Millcreek Mining
GROUP

Phone 801-904-2260
Fax 801-904-2261
1011 East Murray Holladay Road, Suite 100
Salt Lake City, UT 84117

Technical Report
The Blawn Mountain Project
Updated Prefeasibility Report
Beaver County, Utah

PREPARED FOR:

Potash Ridge Corporation

PREPARED BY:

Mr. Steven B. Kerr, C.P.G., Principal Consultant - Geology, Millcreek Mining Group
Mr. Jason N. Todd, QP, Principal Consultant - Geology, Millcreek Mining Group
Mr. Deepak Malhotra, QP, President of Resource Development Inc. (RDI)

Effective Date: January 10, 2017
Report Date: January 18, 2017
Project Number: 16M34

CERTIFICATE OF QUALIFICATIONS

I, Steven B. Kerr, CPG, PG of Salt Lake City, Utah, do hereby certify that:

1. I am currently employed as a Principal Consultant – Geology at Millcreek Mining Group, 1011 East Murray Holladay Road, Suite 100, Salt Lake City, Utah, USA 84117.
2. I attended Utah State University where I earned a Bachelor of Science degree in Geology in 1981 and a Master of Science degree in Geology in 1987.
3. I am a Certified Professional Geologist with the American Institute of Professional Geologists (CPG-10352). I am licensed as a Professional Geologist in the states of Alaska (#512), Utah (#5557442-2250) and Wyoming (PG-2756).
4. I have worked as a geologist for a total of thirty-two years since my graduation from university, working with companies involved in the mining and exploration of industrial minerals, metallic minerals, and solid fuel energy deposits in North America, South America, Africa, and parts of Asia.
5. I have read the definition of “qualified person” set out in National Instrument (NI) 43-101 and that by reason of my education, affiliation with professional associations (as defined in NI 43-101), and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
6. I am responsible for sections 2 through 12, 14, 23, and have contributed to sections 1, 24, 25, 26, 27 and 28 of the technical report titled “Technical Report Updated Prefeasibility Study of the Blawn Mountain Project, Beaver County, Utah” dated Jan. 18, 2017 (Technical Report) with an **Effective Date of January 10, 2017.**
7. I personally visited and inspected the Blawn Mountain Property on several occasions since 2012. The first visit to the property occurred on February 9, 2012 and the most recent visit to the property was on September 9, 2016.
8. I previously contributed to the preparation of three technical reports on the Blawn Mountain Project which were titled (1) “Technical Report Resources and Reserves of the Blawn Mountain Project, Beaver County, Utah” dated December 2, 2013; (2) “Technical Report Preliminary Economic Assessment, Blawn Mountain Project, Beaver County, Utah” dated November 2, 2012; and (3) “Blawn Mountain Project, Beaver County, Utah” dated April 16, 2012.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed for which I am responsible make the Technical Report not misleading. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
10. I have read NI 43-101and Form 43-101 (F1), and the Technical Report has been prepared in compliance with this instrument and form.

Dated at Salt Lake City, Utah this 18th day of January, 2017.

“ORIGINAL SIGNED AND SEALED BY AUTHOR”

Steven B. Kerr, CPG, PG

CERTIFICATE OF QUALIFICATIONS

I, Jason N. Todd, of Salt Lake City, Utah, do hereby certify that:

1. I am currently employed as a Principal Consultant – Mining at Millcreek Mining Group, 1011 East Murray Holladay Road, Suite 100, Salt Lake City, Utah, USA 84117.
2. I attended Montana Tech of the University of Montana where I earned a Bachelor of Science degree in Mining Engineering in 1998.
3. I am a Qualified Professional Member of the Mining and Metallurgical Society of America, Member #0414QP.
4. I have worked as a mining engineer for a total of eighteen years since my graduation from university for mining companies and as a consultant specializing in coal and other solid fuel energy deposits and industrial minerals in North America, South America and parts of Asia.
5. I have read the definition of "qualified person" set out in National Instrument (NI) 43-101 and that by reason of my education, affiliation with professional associations (as defined in NI 43-101), and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
6. I am responsible for sections 15, 16, 18 through 20 and 22 and have contributed to sections 1, 21, 24, 25, 26, 27 and 28 of the technical report titled "Technical Report Updated Prefeasibility Study of the Blawn Mountain Project, Beaver County, Utah" dated Jan. 18, 2017 (Technical Report) with an **Effective Date of January 10, 2017.**
7. I personally visited and inspected the Blawn Mountain Property on several occasions since 2012. The first visit to the property occurred on March 15, 2012 and the most recent visit to the property was on September 9, 2016.
8. I previously contributed to the preparation of two technical reports on the Blawn Mountain Project which were titled (1) "Technical Report Resources and Reserves of the Blawn Mountain Project, Beaver County, Utah" dated December 2, 2013; and (2) "Technical Report Preliminary Economic Assessment, Blawn Mountain Project, Beaver County, Utah" dated November 2, 2012.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed make the Technical Report not misleading. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
10. I have read NI 43-101and Form 43-101 (F1), and the Technical Report has been prepared in compliance with this instrument and form.

Dated at Salt Lake City, Utah this 18th day of January, 2017.

"ORIGINAL SIGNED AND SEALED BY AUTHOR"

Jason N. Todd, QP

CERTIFICATE OF QUALIFICATIONS

I, Deepak Malhotra, of Wheat Ridge, Colorado, do hereby certify that:

1. I am currently the President of Resource Development Inc., (RDI), 11475 W I-70 Frontage Road North, Wheat Ridge, CO 80033.
2. I am a graduate of Colorado School of Mines in Colorado where I earned A Masters of Metallurgical Engineering in 1974 and a PhD in Mineral Economics in 1978.
3. I am a Registered Member in good standing of the Society of Mining, Metallurgy and Exploration Inc. (SME) (License # 2006420) and a member of Canadian Institute of Mining, Metallurgy and Petroleum (CIM).
4. I have worked as an engineer for over 43 years since my graduation from university for mining companies, financial institutions and as a consultant specializing in sulfide and industrial minerals worldwide.
5. I have read the definition of “qualified person” set out in National Instrument (NI) 43-101 and that by reason of my education, affiliation with professional associations (as defined in NI 43-101), and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
6. I am responsible for sections 13 and 17 and have contributed to sections 1, 21, 25, 26, 27 and 28 of the technical report titled “Technical Report - Updated Prefeasibility Study of the Blawn Mountain Project, Beaver County, Utah” dated Jan. 18, 2017 (Technical Report) with an Effective Date of January 10, 2017.
7. I visited the Blawn Mountain Property on September 9, 2016.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed make the Technical Report not misleading. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
9. I have read NI 43-101and Form 43-101 (F1), and the Technical Report has been prepared in compliance with this instrument and form.

Dated at Wheat Ridge, Colorado this 18th day of January, 2017.

