery % = Mass out X 100 / Mass In

Total Recovery, % = 100.
(W/ Condensate)

Impinger Data:
Starting Wt., g Ending Wt., g



Condensate Net Gain, g _____

Observations :
0600 Heat on for 1050°C Burnout
Test of Tests 1, 3 & 5 Calcine
for 1050°C Reheating.
0720 End @ 1050°C, start 1 hr RT
0820 End Test 9, Switch to Argon Gas for Cooling

After 100g Head Sample Total 1050°C Calcine for Test = 2553 g

APPENDIX G

XRD Analyses of Flotation Concentrate and Calcines

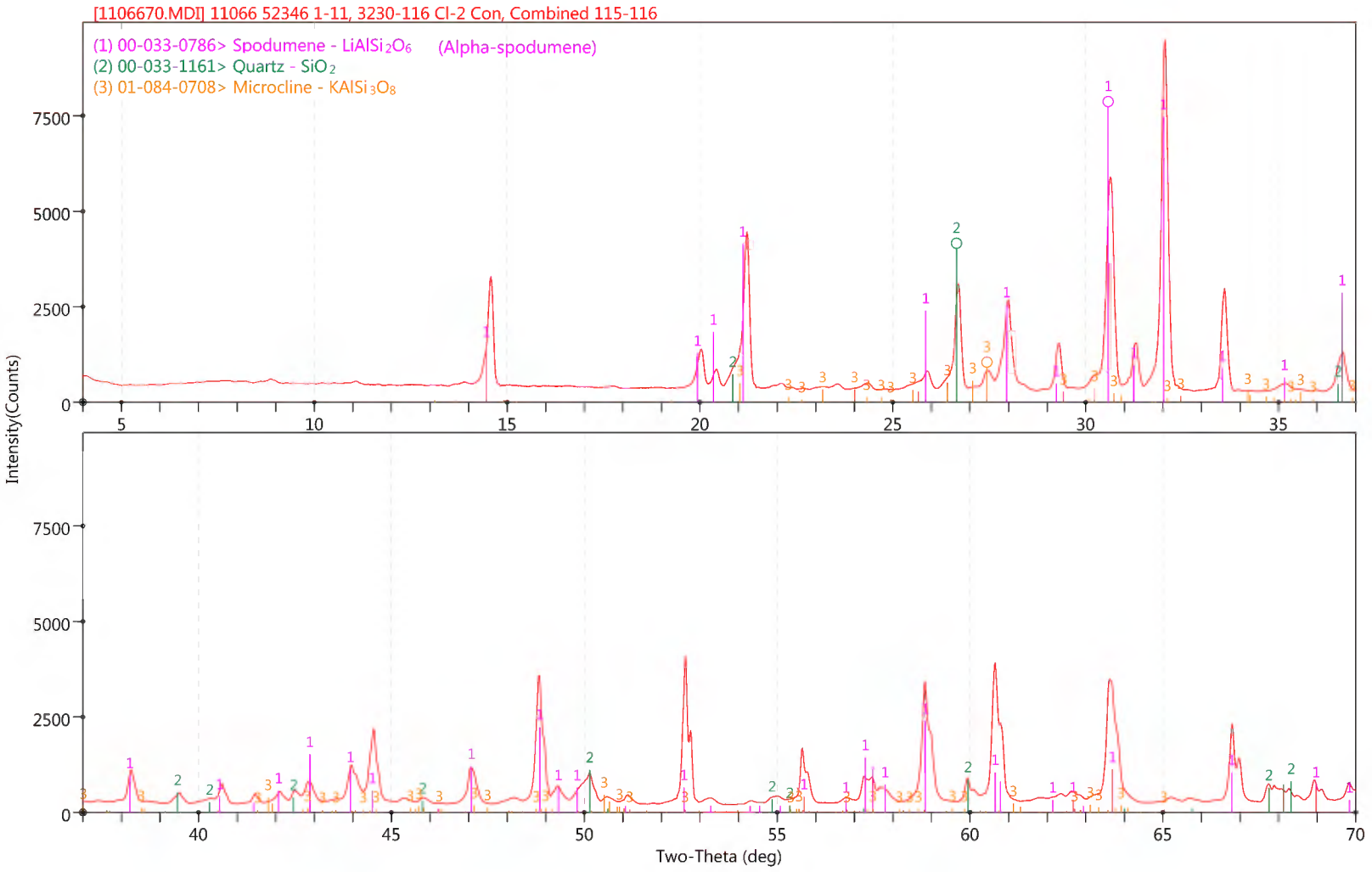


Figure G1. XRD Pattern, Feed Sample (Composite Dikes 7.2–15.1)