“ORIGINAL SIGNED AND SEALED BY AUTHOR”

Deepak Malhotra, QP

TABLE OF CONTENTS

LIST OF TABLES & CHARTS

LIST OF FIGURES

1 SUMMARY

This technical report was prepared by Millcreek Mining Group (Millcreek) for Potash Ridge Corporation (PRC), a publicly traded mineral exploration and development company with its corporate office in Toronto, Ontario, Canada. This report summarizes an updated Prefeasibility Study (PFS) performed on the Blawn Mountain Project by Millcreek and Resource Development Inc. (RDI) that focuses on the mining and processing of alunite ore for the production of sulfate of potash (SOP). This report incorporates results of trade-off and scaling studies performed by SNC Lavalin (SNC) for the processing facility. For reference and clarity, this Technical Report serves as an update to the Technical Report that PRC completed in November, 2013.

The purpose of this Technical Report is to present a more realistic and accurate estimate of project value. Given the investment market's appetite and interest in similar capital projects, PRC proposes an optimum annual production rate of 3.4Mtpy (down from the previously proposed 10.4Mpty). However, the Life-of-Project has been constrained to approximately 46 years to maintain the level of confidence required to define economic reserves. While this leads to an overall decrease in economic reserves that can be reported in this study, it does not impact resources available for consideration after the Life-of-Mine (see Section 15.3).

The previous 2013 Technical Report reported mineable economic reserves of 425.8Mt. This is in contrast to estimated economic reserves of 153.3Mt presented in this study. The significant changes in these estimates are **not** attributed to a decrease in mineable resources or newly-identified adversities in mining conditions, operating costs or other economic factors (such as price). Rather, the current reserves estimate reflects an improved and fundamentally different approach to the project with respect to annual production rate. The reserves identified in this PFS extract approximately 27% of the available resources defined to date. A substantial resource will remain at Blawn Mountain should an expanded operation be given consideration in the future.

1.1 LOCATION AND TENURE

The Blawn Mountain Project consists of 15,403.72 acres (ac) of Utah State surface and mineral tracts, administered by the State of Utah School and Institutional Trust Lands Administration (SITLA). PRC has rights to the property through an Exploration/Option Agreement (ML 51983.0 OBA) and two state mineral leases (ML 52513, and ML 52364) administered by SITLA. The agreement consists of a main tract of land that covers 14,923.72ac and six individual 80ac tracts located 3.5 to 4.5 miles northeast of the main tract of land.

The property is located approximately 30 air miles southwest of the town of Milford, Utah and 30 air miles from the Nevada state border, as shown in Figure 4.1. The area is accessed from Interstate 15 (I-15), the main north-south travel corridor through Utah, by traveling west on the surfaced State Route 21 (SR-21) to the town of Milford, from Milford 24 miles farther west on SR-21, turning south onto a graveled secondary road and traveling approximately 17 miles. The property is located about 20 air miles west of the Union Pacific (UP) Railroad route, running north-south and connecting Salt Lake City with Las Vegas and farther points on the UP-rail system.

SITLA lease ML 51983.0 OBA is comprised of 17 full sections and two half sections of SITLA land and potash mineral rights. PRC's entitlement is through the Exploration/Option Agreement with SITLA which was executed in the spring of 2011 and issued to Utah Alunite, LLC, a 100% owned entity of PRC. In April 2012, Utah Alunite, LLC was merged into Utah Alunite Corporation (UAC) and the mineral lease was assigned by Utah Alunite, LLC to UAC.

UAC added five full sections adjacent to the north of ML 51983.0 OBA through the acquisition of SITLA lease ML 52513, effective June 1, 2013. The six individual 80ac parcels were acquired from SITLA under lease ML 52364, effective January 7, 2013.

Two small mineral leases occur within the Blawn Mountain Project, controlled by other parties. One lease is a 40ac tract located along the western edge of the project area and the second lease is a 155ac tract within the boundaries of the project area. Both of these mineral leases are for metallic minerals only and do not include potash mineral rights. Both leases are administered by SITLA.

PRC's Exploration/Option Agreement with SITLA was for a 3-year term and required a front-end payment of $200,000[1], annual payments of $69,300 ($6ac) and a $1,020,000 bonus for lease issuance which was due on or before March 31, 2014. Primary lease term would be 10 years, renewable in 5-year extensions. Annual rental amounts would apply to the lease agreement as well as 4% gross royalty for metalliferous minerals and 5% for potash minerals. The initial lease terms included a provision to provide a "Positive PFS" to SITLA, documenting the project's economic viability, possible markets, mining methods and potential environmental issues as well as providing evidence showing the possible continuity and grade of the ore. On March 25, 2013 PRC exercised its right under the Exploration/Option Agreement to convert this agreement into a long-term mining lease.

[1] Currency in US Dollars

In June 2015, PRC entered into a modification of the Blawn Mountain Project mining lease agreement with SITLA. The modification cured the event of default under the lease that occurred on March 31, 2015. Under the terms of the modification, SITLA has agreed to forbear from exercising its rights and remedies resulting from PRCs failure to make lease and minimum royalty payments to SITLA under the terms of the lease. The forbearance period is from March 31, 2015 to April 1, 2017. The total amount payable to SITLA, including accrued interest, on April 1, 2017 is approximately $1.2 million.

PRC was obligated to pay accrued and unpaid interest by March 31, 2016 or when it raises US$1.5 million in new funds for the development of the Blawn Mountain Project, whichever occurs first.

Once PRC raises US$3 million or more of new funds for the development of the Blawn Mountain Project then all outstanding amounts currently due under the lease, plus accrued interest, will become due.

PRC will pay interest to SITLA on unpaid lease and minimum royalties payments which will accrue annually at a rate of SITLA's published prime rate plus 2% (currently equivalent to 5.25%) or 6.0%, whichever is greater, with the first interest payment having been due on March 31, 2016. PRC made the required accrued interest payment to SITLA on March 30, 2016 and August 31, 2016 and is current on all lease obligations.

Mineral leases ML52513 and ML 52364 are standard metalliferous mineral leases, each with a 10-year primary term and option to extend beyond the primary term. SITLA mineral leases carry an annual rental rate of $1ac, a gross production royalty of 8% on fissionable minerals and 4% on non-fissionable minerals. Advance royalty payments equating to three times the annual rental rate were assessed at the time of issuance.

1.2 HISTORY

Mining operations have been conducted on alunite occurrences in southwest Utah since the early 1900s. Early extraction targeted both potash and aluminum. The Blawn Mountain Project area was the subject of extensive study and exploration activity conducted by a Denver-based exploration and development company, Earth Sciences, Inc. (ESI). However, much of this work targeted the property's aluminum potential. Exploration and geological studies were augmented by mining and processing evaluations as well. They delineated four distinct areas of alunite mineralization at Blawn Mountain, then known as the NG Alunite property. Their historic areas A through D correspond to the PRC nomenclature of Areas 1 through 4. The project was taken to advanced stages of development but was eventually abandoned due to challenging economic conditions and depressed pricing for alumina and potash in the 1980s.

Previous resource estimates are difficult to relate to current assessments primarily due to focus of past programs on alumina production with potash as a secondary product. Cut-off grades were based on alumina (Al_2O_3) content and therefore skew the potassium (K_2O) estimates since potassium was not optimized. Historic estimates ranged from 142.6 million short tons (Mt) to 151.8Mt of in-place alunite resource proven and probable (relates to measured and indicated resource), with corresponding grade estimates of K_2O ranging from 3.85% to 4.15% and of Al_2O_3 ranging from 13.03% to 14.13%. (Walker, 1972; Chapman 1974; Couzens, 1975) None of these studies are deemed to be National Instrument 43-101 (NI 43-101) compliant, although reasonable methodologies were applied at the time. Furthermore, a qualified person has not done sufficient work to classify historical estimates as current mineral resources. PRC is not treating the historical (pre-2011) estimates as current mineral resources.

In 2011, PRC initiated a 34-hole drilling program on Area 1, primarily to validate the previous exploration efforts. It was concluded that the PRC validation drilling program had adequately tested the Area 1 deposit, both spatially and in number of twinned drilling locations. A three-dimensional geological block model (3DGBM) was constructed for Area 1 using historic and PRC validation drilling data. At a 1% K_2O cut-off grade, the combined measured plus indicated resource was estimated to be 162Mt, carrying an average grade of 3.23% K_2O and 13.90% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade was estimated to be 5.98%. As of April 16, 2012, approximately 66% of the identified resource was classified as measured resource and 34% as indicated resource.

Between July and September, 2012, PRC completed a 50-hole infill drilling program in Area 2 to define the extent of alunite mineralization. Previous exploration in Area 2 was limited to 18 drill holes completed by ESI. Following the 2012 drilling program, there was sufficient geologic and analytical data to support a resource estimate for Area 2 in addition to Area 1. The Area 2 historic resources were reported as part of a Preliminary Economic Assessment (PEA) that included Area 1 and Area 2 that was prepared by Norwest Corporation in 2012.

The measured, plus indicated, historic resources reported in the 2012 PEA were 156.3Mt for Area 1 and 464.4Mt for Area 2.

Between January and February of 2013, a third reverse circulation drilling program was undertaken to further improve resource delineation and geologic assurance. In addition to exploration drilling, PRC completed 10 monitor wells surrounding the alunite deposits and completed three test holes on ML 52364 to determine the groundwater potential of the Wah Wah Valley. Two holes were converted to monitor wells to further assess future development of a wellfield.

In December, 2013, a PFS was completed on Blawn Mountain Project for PRC. Resources were once again updated to reflect the latest drilling along with mine-planning, metallurgy, processing design, infrastructure requirements and environmental assessments. Highlights from the 2013 PFS include:

- 425Mt of Proven and Probable Reserves;
- Average grade of 3.51% K_2O (6.49% K_2SO_4);
- Mine production of 10.6Mtpy after 3-year ramp-up to full production;
- 24 years of active mining plus 16 years of stockpiling for 40-year supply of ore;
- 645,000tpy production of SOP;
- 1.44Mtpy production of sulfuric acid.

In August, 2014, PRC secured its Large Mining Permit for Blawn Mountain from the Utah Division of Oil, Gas and Mining. The Large Mining Permit is the primary permit required in Utah for project development.

There has been no known mining of alunite or any other mineral resource from the Blawn Mountain Property.

1.3 GEOLOGICAL SETTING AND MINERALIZATION

The Blawn Mountain Project is located in the southern Wah Wah Mountains, of the eastern Basin and Range province, in an area characterized by a thick Paleozoic sedimentary section that was:

- Thrust-faulted during the Sevier Orogeny;
- Buried under a thick layer of regionally distributed Oligocene volcanic rocks and locally-derived volcanic rocks;
- Extended to the west by the Basin and Range event;
- Altered by H_2S rich hydrothermal alteration related to a postulated shallow laccolithic intrusive which domed and altered the overlying calc-alkaline volcanic rock (Hofstra, 1984);
- Affected by continual erosion of the ranges contributing to colluvial and alluvial deposition in the valleys.

The geologic characterization of the deposit is essentially that of an altered volcanic tuff. The host tuff deposit ranges in thickness from several hundred to one thousand feet at its thickest point. The property is moderately faulted with normal faults related to Basin and Range extensional block faulting. The deposit is controlled by its original alteration geometry, block faulting and by erosion.

The Blawn Mountain deposit occurs along four ridges, three of which occur within PRC's exploration tracts. Alteration tends to be in linear bodies, reflecting the role of normal faults in controlling the mineralization. Alteration is zoned away from the point of hydrothermal fluid upwelling. The mineralized ridges are erosional remnants of a once larger altered area.

1.4 EXPLORATION

The Blawn Mountain Project was first evaluated in 1969 by ESI as part of a nationwide alunite exploration program which included literature searches, aerial reconnaissance for the bleached alunite zones, and field studies. In 1970, ESI started the first systematic exploration of the Blawn Mountain deposit. ESI completed a total of 320 drill holes throughout the property.

Area 1 had been the most extensively delineated area by the advancement of 230 drill holes. Approximately 33 drill holes terminated in the alunite deposit, therefore mineralization may continue vertically downward in places. Areas 2, 3 and 4 were not fully delineated horizontally or vertically; 12 drill holes were advanced in Area 3 (one of which stopped in the mineral deposit), 17 drill holes were advanced in Area 2 (four of which stopped in the mineral deposit) and three drill holes were advanced in Area 4 (one of which stopped in the mineral deposit). Previous drill samples no longer exist so additional study of these samples is not possible.

After acquiring the property in 2011, PRC initiated a validation drilling program on Area 1, primarily to validate the previous exploration efforts. Under the guidance of North American Exploration Company (NAE), a combination of 19 core holes and 16 reverse circulation holes were completed on Area 1 between October 2011 and February 2012. All 35 drill holes were twinned to locations of previous drill holes completed by ESI. PRC's validation drilling program was followed by an infill drilling program during the summer of 2012 in Areas 1 and 2 combined with exploration/reconnaissance drilling in other areas. The drilling program included 17 additional holes on Area 1, 50 holes on Area 2, 2 holes on Area 4, and 21 holes on the ridgeline extending southwest of Area 1 referred to as the Southwest Extension. A third reverse circulation drilling program was conducted in January and February of 2013. The program included two holes on Area 1 and 17 holes on Area 2 for a combined total of 8,310ft. The primary purpose of the drilling was to further increase geologic assurance for resource assessment.

A total of 90 drill holes were completed, including 74 reverse circulation holes, 8 HQ (2.5 inch (in.) diameter) core holes and 8 PQ (3.4in. diameter) core holes. PQ core holes were completed to collect material for metallurgical testing. A total of 32,392ft were completed in the reverse circulation and core drilling program. In addition to the resource drilling, PRC completed 10 groundwater monitor wells in valley fill material to begin baseline characterization of near-surface hydrology. The 10 monitor wells represent a total drilling footage of 2,400ft. An onsite quality assurance/quality control (QA/QC) manager oversaw all procedures being employed in data collection and sampling. The QA/QC manager was responsible for ensuring that geology logs, geophysics, sampling and surveying met established protocols and procedures and that a proper chain of custody was followed for the disposition of all samples. Figures 9.2 and 9.3 in Section 28 show the drilling completed by PRC in Area 1 and Area 2, respectively.