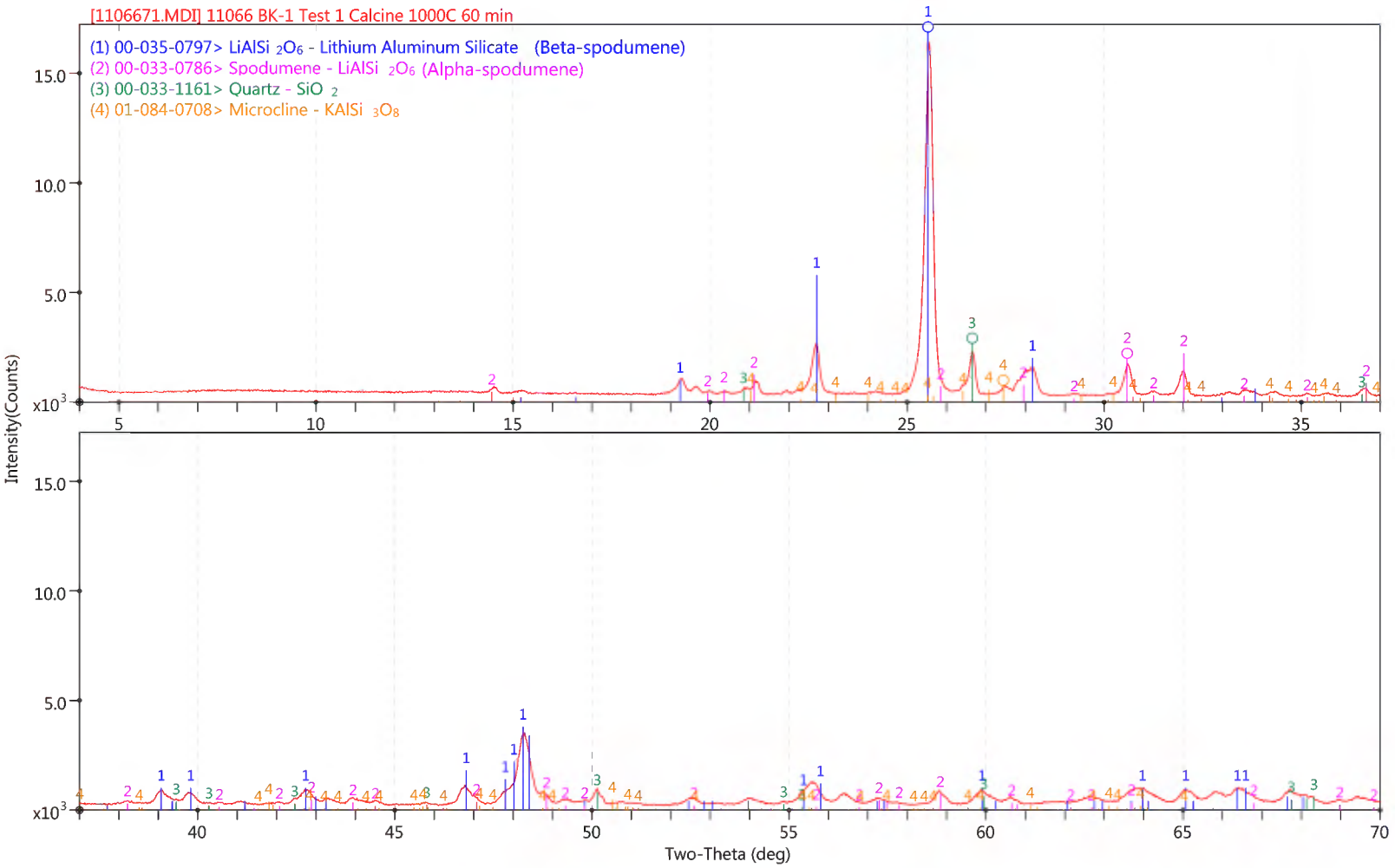


Figure G2. XRD Pattern, Test 1 Calcine, 1,000°C, 60 min

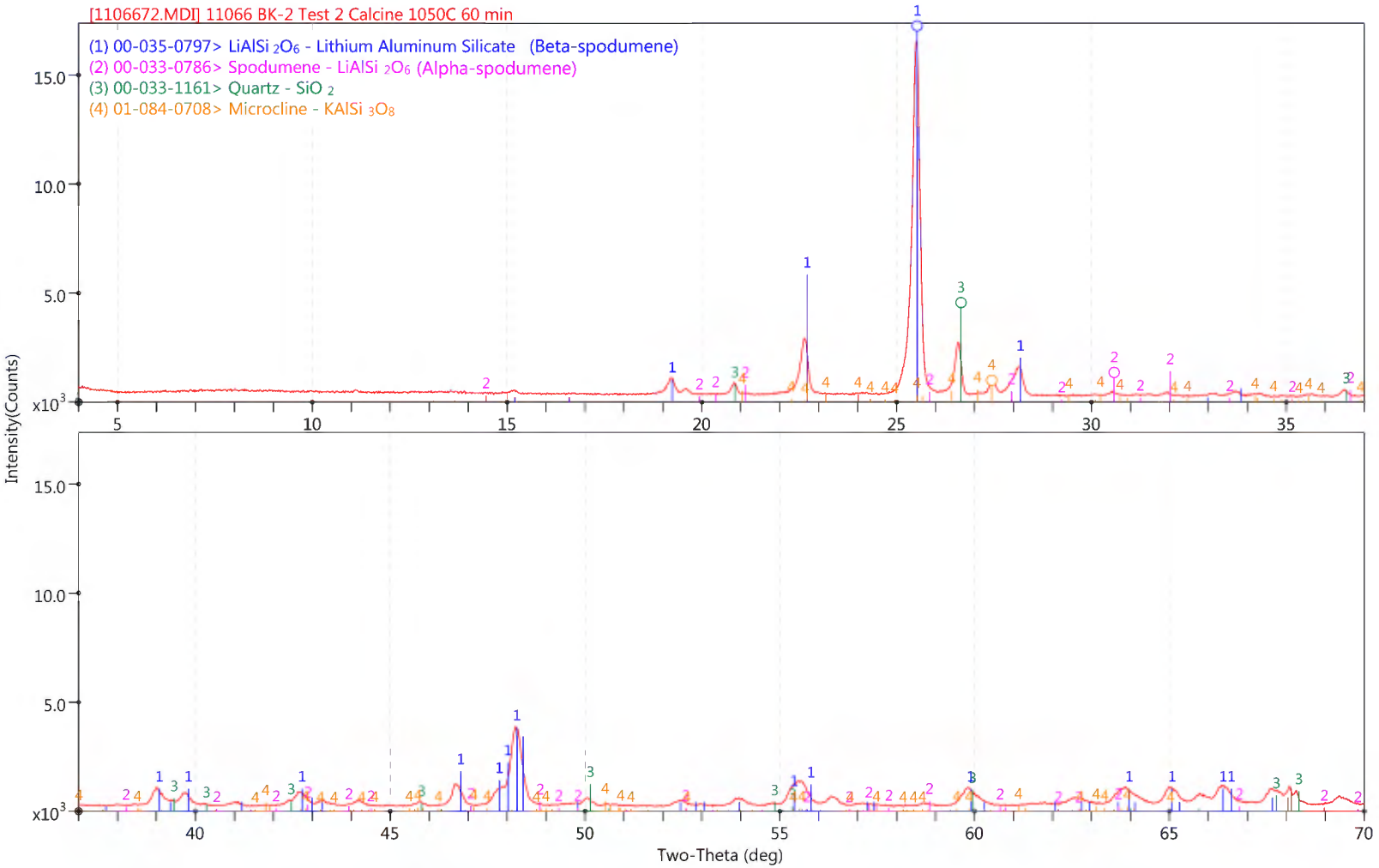


Figure G3. XRD Pattern, Test 2 Calcine, 1,050°C, 60 min

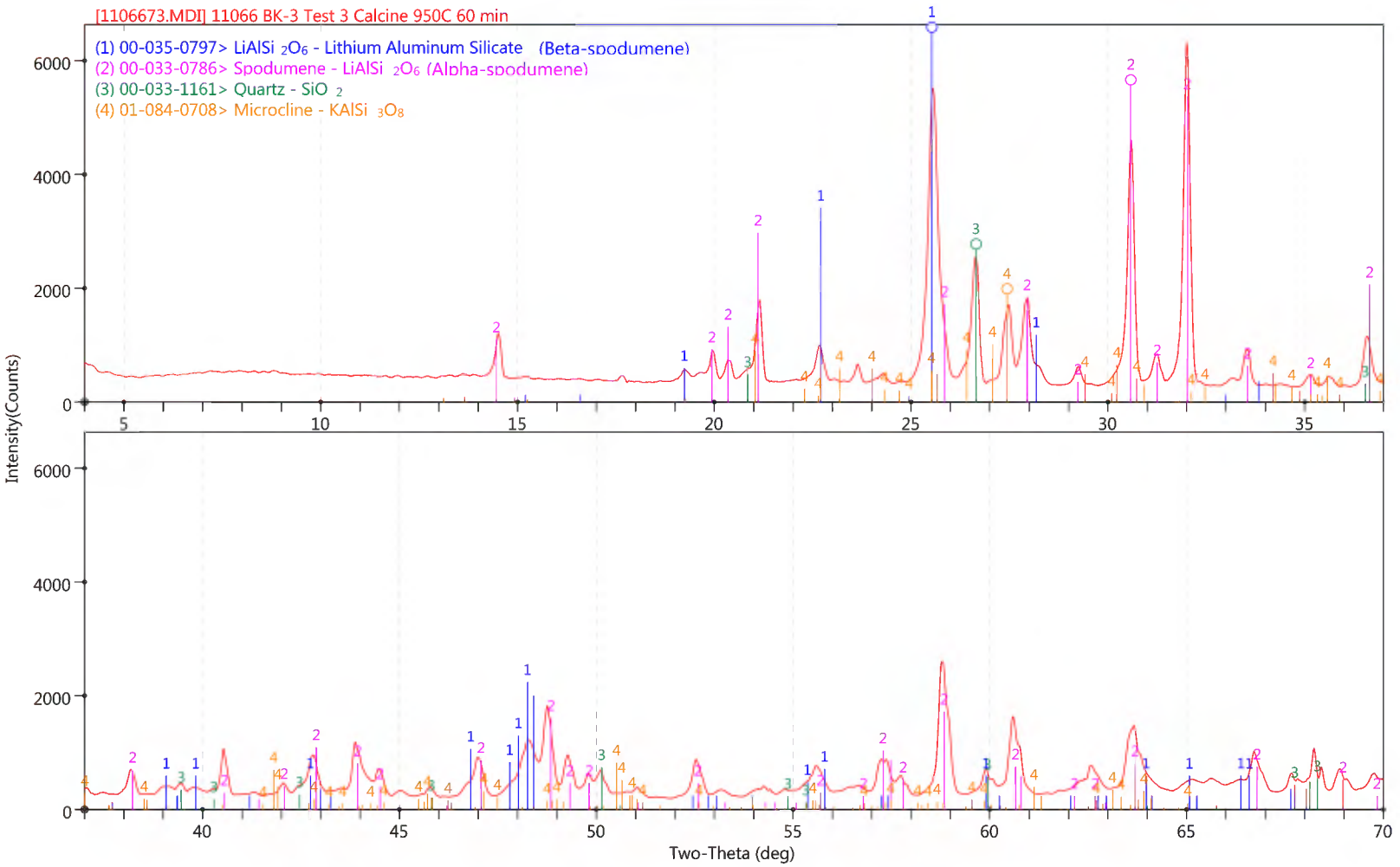


Figure G4. XRD Pattern, Test 3 Calcine, 950°C, 60 min

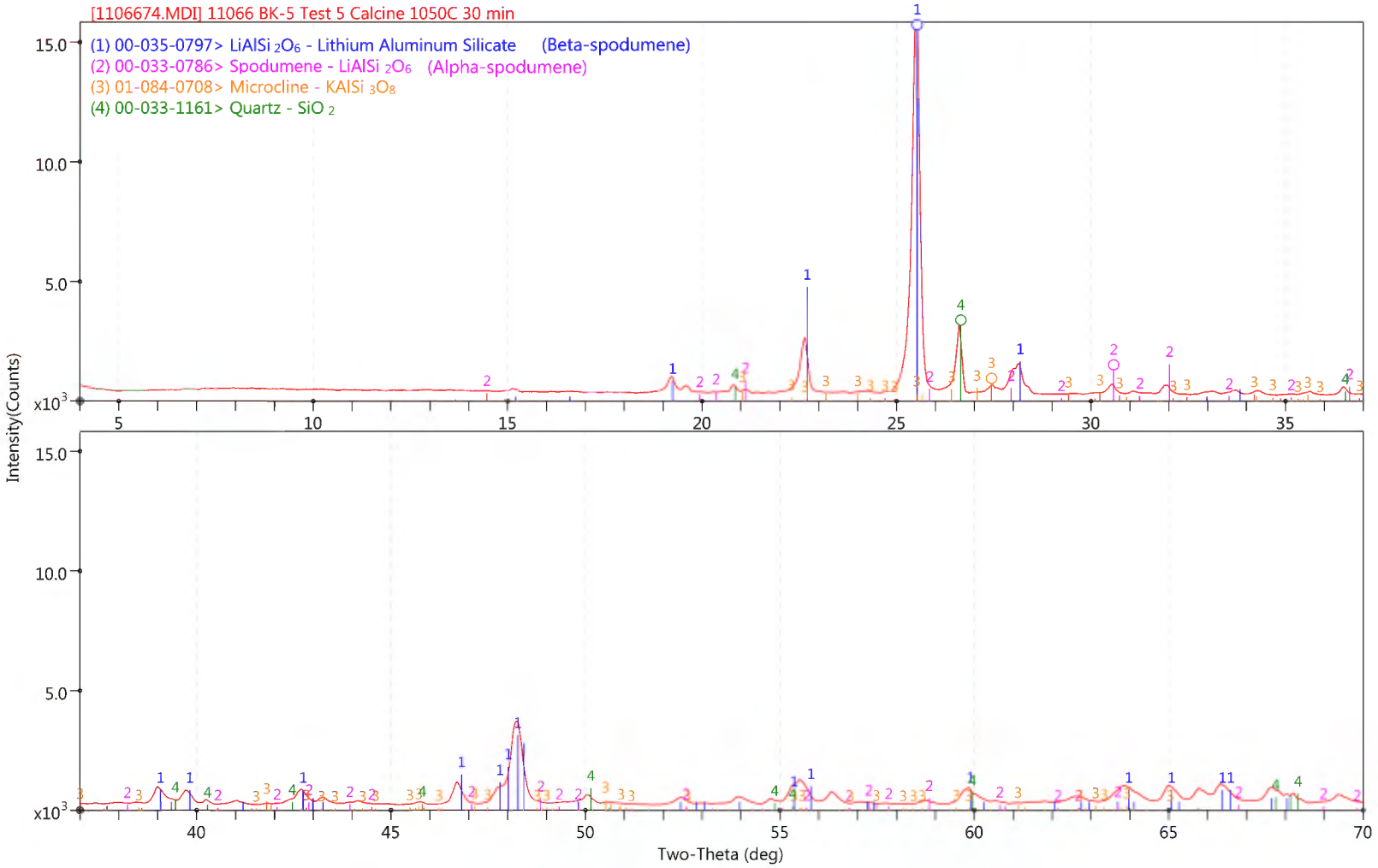