1.5 MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical testing of the Blawn Mountain deposit has been performed during three major programs. The earliest was performed under the direction of Earth Sciences Inc. (ESI) during the 1970’s. The next two programs were performed under the direction of Potash Ridge Corporation (PRC) during the period 2011 to 2013. The majority of testing has been performed by Hazen Research Inc. in Golden, Colorado.

In 1970, ESI contracted Hazen Research Inc. (HRI) in Golden, CO to develop and perform an extensive metallurgical testing program on alunite composite samples from the NG Alunite project.

Beginning in 2011, PRC engaged HRI to perform a series of bench-scale experiments on the alunite samples taken from the test pit developed by ESI in the 1970’s, to recover SOP and possibly Al_2O_3 from alunite. The objective of the program was to confirm the results of the test-work performed in the 1970’s and provided the basis for the PEA issued by PRC in November 2012.

The 2013 program was a continuation of the work using HRI as the main laboratory, PRC also enlisted the services of several laboratories, turnkey proprietary technology suppliers and equipment vendors to assist in process-optimization through additional testing of run of mine (ROM) ore samples, pilot plant test residues, desktop process simulation and modeling studies. The laboratories included:

- Hazen Research, Inc. (HRI) Golden, CO
- ALS Metallurgy Kamloops, BC, Canada
- JK Tech, Brisbane, Queensland, Australia
- Phillips Enterprises LLC, Golden, CO

Metallurgical tests performed to date include size-reduction of ROM ore, concentration of alunite by flotation, drying, calcination and roasting of whole ore and concentrate including an assessment of the composition of roaster off-gases as feed to the sulfuric acid plant, calcine leaching for extracting soluble SOP, solid/liquid separation of leached slurry to recover the brine (filtrate), evaporation and crystallization of brine to recover the product, and management of alumina-rich process residues.

Results of metallurgical experiments from the 2013 test program at HRI (T. J. Salisbury, June 24, 2013) including test-work subsequent to the June report, from the 2011 and 2012 HRI test program (R. J. Mellon, May 21, 2012) as well as those from 1972 for Earth Sciences Inc., ESI, also by HRI (F. J. Bowen, et al. April 12, 1973) are summarized in the following sections.

Metallurgical tests were performed at HRI on four distinct bulk and PQ core composite samples, identified as Master Composites MC-A, MC-B, MC-C, and Bulk 2 Composite, prepared from 200kg of samples received in May 2013.

Table 1.1 presents the results of analyses of the master composite head samples.

Table 1.1 ICP-OES Chemical Head Assays of Major Constituent Elements*

Sample ID	Analysis, %					
	K	S	Al	Na	Fe	Ti
Bulk 2	4.47	NA	10.2	0.190	0.820	0.284
MC-A	2.74	5.47	6.82	0.388	1.88	0.306
MC-B	3.31	5.47	7.50	0.221	1.13	0.245
MC-C	2.48	4.12	7.25	0.343	1.57	0.201

* Adopted from T. J. Salisbury, June 24, 2013

Determination of mineral composition and fragmentation characteristics using Particle Mineral Analysis (PMA) via Quantitative Evaluation of Materials by Scanning Microscopy (QEMSCAN™) revealed that alunite and quartz are the dominant minerals in the samples and for >75µm the assays ranged, respectively, between 26 and 41% and 53 and 62%. As the alunite concentration increases the quartz content decreases.

During 2013, HRI completed a comprehensive comminution testing program utilizing the PQ core composite samples, which included the following tests:

Table 1.2 provides a summary of test results of abrasion index and work indices.

Table 1.2 Summary of Ai, BWi, RWi, and CWi Results*

PRC ID	Ai, g	BWi, kWh/t	RWi, kWh/t	CWi, kWh/t
MC-A	0.4057	12.0	10.6	10.3
MC-B	0.4132	14.7	13.3	10.0
MC-C	0.4838	14.5	10.9	10.2

*T. J. Salisbury, June 24, 2013.

Table 1.3 is a summary of the semi-autogenous grinding Mill Comminution test results.

Table 1.3 Summary of SMC Test Results*

Parameter	MC-A	MC-B	MC-C
Specific gravity	2.60	2.49	2.32
A - Maximum Breakage	79.8	70.9	75.4
b – Relation between energy and impact breakage	1.07	1.25	1.64
A x b = Overall AG-SAG hardness	85.4	88.6	123.7
DW_i = Drop-weight index, kWh/m^3	3.04	2.8	1.88
DW_i %	17	15	8
M_{ia} = Coarse particle component, kWh/t	10.7	10.5	8.2
M_{ih} = High-pressure grinding roll (HPGR) component, kWh/t	6.8	6.5	4.7
M_{ic} = Crusher component, kWh/t	3.5	3.4	2.4
T_a = Low-energy abrasion component of breakage	0.85	0.92	1.38

*T. J. Salisbury, June 24, 2013; AG = Autogenous Grinding

Rougher flotation tests were performed at a primary grind size of 80%, passing 80µm at pH 10 with potassium hydroxide (KOH) for pH control, oleic acid and diesel fuel oil as collectors and a methyl isobutyl carbinol (MIBC) frother. Approximately 99% of the alunite in the feed (ROM ore) was recovered, averaging 77% of the mass of the feed from the MC-A and MC-B samples. Approximately 90% of the alunite in the feed was recovered, which averaged 51% of the mass of the feed from the MC-C sample.

Cleaner flotation tests performed at pH 10 with KOH for pH control, oleic acid and diesel fuel oil as collectors, methyl isobutyl carbinol (MIBC) frother and several depressants such as sodium silicate (Na_2SiO_4), starch-based PE_2S as well as Cyquest 40E and Cyquest 32243 for gangue minerals.

Drying and Roasting Experiments:

- During 2013, a parallel path of process development consisted of experiments evaluating drying and calcination of both whole ore and alunite flotation concentrate.
- Moisture content of the filter cake is an important factor because drying and roasting are energy-intensive and the filter cake is the feed to the thermal processing units. Additionally, the amount of filtrate recovered has a significant role in water conservation in the operations. Therefore, PRC commissioned Pocock Industrial to perform sedimentation and filtration tests on the water-leach slurry.
- In the 3rd quarter of 2013, PRC contracted FLSmidth Pyrometallurgy group in Bethlehem, PA to evaluate the most energy-efficient method of drying, calcining and roasting either alunite flotation concentrate or whole ore filter cake at an estimated 10wt% moisture and P80 of 1000µm (1.2mm maximum).
- FLSmidth has performed desktop simulation studies on flash dryer/calciner/roaster circuit. Preliminary evaluations estimated that four thermal processing units, each rated at 330tph with a residence time of 2 seconds, are required.
- FLSmidth also reviewed the effect of the size of the particles in the feed and the temperature regime of treatment on selection of the rotary kiln, or the fluidized bed reactor, or the gas suspension calciner as the thermal processing unit.
- FLSmidth estimates from simulation studies the concentration of SO_2 and SO_3 at 9.44 vol. % in the off gases as feed to the acid plant.

Leaching Calcine with Water and Effects of pH:

- During 2013, HRI completed a total of 38 calcine leaching experiments to determine the staged roasting concept, establish operating conditions for the leaching stage and produce brine or leach liquors for the crystallization experiments.
- During 2013, a series of water leach tests were completed on calcines generated using reductants such as sulfur, carbon monoxide/carbon dioxide, hydrogen sulfide and natural gas;
- The results of these tests show that sulfur and natural gas give the best possible overall calcine-leach results.
- It is estimated that better than 90% of the potassium can be extracted by leaching the calcine with water.
- A series of calcine leach tests were completed using Master Composite B to evaluate the effect of particle size of the feed on calcine-leach performance. The tests used natural gas as the reductant. The feed size varied between 100 and 1500µm in five batch kiln tests, followed by leaching the calcine with water.
- Preliminary test results indicated that potassium can be leached from calcine feed of 1.5mm with only a minor reduction in extraction. 1000µm was chosen as the design particle size.

During 2012, HRI leached calcines with water at 90°C. To evaluate the effect of leaching temperature on extraction, a sample of calcine generated at 800°C and 30 minutes residence time was leached at room temperature (25°C):

- Leaching at 25°C produced a potassium extraction of only 61% and 4% for aluminum.
- Comparable leaches conducted at 90°C resulted in 83% extraction for potassium and 9% for aluminum.

Leaching Alumina:

During 2013, HRI conducted medium and high temperature sodium hydroxide leach tests using autoclaves to determine whether the alumina contained in the SOP leach residue, which was produced by low temperature calcining, remained in a form that is soluble in hot sodium hydroxide solutions similar to those used in the Bayer alumina refining process. The tests were successful and confirmed that, with low temperature calcining, the alumina remained in a soluble form.

Solid Liquid Separation Testing:

Solid liquid separation tests were performed by Pocock Industrial on samples of roaster feed and leach residue material for each of the master composites. The results of tests performed in October 2013, by Pocock Industrial on Composite A-, Composite B- and Composite C Calcine Leach Slurry samples, respectively, are summarized below:

- Physical Properties: The respective values for Composite A-, Composite B- and Composite C calcine leach slurry samples are:
    - Liquid specific gravity: 1.04, 1.05 and 1.05;
    - Solids specific gravity: 2.76, 2.78, and 2.90; and *pH:* 5.5, 5.6, and 5.2
- Flocculant Screening: Hychem AF 304, a medium-to-high molecular weight, 15% charge density, anionic polyacrylamide is the recommended flocculant. For Composite A-, Composite B- and Composite C slurry samples at 20°C and initial pulp density of 30%, 20% and 30% solids, respectively, the maximum effective dosage at flocculant concentration of 0.1g/L was in the range 20 – 30g/Mt, 25 – 35g/Mt and 30 – 40g/Mt.
- Conventional (static) thickening: At recommended flocculant dosages for Composite A-, Composite B- and Composite C calcine leach slurry samples, the respective maximum unit area for conventional (static) thickener sizing is 0.125m^2/Mtpd, 0.160 – 0.205m^2/Mtpd and 0.125m^2/Mtpd.

- The maximum solids content of the feed to the thickeners is 25 – 35wt%, 30 – 40wt%, and 20 – 30wt% and the estimated underflow pulp density is 69 – 73wt%, 61 – 65wt% and 69 – 73wt%, respectively, for the three slurry samples
- Pulp rheology: pulp viscosity data was collected using the Fann (Model 35A) Viscometer, after destroying the long-chain molecular structure of the flocculant used. The data classified the thickener underflow pulps as a pseudoplastic class of "non-Newtonian fluids," or one in which the flow behavior is dependent on the shear rate and changes with applied stress.
- Vacuum Filtration: The design criteria for selection of the horizontal belt vacuum filter, based on tests performed on Composite A-, Composite B- and Composite C calcine leach slurry samples, are as follows:
  - Filter feed solids = 71.3%; pH = 7.2; vacuum level = 67.7kPa; moisture content of filter cake ranged from 17.8wt% to 22.5wt% for cake thickness in the 10mm to 15mm range
  - For Composite A-, Composite B- and Composite D samples the production rate ranged, when flocculant was *not* used, respectively, from 487 to 594kg/m^2.hr, 163 to 225kg/m^{2}hr, and 166 to 231kg/m^{2}hr and at flocculant dosage of 70g/Mt ranged, respectively, from 1089 to 1363kg/m^{2}hr, 1185 to 1564kg/m^{2}hr, and 1189 to 1608kg/m^{2}hr
  - Pressure Filtration: The results of two sets of Automatic Pressure Filter tests performed on each of Composite A, Composite B and Composite D samples are as follows:
  - Feed solids ≈ 70wt%; pH = 7.2; feed pressure = 551.6kPa; cake thickness = 60mm; moisture content of filter cake from 11.5wt% to 14.2wt%; Total cycle time from 16 to 18 minutes; sizing basis in m^3 of pressure filter volume per metric tons of dry solids ranged from 10.74 to 16.19m^3/cycle, including 1.25 scale-up factor.

Sulfuric Acid Plant: Based on information developed by FLSmidth by desktop simulations for the whole ore case, Du Pont-MECS estimates that two sulfuric acid plants each, with a throughput capacity of 2,000tpd of concentrated acid be constructed, each dedicated to process the off-gases from two (2) lines of roasters and with provisions for treating off-gases bypassing any of the Roasters being serviced.

Recovery of elemental sulfur from the roaster off-gases was considered as an alternative to manufacture of sulfuric acid at the Project site. PRC contracted Fluor Corporation (Fluor) to evaluate the technical feasibility and economic viability of a Sulfur Recovery Unit (SRU):

Fluor identified the following two technologies: the Fluor SO_x® Claus process for production of bright yellow sulfur and use of commercially-proven technology which allows achievement of 99.9 plus percent sulfur recovery. Manufacture of sulfuric acid has been adopted as the preferred option at the project site.

Crystallization of SOP:

- During 2012, HRI performed one crystallization experiment with the goal to generate crystals in several stages so that the co-precipitation of impurities can be evaluated.
- The crystallization experiment recovered 43% of the potassium in the feed solution by reducing the liquor volume 91% by evaporation and the crystals assayed 25wt% K with small amounts of other contaminants.
- The Cycle 1 filtrate as feed to Cycle 2 experiments was evaporated from 327mL, or an additional 79%. After washing, 200mg of crystals were collected, which assayed 25wt%K, < 0.05wt%Al, and < 0.001wt%Ti, 0.021wt%Na, < 0.025wt%Fe, and < 0.0025wt%Mg.
- In a commercial-scale operation, a limited amount of K_2SO_4 will be recovered until other salts begin to crystallize, contaminating the product SOP
- A bleed from the crystallizer will be necessary to prevent contamination of the K_2SO_4 product. Impurity build-up should be controlled in the recirculated centrate by establishing a purge or "bleed" stream from the Centrifuge dewatering circuit.
- Bench-scale investigations will be required to determine the amount of bleed and to identify a method of treatment to recover the contained values. After treatment, the barren solution essentially containing sodium chloride can be routed to an evaporation pond.