Figure G5. XRD Pattern, Test 5 Calcine, 1,050°C, 30 min

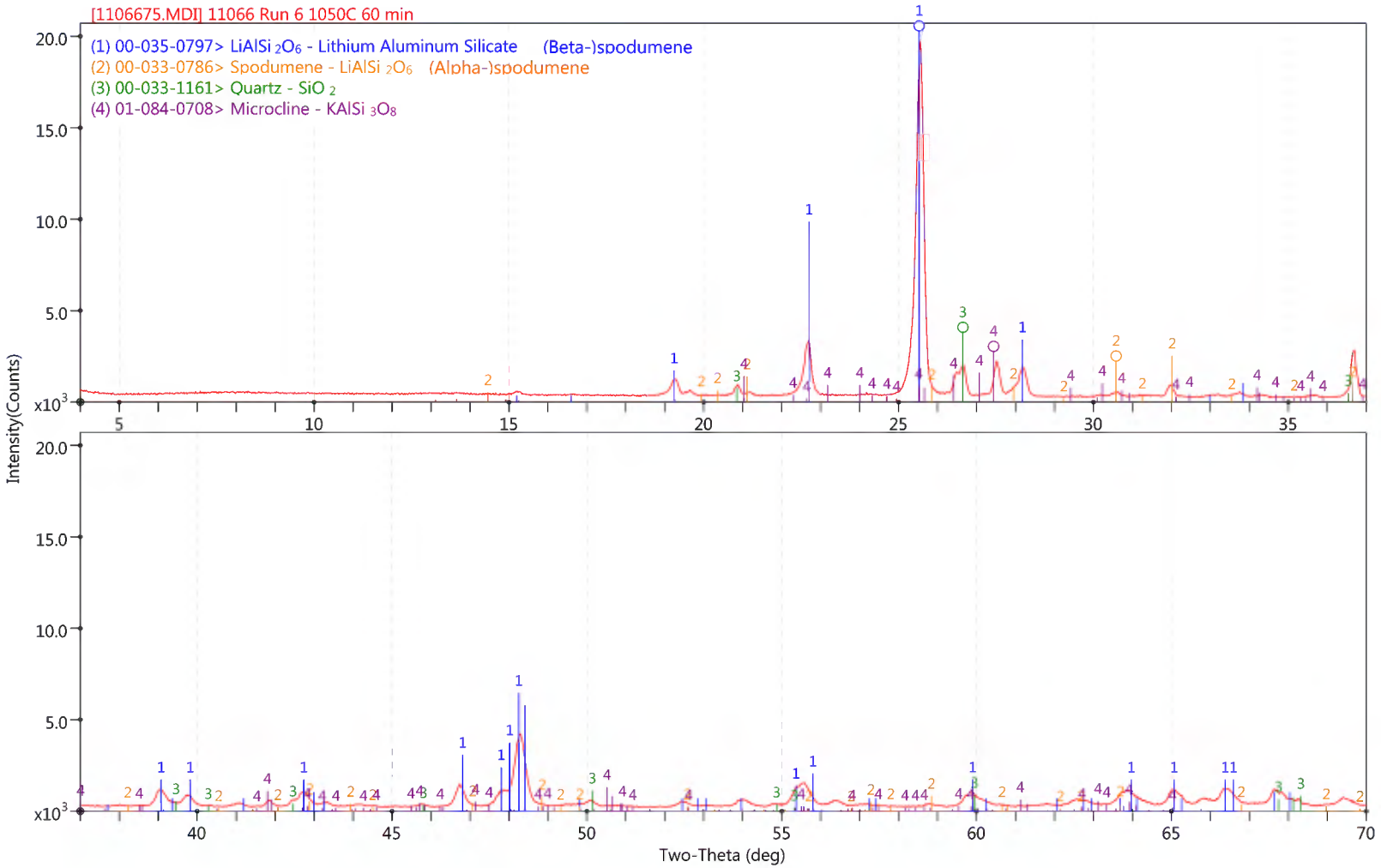


Figure G6. XRD Pattern, Test 6, 1,050°C, 60 min

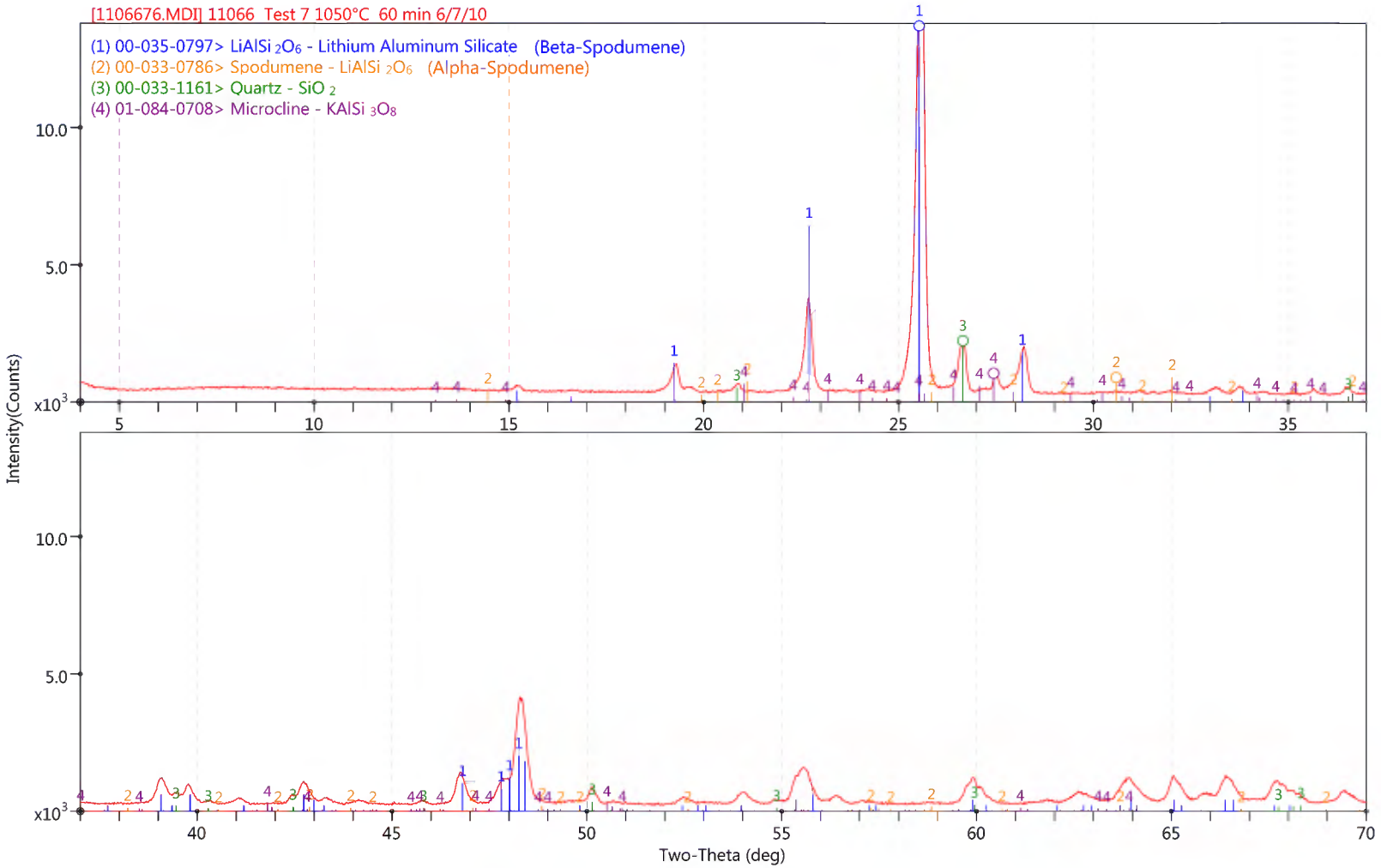


Figure G7. XRD Pattern, Test 7, 1,050°C, 60 min

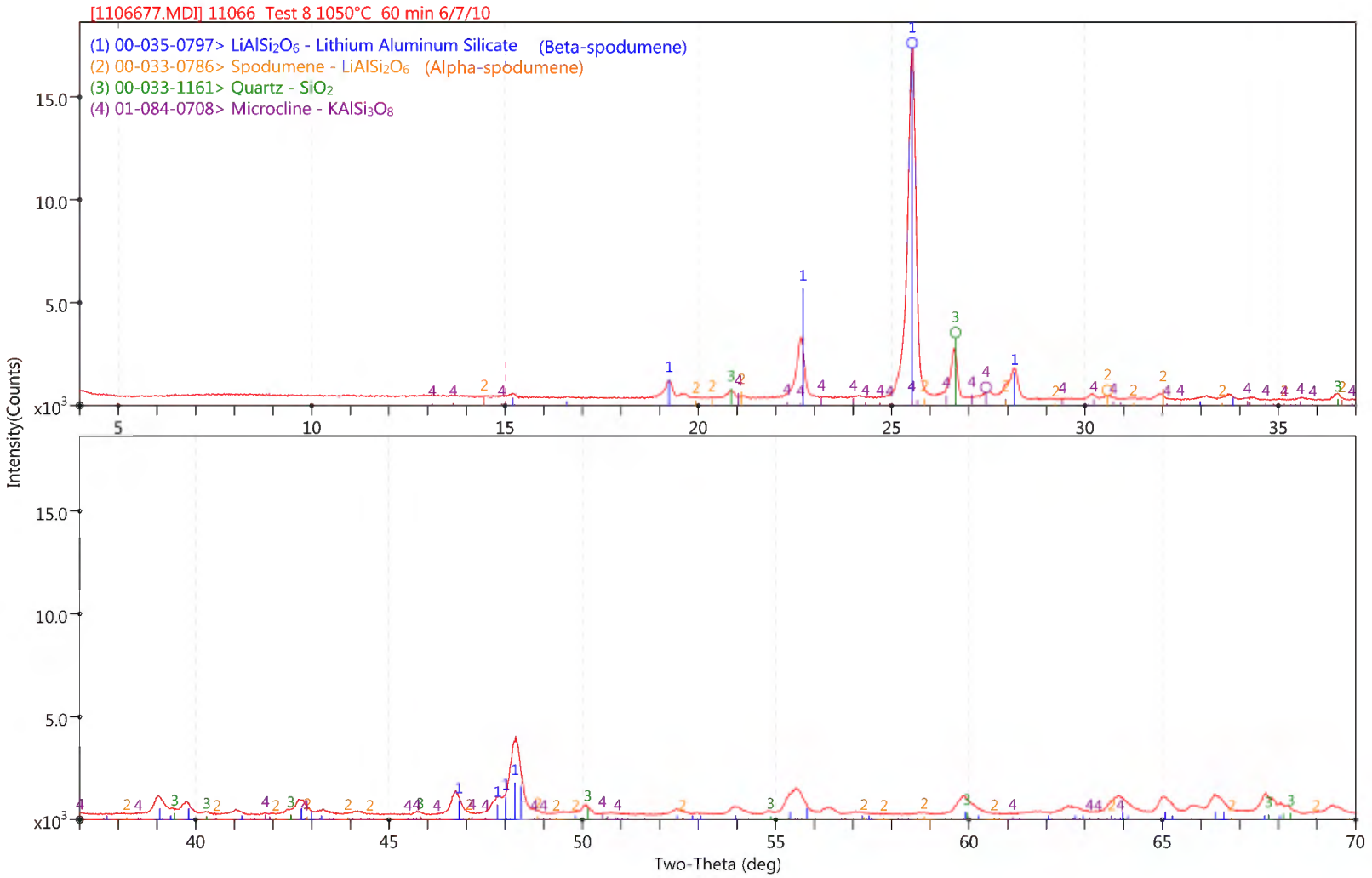


Figure G8. XRD Pattern, Test 8, 1,050°C, 60 min

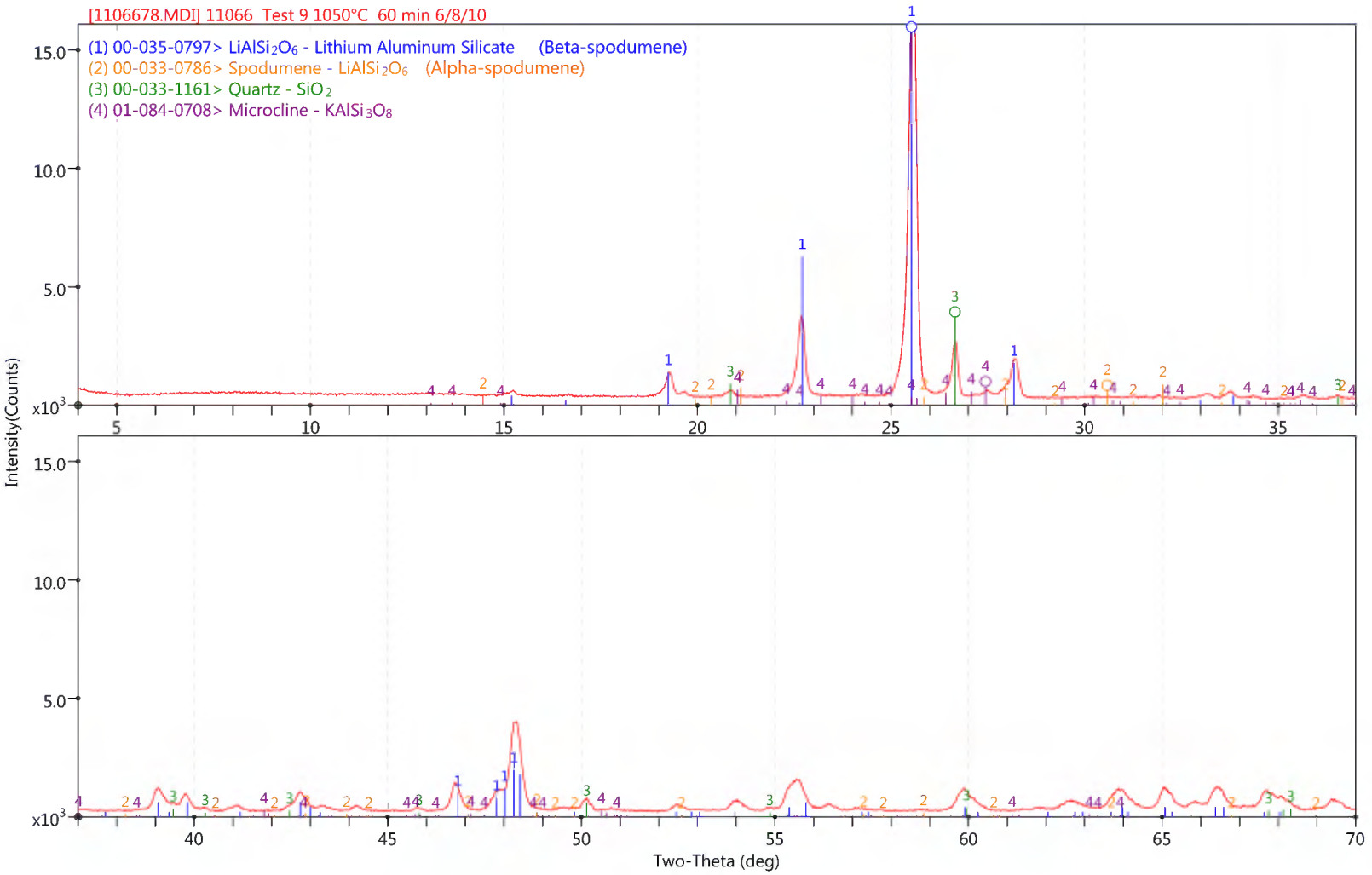


Figure G9. XRD Pattern, Test 9, 1,050°C, 60 min

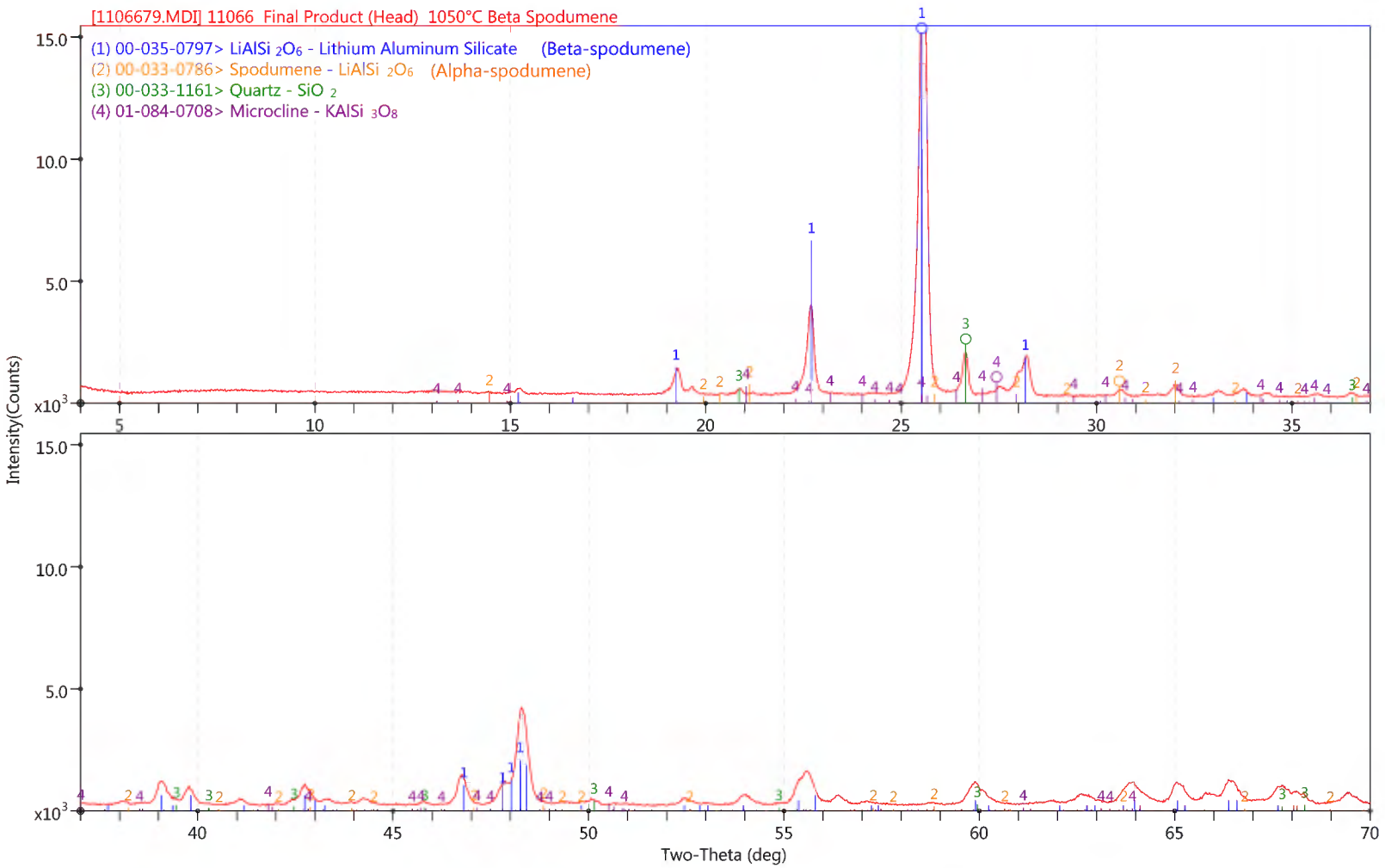


Figure G10. XRD Pattern, Final Product, 1,050°C

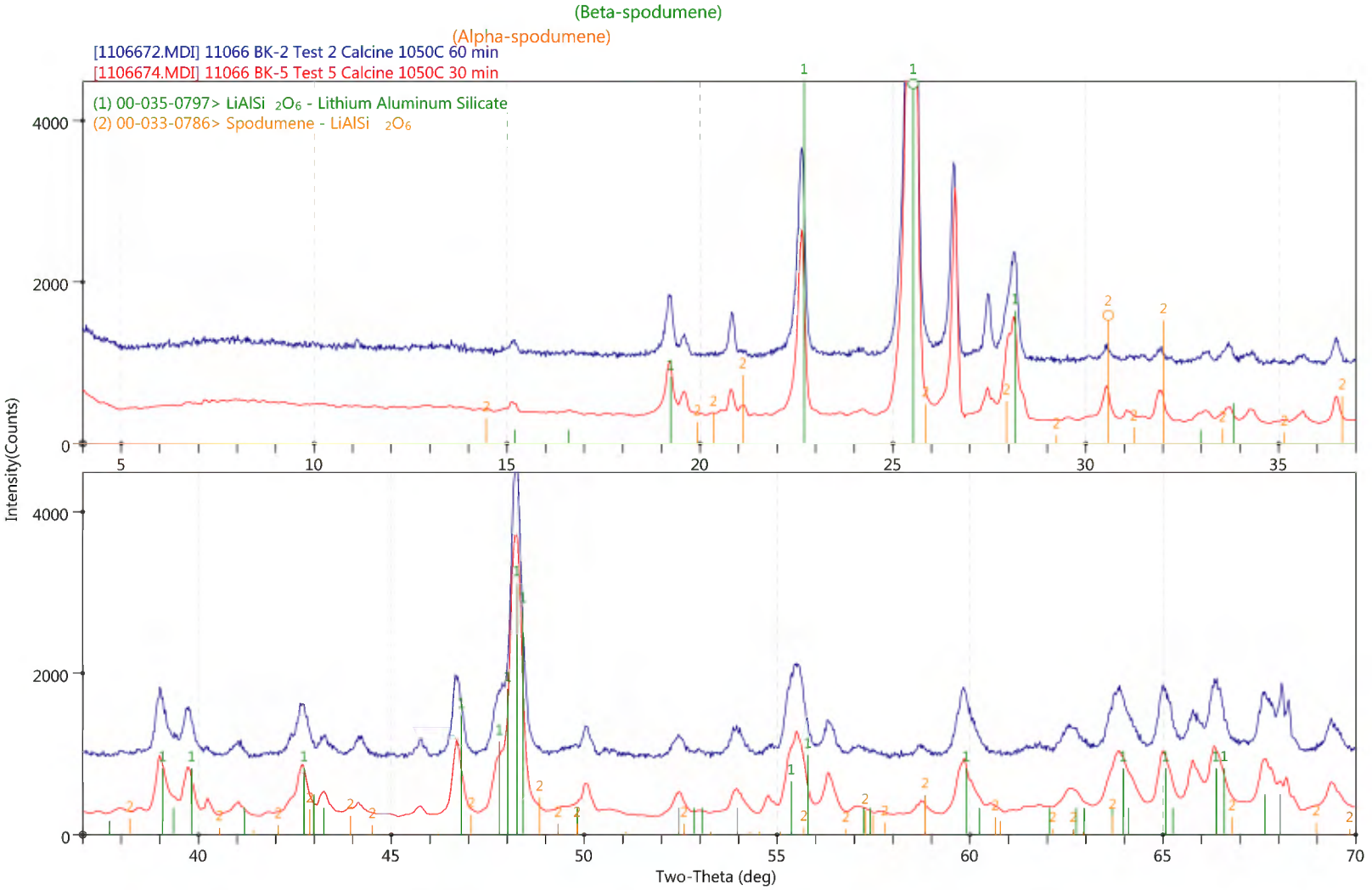


Figure G11. Overlay of XRD Patterns, Tests 2 and 5
(1,050°C, Calcines at 30 and 60 min) (showing only Alpha and Beta Spodumene)