1.6 MINERAL RESOURCES AND RESERVES

Resources estimates for Blawn Mountain were last determined and presented in the 2013 PFS. There have been no changes to the resource estimates since the 2013 PFS.

Four potential mine development targets have been identified within the Blawn Mountain Project area. Only Area 1 and Area 2 have sufficient geologic and analytical data to support resource estimation at this time. Areas 3 and 4 are defined by a limited number of historical holes and surface mapping along with only two validation holes in Area 4. Both areas are recognized as future exploration targets.

Resources have been estimated from two 3DGBM's constructed in MineSight®, a software package developed by Hexagon Mining Inc. The estimate was prepared in compliance with NI 43-101 requirements for the definition of mineral resources. The 3DGBM's are based on the assays and lithologies of the current drilling database and on a series of 30 interpreted geological cross-sections constructed through Area 1 and 29 cross-sections constructed through Area 2.

Resource classification is based on set distances from drill hole sample intervals in 3D space. These distances were based on semi-variogram analysis of K_2O sample data (Table 1.4).

Table 1.4 Resource Assurance Criteria from Variography

Compound	Measured	Indicated	Inferred
K_2O	<150ft	<350ft	<2,000ft

Other estimation criteria include assumed density of alunite and waste established at 153.8 pounds per cubic foot (lb/ft^3) or specific gravity of 2.46 grams per cubic centimeter (g/cc).

Resource classification is based on the CIM Standards on Mineral Resources and Reserves, a set of definitions and guidelines established by the Canadian Institute of Mining and Metallurgy and Petroleum. Table 1.5 shows the estimated classified resource for Area 1 at increasing incremental K_2O cut-off grades and Table 1.6 shows the estimated classified resource for Area 2 at increasing incremental K_2O cut-off grades.

Table 1.5 Classified Resource Estimate for the Blawn Mountain Area 1 Alunite Deposit

			IN SITU GRADES						CONTAINED RESOURCES				
RESOURCE CLASSIFICATION	CUTOFF GRADE K_2O (%)	IN SITU (TONS)	K_2O (%)	K_2SO_4 (%)	AL_2O_3 (%)	SO_4 (%)	Alunite based on K_2O (%)	Alunite based on Al_2O_3 (%)	K_2O (TONS)	K_2SO_4 (TONS)	Al_2O_3 (TONS)	Alunite based on K_2O (TONS)	Alunite based on Al_2O_3 (TONS)
MEASURED	0.00	72,400,282	3.42	6.32	15.68	2.09	30.04	42.45	2,473,700	4,574,597	11,353,088	21,746,273	30,734,994
	1.00	71,529,372	3.45	6.39	15.71	2.06	30.36	42.52	2,469,909	4,567,586	11,235,691	21,712,945	30,417,176
	2.00	64,979,040	3.64	6.73	16.19	2.09	32.01	43.83	2,366,212	4,375,819	10,520,042	20,801,341	28,479,776
	2.50	56,872,179	3.84	7.10	16.33	2.13	33.75	44.21	2,183,323	4,037,604	9,286,829	19,193,568	25,141,232
	3.00	48,362,178	4.03	7.44	16.62	2.19	35.39	45.01	1,946,916	3,600,418	8,040,067	17,115,319	21,766,008
	3.50	34,526,334	4.33	8.00	17.31	2.31	38.03	46.87	1,493,713	2,762,313	5,976,957	13,131,213	16,180,773
	4.00	19,624,648	4.78	8.84	18.71	2.55	42.01	50.64	937,725	1,734,127	3,671,046	8,243,526	9,938,226
INDICATED	0.00	93,823,555	3.26	6.03	15.19	2.03	28.65	41.12	3,057,710	5,654,601	14,250,203	26,880,292	38,578,041
	1.00	93,313,743	3.27	6.05	15.19	2.02	28.78	41.12	3,054,532	5,648,725	14,172,865	26,852,358	38,368,671
	2.00	83,493,484	3.48	6.43	15.88	2.05	30.57	42.98	2,902,985	5,368,470	13,255,676	25,520,109	35,885,665
	2.50	74,184,688	3.63	6.71	16.06	2.09	31.91	43.47	2,692,385	4,979,008	11,911,168	23,668,726	32,245,822
	3.00	57,939,557	3.87	7.15	16.40	2.14	34.01	44.40	2,241,624	4,145,419	9,503,304	19,706,088	25,727,272
	3.50	36,959,714	4.21	7.78	17.30	2.29	36.97	46.84	1,554,489	2,874,705	6,395,250	13,665,492	17,313,172
	4.00	17,565,100	4.73	8.75	19.11	2.57	41.61	51.73	831,391	1,537,486	3,356,462	7,308,752	9,086,589
MEASURED AND INDICATED	0.00	166,223,837	3.33	6.15	15.40	2.06	29.25	41.70	5,531,410	10,229,198	25,603,291	48,626,565	69,313,034
	1.00	164,843,115	3.35	6.20	15.41	2.04	29.46	41.73	5,524,441	10,216,310	25,408,555	48,565,303	68,785,847
	2.00	148,472,524	3.55	6.56	16.01	2.07	31.20	43.35	5,269,197	9,744,288	23,775,718	46,321,450	64,365,441
	2.50	131,056,867	3.72	6.88	16.17	2.11	32.71	43.79	4,875,708	9,016,612	21,197,996	42,862,294	57,387,054
	3.00	106,301,735	3.94	7.29	16.50	2.17	34.64	44.68	4,188,540	7,745,837	17,543,371	36,821,407	47,493,280
	3.50	71,486,048	4.26	7.89	17.31	2.30	37.49	46.85	3,048,201	5,637,017	12,372,207	26,796,705	33,493,946
	4.00	37,189,748	4.76	8.80	18.90	2.56	41.82	51.16	1,769,116	3,271,614	7,027,508	15,552,278	19,024,815
INFERRED	0.00	2,255,374	3.18	5.87	14.62	2.04	27.92	39.57	71,626	132,458	329,697	629,665	892,554
	1.00	2,255,374	3.18	5.87	14.62	2.04	27.92	39.57	71,626	132,458	329,697	629,665	892,554
	2.00	1,919,126	3.48	6.44	15.87	2.10	30.60	42.95	66,797	123,527	304,485	587,213	824,299
	2.50	1,793,895	3.56	6.59	15.98	2.11	31.33	43.26	63,938	118,240	286,657	562,078	776,036
	3.00	1,429,416	3.77	6.97	16.36	2.11	33.15	44.29	53,899	99,675	233,868	473,825	633,126
	3.50	665,917	4.37	8.08	18.68	2.44	38.41	50.56	29,097	53,809	124,375	255,794	336,706
	4.00	407,414	4.78	8.84	20.45	2.68	42.05	55.35	19,486	36,036	83,298	171,303	225,504