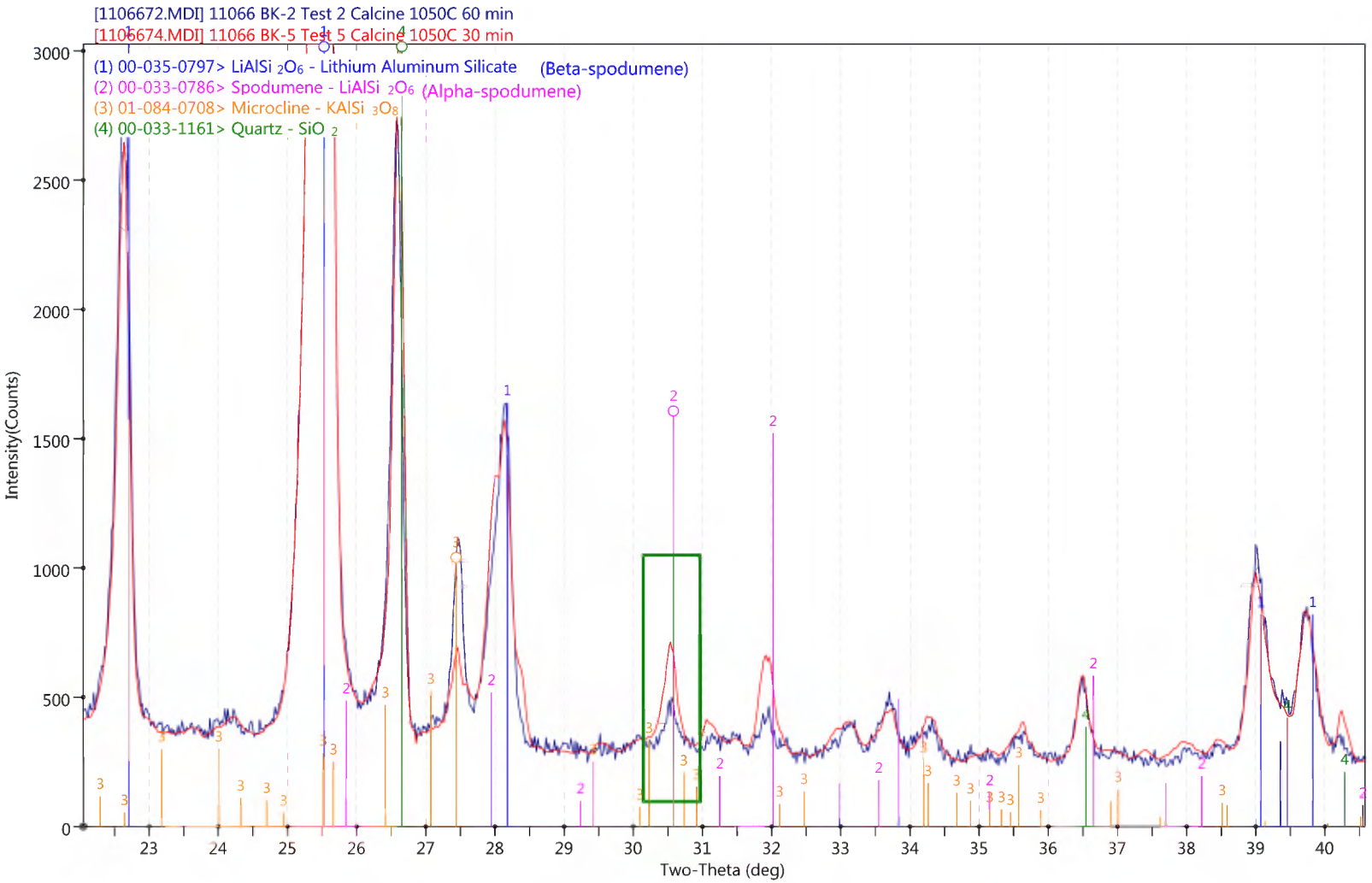


Figure G12. Expanded Area from Figure 12 (enclosed area shows main alpha-spodumene peak)