Table 1.6 Classified Resource Estimate for the Blawn Mountain Area 2 Alunite Deposit

RESOURCE CLASSIFICATION	CUTOFF GRADE K_2O (%)	IN SITU (TONS)	IN SITU GRADES K_2O (%)	K_2SO_4 (%)	AL_2O_3 (%)	SO_4 (%)	Alunite based on K_2O	Alunite based on Al_2O_3	CONTAINED RESOURCES K_2O (TONS)	K_2SO_4 (TONS)	Al_2O_3 (TONS)	Alunite based on K_2O (TONS)	Alunite based on Al_2O_3 (TONS)
MEASURED	0.00	110,497,331	2.87	5.31	12.42	1.94	25.23	33.62	3,170,721	5,863,592	13,720,785	27,873,773	37,144,805
	1.00	104,377,825	3.02	5.58	13.05	2.04	26.54	35.33	3,150,958	5,827,044	13,622,767	27,700,035	36,879,452
	2.00	93,679,360	3.18	5.87	13.38	2.11	27.91	36.22	2,974,320	5,500,388	12,532,237	26,147,212	33,927,177
	2.50	79,064,980	3.34	6.17	13.60	2.20	29.33	36.81	2,638,240	4,878,879	10,750,703	23,192,741	29,104,220
	3.00	50,041,863	3.68	6.80	14.26	2.43	32.34	38.60	1,841,190	3,404,900	7,134,769	16,185,883	19,315,191
	3.50	28,969,753	3.98	7.36	14.75	2.62	34.97	39.93	1,152,272	2,130,888	4,272,459	10,129,610	11,566,369
	4.00	9,150,291	4.47	8.26	15.62	2.72	39.27	42.29	408,771	755,937	1,429,550	3,593,501	3,870,067
INDICATED	0.00	307,822,418	2.84	5.25	12.47	1.90	24.96	33.77	8,739,386	16,161,686	38,398,692	76,827,849	103,952,645
	1.00	293,961,004	2.96	5.47	12.97	1.97	25.98	35.12	8,688,605	16,067,777	38,135,561	76,381,434	103,240,300
	2.00	263,614,932	3.10	5.73	13.34	2.05	27.25	36.12	8,171,536	15,111,563	35,174,931	71,835,880	95,225,305
	2.50	212,810,329	3.30	6.10	13.64	2.15	29.00	36.93	7,020,826	12,983,563	29,031,159	61,720,000	78,592,933
	3.00	130,484,506	3.64	6.74	14.25	2.37	32.04	38.57	4,755,377	8,794,086	18,592,476	41,804,470	50,333,410
	3.50	71,126,489	3.96	7.32	14.70	2.57	34.81	39.79	2,816,324	5,208,209	10,454,527	24,758,277	28,302,416
	4.00	20,689,481	4.48	8.28	15.52	2.62	39.37	42.03	926,537	1,713,438	3,211,794	8,145,177	8,694,943
MEASURED AND INDICATED	0.00	418,319,749	2.85	5.27	12.46	1.91	25.03	33.73	11,910,107	22,025,277	52,119,477	104,701,621	141,097,450
	1.00	398,338,829	2.97	5.50	12.99	1.99	26.13	35.18	11,839,563	21,894,821	51,758,329	104,081,470	140,119,752
	2.00	357,294,292	3.12	5.77	13.35	2.07	27.42	36.15	11,145,855	20,611,952	47,707,169	97,983,092	129,152,483
	2.50	291,875,309	3.31	6.12	13.63	2.17	29.09	36.90	9,659,066	17,862,442	39,781,862	84,912,741	107,697,153
	3.00	180,526,369	3.65	6.76	14.25	2.39	32.12	38.58	6,596,568	12,198,986	25,727,245	57,990,353	69,648,601
	3.50	100,096,242	3.96	7.33	14.71	2.59	34.85	39.83	3,968,596	7,339,097	14,726,986	34,887,887	39,868,784
	4.00	29,839,772	4.47	8.28	15.55	2.65	39.34	42.11	1,335,308	2,469,376	4,641,344	11,738,678	12,565,010
INFERRED	0.00	150,481,703	2.64	4.88	12.08	1.67	23.18	32.70	3,967,300	7,336,699	18,175,180	34,876,487	49,203,709
	1.00	134,770,366	2.90	5.37	13.25	1.84	25.51	35.88	3,911,306	7,233,150	17,862,464	34,384,244	48,357,127
	2.00	124,717,186	2.99	5.54	13.46	1.88	26.31	36.44	3,733,035	6,903,475	16,787,557	32,817,068	45,447,146
	2.50	94,690,184	3.23	5.97	13.70	1.97	28.37	37.10	3,055,842	5,651,146	12,974,828	26,863,870	35,125,355
	3.00	55,899,862	3.56	6.59	14.25	2.15	31.31	38.58	1,990,874	3,681,709	7,967,240	17,501,748	21,568,850
	3.50	21,879,368	4.02	7.44	14.72	2.51	35.38	39.85	880,623	1,628,529	3,220,577	7,741,544	8,718,722
	4.00	9,143,043	4.45	8.23	15.46	2.63	39.13	41.86	406,939	752,549	1,413,798	3,577,392	3,827,423

The preferred scenario for resource presentation is a 1% K_2O cut-off grade. At a 1% cut-off grade, the combined measured plus indicated resource for Area 1 is 164.8Mt at an average grade of 3.35% K_2O and 15.41% Al_2O_3. For Area 2, the combined measured plus indicated resource is 398.4Mt at a 1% cut-off grade with an average grade of 2.97% K_2O and 12.99% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade for Area 1 is 6.20% and for Area 2 is 5.50%.

Approximately 43% of the identified resources for Area 1 are classified as measured, 56% as indicated resource and 1% as inferred resource. Approximately 20% of the identified resources for Area 2 are classified as measured, 55% as indicated resource and 25% as inferred resource.

Utilizing the geologic model and resource areas, a mine plan was developed to meet certain criteria related to project economics, grade, target production rates and processing methods. The mine plan was developed by applying various criteria in selecting the method and approach to mining, including:

- Annual production rate (ROM ore) to be constrained by processing capacities of 3.4 Million tons per year (Mtpy).
- Ore cut-off grades of approximately 3.75% K_2O (Area 1) and 3.50% K_2O (Area 2) were utilized during the active mining phase for approximately 28 years of the project for ore going directly to the processing facility. Lower grade ores down to 2.75% K_2O are placed in temporary low-grade stockpiles. Once active mining ceases in Area 1 and Area 2, the low-grade ore stockpiles are reclaimed and sent to the processing facility. Low-grade ore stockpile reclamation occurs for roughly an additional 18 years.
- Maximize economic use of the resource.