APPENDIX H

DTA Thermograms of Calcines

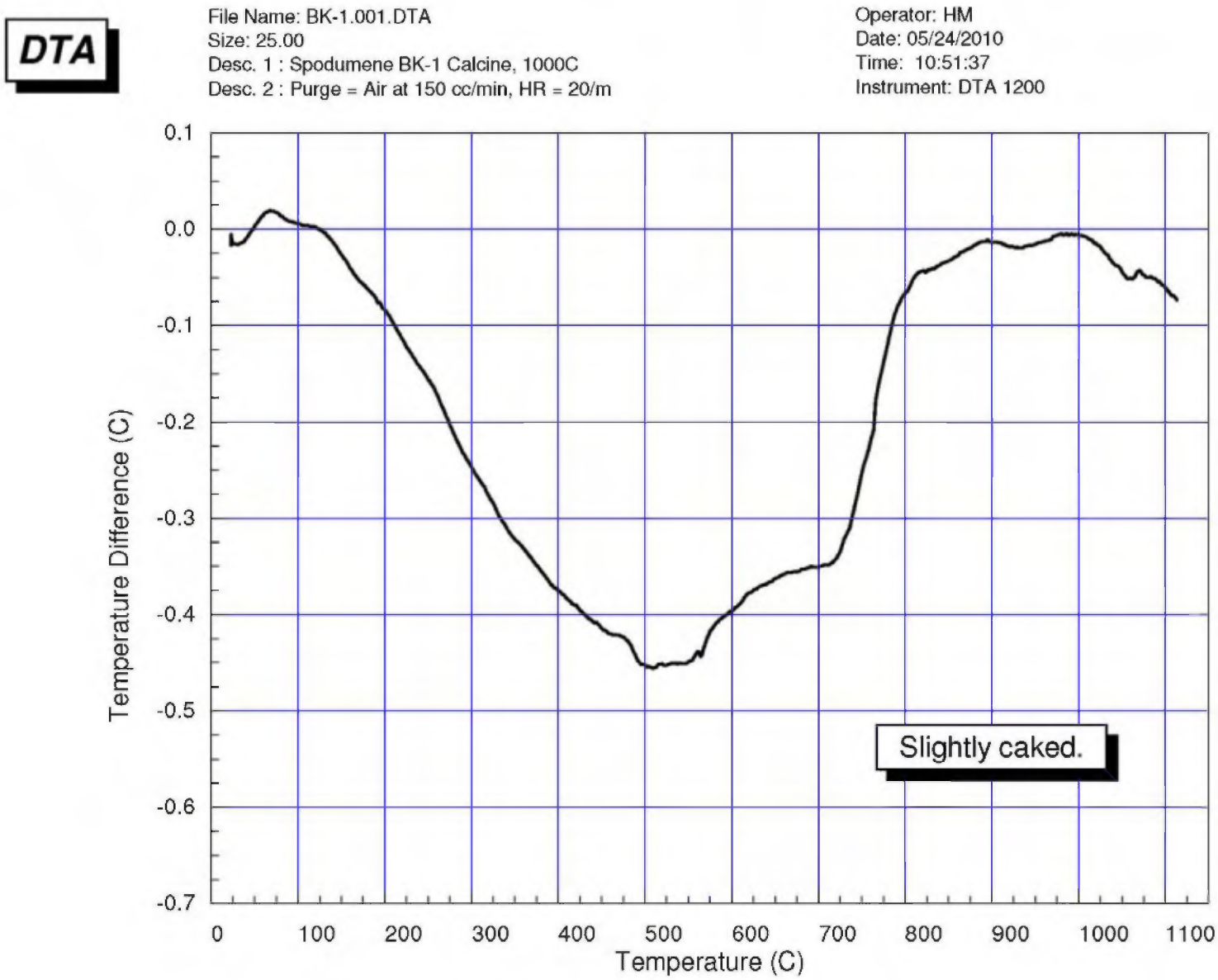


Figure H1. DTA Thermogram of Experiment 1 Calcine, 1,000°C

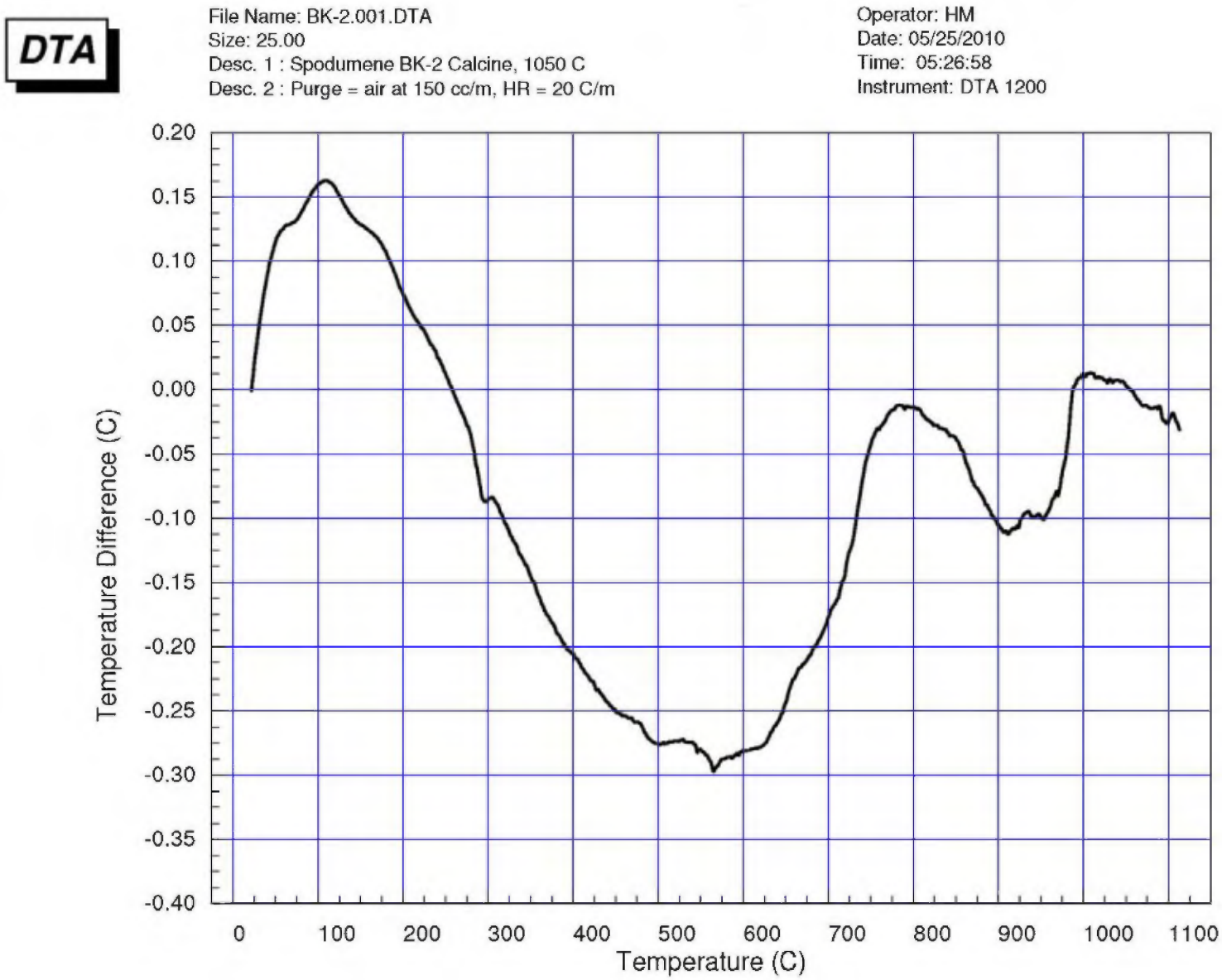


Figure H2. DTA Thermogram of Experiment 2 Calcine, 1,050°C, Example 1

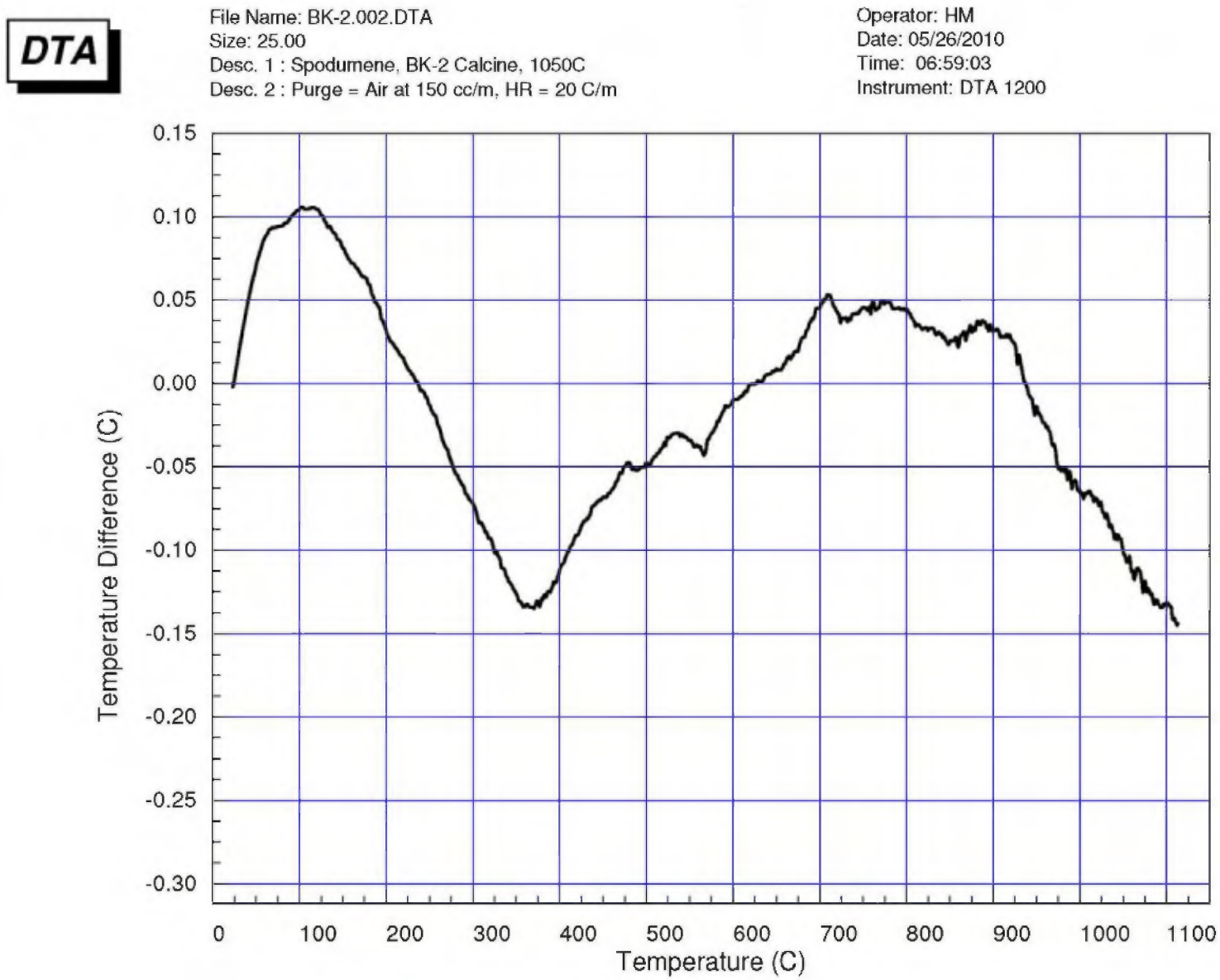


Figure H3. DTA Thermogram of Experiment 2 Calcine, 1,050°C, Example 2

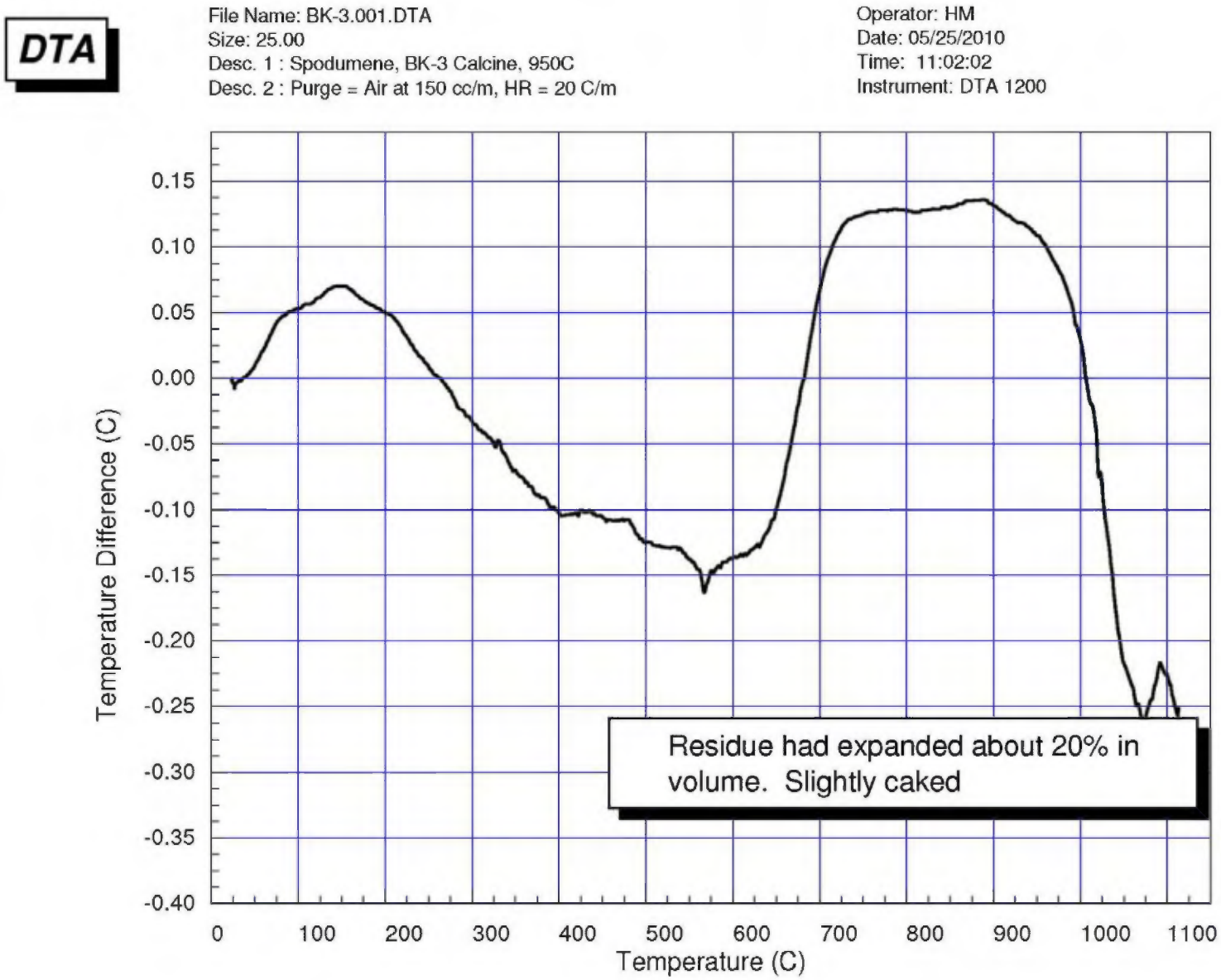


Figure H4. DTA Thermogram of Experiment 3 Calcine, 950°C

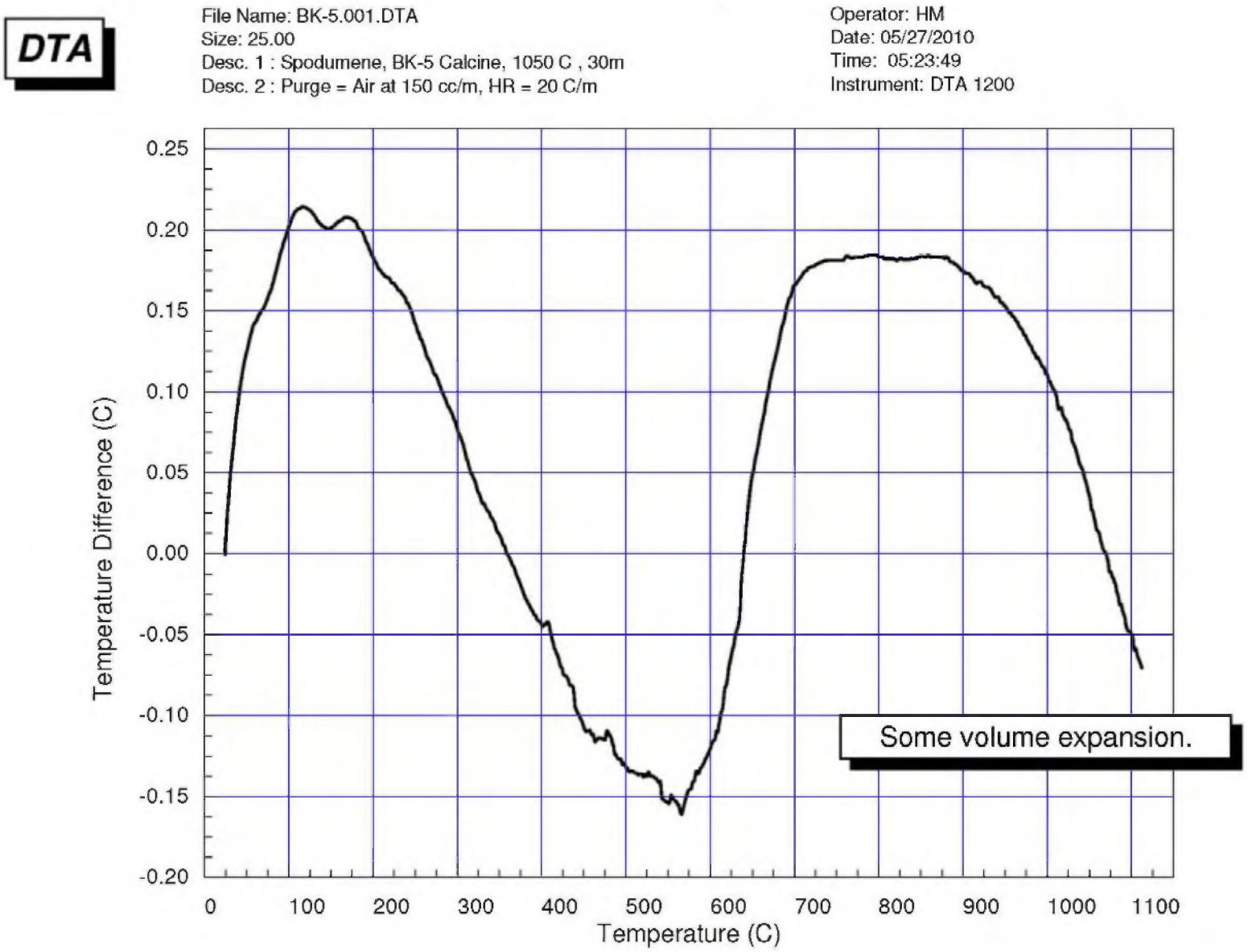


Figure H5. DTA Thermogram of Experiment 5 Calcine, 1,050°C, 30 min

APPENDIX I

Leaching and Purification Data Sheets

Lithium Extraction from β-Spodumene Acid Bake - Experiment 1

Project 11066

Book 3277-71, 72 Operator W Steward Date 6/21/2010

Purpose Extract lithium from 1050°C calcined β-spodumene after an acid bake. P_{80} = 90 mm.

Procedure 1) Spread sample in shallow dish and sprinkle concentrated H_2SO_4 over mass. Mix acid with concentrate. Make dose equivalent to 400 kg/t H_2SO_4.

2) Heat, covered, in muffle furnace in hood at 250°C for 1 h.

3) Cool and break up solids. Obtain weight loss.

4) Water-leach at 10% solids while adjusting pH to 6.0–6.5 using 10% CaO slurry. Leach 30 min at 60°C.

5) Filter and wash with warm DI water.

6) Assay solids and liquids by atomic absorption.

Leach Operating Data

Time, min	Cumulative Lime Slurry, g	pH
0	0	1.71
10	58.37	6.45
20	59.17	6.31
30	59.59	6.27

Weights, Volumes, and Assays

Sample Number	Sample ID	Volume, mL	Weight, g		Analysis, g/L or %
			Wet	Dry	Li
3277-71-0	1050°C β-spodumene calcine, feed			50.00	2.90
	H_2SO_4, 96%		20.83		
	Post-bake sample			69.55	
3277-72-1	Filtrate	600.0	602.6		1.89
3277-72-2	Wash	295.6	295.6		0.426
3277-72-3	Leach residue			62.84	0.323

Metal Distributions and Mass Balances

Sample Number	Sample ID	Volume, mL	Weight, g		Contained Weight, g
			Wet	Dry	Li
3277-71-0	1050°C β-spodumene calcine, feed			50.00	1.45
3277-72-1	Filtrate	600.0	602.6		1.13
3277-72-2	Wash	295.6	295.6		0.126
3277-72-3	Leach residue			62.84	0.203

Results

			Li
H_2SO_4, addition (100% basis), kg/t	434	Extraction (solids basis), %	86
Weight gain, %	25.7	Balance (in vs out), %	99

Lithium Extraction from β-Spodumene Acid Bake- Experiment 2

Project 11066

Book 3277-71, 72 Operator W Steward Date 6/21/2010

Purpose Extract lithium from 10-min grind calcine sample after an acid bake. P_{80} = 74 mm

Procedure 1) Spread sample in shallow dish and sprinkle concentrated H_2SO_4 over mass. Mix acid with concentrate. Make dose equivalent to 400 kg/t H_2SO_4.

2) Heat, covered, in muffle furnace in hood at 250°C for 1 h.

3) Cool and break up solids. Obtain weight loss.

4) Water-leach at 10% solids while adjusting pH to 6.0–6.5 using 10% CaO slurry. Leach 30 min at 60°C.

5) Filter and wash with warm DI water.

6) Assay solids and liquids by atomic absorption.

Leach Operating Data

Time, min	Cumulative Lime Slurry, g	pH
0	0	1.61
10	73.76	6.4
20	74.47	6.36
30	74.84	6.03

Weights, Volumes, and Assays

Sample Number	Sample ID	Volume, mL	Weight, g		Analysis, g/L or %
			Wet	Dry	Li
3277-72-0	10-min grind			54.82	2.68
	H_2SO_4, 96%		22.84		
	Post-bake sample			77.25	
3277-72-4	Filtrate	650.0	656.5		1.77
3277-72-5	Wash	323.1	323.1		0.514
3277-72-6	Leach residue			71.82	0.333

Metal Distributions and Mass Balances

Sample Number	Sample ID	Volume, mL	Weight, g		Contained Weight, g
			Wet	Dry	Li
3277-72-0	10-min grind			54.82	1.47
3277-72-4	Filtrate	650.0	656.5		1.15
3277-72-5	Wash	323.1	323.1		0.166
3277-72-6	Leach residue			71.82	0.239

Results

			Li
H_2SO_4, addition (100% basis), kg/t	434	Extraction (solids basis), %	84
Weight gain, %	31.0	Balance (in vs out), %	94

Lithium Extraction from β-Spodumene Acid Bake - Experiment 3

Project 11066

Book 3277-72, 73 Operator W Steward Date 6/21/2010

Purpose Extract lithium from 2-h grind calcine sample after an acid bake. P_{80} = 28 mm. Examining grind size.

Procedure 1) Spread sample in shallow dish and sprinkle concentrated H_2SO_4 over mass. Mix acid with concentrate. Make dose equivalent to 400 kg/t H_2SO_4.

2) Heat, covered, in muffle furnace in hood at 250°C for 1 h.

3) Cool and break up solids. Obtain weight loss.

4) Water-leach at 10% solids while adjusting pH to 6.0–6.5 using 10% CaO slurry. Leach 30 min at 60°C.

5) Filter and wash with warm DI water.

6) Assay solids and liquids by atomic absorption.

Leach Operating Data

Time, min	Cumulative Lime Slurry, g	pH
0	0	1.62
10	63.63	6.03
20	64.44	6.15
30	64.91	6.16

Weights, Volumes, and Assays

Sample Number	Sample ID	Volume, mL	Weight, g		Analysis, g/L or %
			Wet	Dry	Li
3277-72-7	2-h grind (P_{80} = 28 µm)			50.00	2.84
	H_2SO_4, 96%		20.83		
	Post-bake sample			70.23	
3277-73-1	Filtrate	595.0	599.8		1.85
3277-73-2	Wash	310.9	310.9		0.483
3277-73-3	Leach residue			64.61	0.290

Metal Distributions and Mass Balances

Sample Number	Sample ID	Volume, mL	Weight, g		Contained Weight, g
			Wet	Dry	Li
3277-72-7	2-h grind (P_{80} = 28 µm)			50.00	1.42
3277-73-1	Filtrate	595.0	599.8		1.10
3277-73-2	Wash	310.9	310.9		0.150
3277-73-3	Leach residue			64.61	0.187

Results

			Li
H_2SO_4, addition (100% basis), kg/t	434	Extraction (solids basis), %	87
Weight gain, %	29.2	Balance (in vs out), %	99

Lithium Extraction from β-Spodumene Acid Bake - Experiment 4

Project 11066

Book 3277-73, 74 Operator W Steward Date 6/29/2010

Purpose Extract lithium from 2-h grind calcine sample after mixing 10 min and an acid bake. P_{80} = 28 mm. Examining more intimate mixing.

Procedure 1) Spread sample in shallow dish and sprinkle concentrated H_2SO_4 over mass. Mix acid for 10 min with concentrate. Make dose equivalent to 400 kg/t H_2SO_4.

2) Heat, covered, in muffle furnace in hood at 250°C for 1 h.

3) Cool and break up solids. Obtain weight loss.

4) Water-leach at 10% solids while adjusting pH to 6.0–6.5 using 10% CaO slurry. Leach 30 min at 60°C.

5) Filter and wash with warm DI water.

6) Assay solids and liquids by atomic absorption.

Leach Operating Data

Time, min	Cumulative Lime Slurry, g	pH
0	0	1.32
10	71.22	6.18
20	71.22	6.29
30	71.22	6.30

Weights, Volumes, and Assays

Sample Number	Sample ID	Volume, mL	Weight, g		Analysis, g/L or %
			Wet	Dry	Li
3277-72-7	2-h grind (P_{80} = 28 µm)			50.00	2.84
	H_2SO_4, 96%		20.83		
	Post-bake sample			70.34	
3277-73-6	Filtrate	564.0	570.2		1.89
3277-73-7	Wash	290.5	290.5		0.781
3277-73-8	Leach residue			64.9	0.181

Metal Distributions and Mass Balances

Sample Number	Sample ID	Volume, mL	Weight, g		Contained Weight, g
			Wet	Dry	Li
3277-72-7	2-h grind (P_{80} = 28 µm)			50.00	1.42
3277-73-6	Filtrate	564.0	570.2		1.07
3277-73-7	Wash	290.5	290.5		0.227
3277-73-8	Leach residue			64.9	0.117

Results

			Li
H_2SO_4, addition (100% basis), kg/t	434	Extraction (solids basis), %	92
Weight gain, %	29.8	Balance (in vs out), %	101

Lithium Extraction from β-Spodumene Acid Bake - Experiment 5

Project 11066

Book 3277-73, 74 Operator W Steward Date 6/29/2010

Purpose Extract lithium from 2-h grind sample after an acid bake using 10% more acid than before. P_{80} = 28 µm.
Examine grind size and additional acid with intimate mixing.

Procedure 1) Spread sample in shallow dish and sprinkle concentrated H_2SO_4 over mass. Mix acid for 10 min with concentrate. Make dose equivalent to 440 kg/t H_2SO_4.

2) Heat, covered, in muffle furnace in hood at 250°C for 1 h.

3) Cool and break up solids. Obtain weight loss.

4) Water-leach at 10% solids while adjusting pH to 6.0–6.5 using 10% CaO slurry. Leach 30 min at 60°C.

5) Filter and wash with warm DI water.

6) Assay solids and liquids by atomic absorption.

Leach Operating Data

Time, min	Cumulative Lime Slurry, g	pH
0	0	1.14
10	74.31	2.86
20	87.81	6.43
30	88.64	6.05

Weights, Volumes, and Assays

Sample Number	Sample ID	Volume, mL	Weight, g		Analysis, g/L or %
			Wet	Dry	Li
3277-72-7	2-h grind (P_{80} = 28 µm)			50.00	2.84
	H_2SO_4, 96%		22.92		
	Post-bake sample			72.28	
3277-73-6	Filtrate	615.0	619.7		1.82
3277-73-7	Wash	280.3	280.3		0.664
3277-73-8	Leach residue			68.8	0.155

Metal Distributions and Mass Balances

Sample Number	Sample ID	Volume, mL	Weight, g		Contained Weight, g
			Wet	Dry	Li
3277-72-7	2-h grind (P_{80} = 28 µm)			50.00	1.42
3277-73-6	Filtrate	615.0	619.7		1.12
3277-73-7	Wash	280.3	280.3		0.186
3277-73-8	Leach residue			68.8	0.107

Results

			Li
H_2SO_4, addition (100% basis), kg/t	478	Extraction (solids basis), %	92
Weight gain, %	37.6	Balance (in vs out), %	101

Lithium Extraction from β-Spodumene Acid Bake - Experiment 6

Project 11066

Book 3277-74 Operator W Steward Date 6/30/2010

Purpose Extract lithium from 1050°C calcined β-spodumene after an acid bake using 10% more acid and longer mixing than before. Examine as-received calcine; P_{80} = 90 mm.

Procedure 1) Spread sample in shallow dish and sprinkle concentrated H_2SO_4 over mass. Mix acid for 10 min with concentrate.

2) Heat, covered, in muffle furnace in hood at 250°C for 1 h.

3) Cool and break up solids. Obtain weight loss.

4) Water-leach at 10% solids while adjusting pH to 6.0–6.5 using 10% CaO slurry. Leach 30 min at 60°C.

5) Filter and wash with warm DI water.

6) Assay solids and liquids by atomic absorption.

Leach Operating Data

Time, min	Cumulative Lime Slurry, g	pH
0	0	1.36
10	69.31	6.43
20	70.14	6.34
30	70.51	6.07

Weights, Volumes, and Assays

Sample Number	Sample ID	Volume, mL	Weight, g		Analysis, g/L or %
			Wet	Dry	Li
3277-71-0	1050°C β-spodumene calcine, feed			50.00	2.90
	H_2SO_4, 96%		22.92		
	Post-bake sample			72.29	
3277-74-4	Filtrate	525.6	530.6		1.84
3277-74-5	Wash	373.3	373.3		0.855
3277-74-6	Leach residue			67.46	0.193

Metal Distributions and Mass Balances

Sample Number	Sample ID	Volume, mL	Weight, g		Contained Weight, g
			Wet	Dry	Li
3277-71-0	1050°C β-spodumene calcine, feed			50.00	1.45
3277-74-4	Filtrate	525.6	530.6		0.97
3277-74-5	Wash	373.3	373.3		0.319
3277-74-6	Leach residue			67.46	0.130

Results

			Li
H_2SO_4, addition (100% basis), kg/t	478	Extraction (solids basis), %	91
Weight gain, %	34.9	Balance (in vs out), %	102

Lithium Extraction from β-Spodumene Acid Bake - Experiment 7

Project 11066

Book 3277-74 Operator W Steward Date 7/9/2010

Purpose Extract lithium from 1050°C calcined β-spodumene after an acid bake using 10% more acid and longer mixing than before. Examine as-received calcine; P_{80} = 90 mm. Repeat of Experiment 6 with no CaO addition.

Procedure 1) Spread sample in shallow dish and sprinkle concentrated H_2SO_4 over mass. Mix acid for 10 min with concentrate.

2) Heat, covered, in muffle furnace in hood at 250°C for 1 h.

3) Cool and break up solids. Obtain weight loss.

4) Water-leach approximately half the sample at 10% solids for 30 min at 60°C.

5) Repeat the leach on the other half of the sample at room temperature.

6) Filter each and wash with warm DI water.

7) Assay solids and liquids by atomic absorption.

Weights, Volumes, and Assays

Sample Number	Sample ID	Volume, mL	Weight, g		Analysis, g/L or %
			Wet	Dry	Li
3277-71-0	1050°C β-spodumene calcine, feed			50.00	2.90
	H_2SO_4, 96%		22.92		
	Post-bake sample			72.50	
	60°C Leach				
	Post-bake sample			35.95	
3277-74-10	Filtrate	305.5	311.14		2.07
3277-74-11	Wash	182.0	182.80		0.200
3277-74-12	Leach residue			23.08	0.255
	Room temperature leach				
	Post-bake sample			36.20	
3277-74-7	Filtrate	315.0	318.2		2.00
3277-74-8	Wash	172.0	172.4		0.163
3277-74-9	Leach residue			23.54	0.267

Metal Distributions and Mass Balances

Sample Number	Sample ID	Volume, mL	Weight, g		Contained Weight, g
			Wet	Dry	Li
	60°C leach				
3277-71-0	1050°C β-spodumene calcine, feed			24.91	0.72
3277-74-10	Filtrate	305.5	311.14		0.63
3277-74-11	Wash	182	182.8		0.036
3277-74-12	Leach residue			23.08	0.059
	Room temperature leach				
3277-71-0	1050°C β-spodumene calcine, feed			25.09	0.73
3277-74-7	Filtrate	315.0	318.2		0.63
3277-74-8	Wash	172	172.4		0.028
3277-74-9	Leach residue			23.54	0.063

Results

H_2SO_4, addition (100% basis), kg/t	478		**Li**
60°C leach		Extraction (solids basis), %	92
	Weight gain, % -7.4	Balance (in vs out), %	101
Room temperature leach		Extraction (solids basis), %	91
	Weight gain, % -6.2	Balance (in vs out), %	99

Preparation of Li_2CO_3

Project 11066

Book 3277-79–81 Operator W Steward Date 7/27/2010

Procedure

1) Spread sample in shallow dish and sprinkle concentrated H_2SO_4 over mass. Mix quickly, for 10 min total.
2) Heat covered in muffle furnace in hood at 250°C for 1 h.
3) Cool and break up solids. Obtain weight loss.
4) Water-leach at 10% solids while adjusting pH to 6.0–6.5 using 10% $Ca(OH)_2$. Leach 30 min at 60°C.
5) Filter and wash with warm DI water.
6) Boil down filtrate to ~$^1/_2$ volume.
7) Raise pH of filtrate to 11.5 and hold with mixing at room temperature.
8) Filter on Buchner funnel and do small washes with lime water. Boil primary filtrate (PF) down by ~$^1/_2$.
9) Precipitate $CaCO_3$ in filtrate by adding 2.26 g Na_2CO_3 dissolved in 10 mL water over 10 min with stirring.
10) Filter and do small washes with 5 g/L Na_2CO_3.
11) Precipitate Li by adding 28.31 Na_2CO_3 dissolved in 90.53 g water. Heat to 90–93°C and hold for 15 min.
12) Centrifuge while hot and rinse Li_2CO_3 with 1.98 mL hot DI water while spinning.

Data

Bake

Description	Sample Start Wt, g	96% H_2SO_4, g	End Wt, g
1050°C Beta Spodumene Calcine, Feed	200.0	91.68	290.26

Leach

Time, min	pH	Total 10% $Ca(OH)_2$, g
0	1.56	0
10	2.28	256.1
20	6.49	391.79
30	6.22	394.34

Leach Assays

Description	ID	Wt, g	Vol, mL	Analysis, wt% or g/L	
				Li	Mg
Feed (before bake)	3277-71-0	200.00	-	2.90	-
PF	3277-80-1	2467.3	2440.0	-	-
PF after boiled down	3277-80-4	-	1119.0	3.61	0.0934
Wash	3277-80-2	789.9	789.0	0.955	0.0332
Leach residue	3277-80-3	495.41 wet 280.45 dry	-	0.344	-

Preparation of Li_2CO_3

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Raise pH to 11.5

Time, min	pH	Total 20% $Ca(OH)_2$, g
0	8.44	0
0	11.2	2.94
10	11.22	3.39
20	11.5	4.34
35	11.5	4.53

pH 11.5 Step Assays

Description	ID	Wt, g	Vol, mL	Analysis, wt% or g/L		
				Li	Ca	Mg
PF	3277-80-5	1103.10	1075	3.65	0.728	-
Wash	3277-80-7	35.15	34.7	1.68	0.486	-
Residue	3277-80-6	3.01	-	0.041	19.6	3.83

Evaporation

Start Volume, mL	End Volume, mL
1072	463

Precipitate $CaCO_3$

Description	ID	Wt, g	Vol, mL	Analysis, wt% or g/L	
				Li	Ca
PF	3277-81-1	491.2	467.5	8.00	-
Wash	3277-81-2	24.05	24.0	0.973	-
Residue	3277-81-3	2.65 wet 1.36 dry	-	0.782	32.5

Li Precipitation

Description	ID	Wt, g	Vol, mL	Analysis, wt% or g/L			
				Li	Mg	Na	K
PF	3277-81-5	520.2	485.0	2.30	<0.0005	26.5	0.216
Wash	3277-81-6	1.06	<1.0	-	-	-	-
Li_2CO_3 Product	3277-81-4	13.99 wet 12.39 dry	-	17.1	0.002	0.650	0.008

Li_2CO_3 Product Addition Assays

Analysis, wt%							
Ca	Fe	Al	Mn	Cu	Pb	Cr	S^{tot}
0.117	0.006	0.020	0.001	0	<0.01	0	0.43

Cl^-	PO_4^{3-}	Be	B	Rb	CO_3^{2-}
<0.01	<0.01	<0.005	<0.005	<0.001	72.1