Taking into consideration the above, mine-plans were developed (detailed further in Section 16) that use standard surface mining 'truck-shovel' techniques to mine the deposit. Mine development was scheduled using MineSight® software to generate a LOM schedule of waste and ore volumes. The mine-plan formed the basis of workforce demands and schedules leading ultimately to estimates of capital and operating costs. Taking into account commodity pricing and market conditions, a cash flow of revenues and direct and indirect costs for both the mine and the processing plant were developed, which ultimately led to an estimate of project economics and value. The mine-plan, at a prefeasibility level of assurance, was found to be of positive economic value and forms the basis of mineral reserves reported in Table 1.7.

Table 1.7 Mineral Reserves, by Category

Alunite Ore (ROM tons)	Grade (% K_2O)	Grade (%K2SO4)	Reserve Category Proven ('000 tons)	Reserve Category Probable ('000 tons)	Total
Direct Feed to Mill	4.32	7.99	36,882	53,917	90,799
Medium Grade Stockpile	3.49	6.47	14,916	23,158	38,074
Low-grade Stockpile	2.99	5.52	7,983	16,473	24,457
Total	**3.90**	**7.22**	**59,782**	**93,548**	**153,330**
Products					
SOP tons			**4,135**	**6,468**	**10,603**
Sulfuric Acid (tons) @ 93% Purity			**9,410**	**14,725**	**24,135**

This estimate of resources and reserves was generated using the best information available concerning issues related to environmental, permitting, legal, title, taxation, socio-economics, marketing and political factors that could have a material influence on

Millcreek's findings. Millcreek is not aware of any additional factors which may affect our reserve estimate.

The accuracy of resource and reserve estimates is, in part, a function of the quality and quantity of available data and of engineering and geological interpretation and judgement. Given the data available at the time this report was prepared, the estimates presented herein are considered reasonable. However, they should be accepted with the understanding that additional data and analysis available subsequent to the date of the estimates may necessitate revision. These revisions may be material. There is no guarantee that all or any part of the estimated resources or reserves will be recoverable.

1.7 MINING METHODS

Mining operations will use a conventional open-pit, truck and shovel mining approach. This is a typical and standard approach for many surface mining applications and takes advantage of the flexibility of the mining equipment. For Blawn Mountain, Area 1 and Area 2 will be developed in phases that will allow for optimizing the ore grades encountered in the deposit, while providing flexibility to the operation.

The mining plan for this PFS uses a nominal 3.75% K_2O ore grade cut-off for Area 1 and a 3.50% K_2O cut-off for Area 2. These cut-off grades were utilized in the development of the pit shells and mining areas for Area 1 and 2. In addition, two low-grade ore stockpiles will be developed, one for Area 1 and Area 2 respectively. Ore that is less than 2.75% K_2O is considered waste material and will be placed in the waste dumps. The Area 1 low-grade stockpile will contain ore ranging in grade from 2.75% K_2O to 3.75% K_2O and the ore in the Area 2 low-grade stockpile will range from 2.75% K_2O to 3.50% K_2O. This low-grade ore will be fed to the processing plant after mine operations have concluded in year 28.

The mining production schedule (Table 1.8) is driven by the capacity of the processing plant. The ROM ore production schedule for direct plant feed is approximately 3.4Mtpy. Mining operations commence in Area 1 in Year 1. For purposes of this PFS, Year 1 is anticipated to be 2020. Year 1 is considered a construction period for stockpile area development, haul-road construction and initial mining area development. Year 1 is considered a "ramp-up" period and during this time the mining area within Area 1 will be developed in such a way that full production can be achieved beginning in Year 2 (2021). The initial phase of mining in Area 2 is developed in Year 3 in coordination with operations in Area 1.

Table 1.8 Mining Schedule

Year	-1	1	2	3	4	5	6-10	11-15	16-20	21-25	26-30	31-46
PGM(kyd^3)	358	160	90	100	117	82	377	239	213	150	132	314
Waste (kyd^3)	45	103	700	873	1,237	943	4,258	7,081	11,924	7,888	861	
Waste (kt)	94	215	1,455	1,816	2,572	1,962	8,856	14,728	24,801	16,407	1,791	
Ore (kt)	218	1,684	3,365	3,405	3,367	3,362	16,830	16,809	16,813	16,849	8,098	
K_2O (%)	3.9	4.5	4.5	4.6	4.2	4.2	4.3	4.4	4.4	4.1	4.6	
Al_2O_3 (%)	15.2	16.7	17.3	17.0	15.7	15.6	16.3	17.2	16.9	14.9	16.4	
*LGO Stockpile (kt)	83	276	1,107	2,176	3,196	4,225	19,353	13,081	11,024	7,507	502	
*LGO Reclaim (kt)	0	0	0	0	0	0	0	0	0	0	8,729	53,802
K_2O (%)	3.5	3.5	3.4	3.4	3.4	3.4	3.4	3.2	3.1	3.3	3.5	3.3
Al_2O_3 (%)	14.5	14.0	14.7	14.4	14.4	14.4	14.6	14.0	12.2	12.6	14.3	13.7
Acid (kt)	34	288	600	596	545	541	2,814	2,969	2,928	2,588	2,657	7,574
SOP (kt)	15	132	268	277	251	247	1,271	1,309	1,303	1,226	1,198	3,106

*Low-grade Ore

Before mining operations commence, salvageable plant growth material (PGM), also referred to as topsoil, will be removed and placed in temporary storage areas. Minimal waste material will be encountered during operations. The average strip ratio (yd^3/ton ore) is 0.25:1. The majority of the waste material encountered in Areas 1 and 2 will be placed in out-of-pit waste dumps adjacent to the mining areas. Some of the waste material will be used to construct haul roads needed to access Areas 1 and 2.

The production schedule and mining sequence was utilized to develop an equipment fleet that would adequately meet the needs of the mining operation. Table 1.9 presents the type, size and maximum quantity of major mining equipment required to achieve the mine-plan. This equipment was selected because it provides flexibility to support the phased mining approach simultaneously working in Areas 1 & 2. The economics supporting the PFS assumes contract mining with the contractor providing the equipment.

Table 1.9 Major Mining Equipment

Primary Equipment		
FEL	16yd^3	2
End-Dump Truck	100t	12
Support Equipment		
Water Truck	12,000gal	1
Grader	297Hp	2
Dozer	580Hp	5
Drill	45,000lb	1

The major mining equipment will be supported by a fleet of smaller support equipment, including pumps, light plants, lube and fuel trucks, service trucks, pick-up trucks, etc.

The production requirements necessitate a 4-day per week schedule operating 20 hours per day for approximately 208 days per year. The workforce needed to support the mining operation is presented below in Table 1.10. Note that after Year 28, mining operations will have concluded and reclaiming of the low-grade stockpiles will require a significantly smaller work force.

Table 1.10 Average Mining Workforce Requirements

Category	Y-1 – Y5	Y6-Y10	Y11-Y20	Y21-28	Y29-Y46
Hourly Workers					
FEL Operators	2	3	4	3	3
Truck Drivers	8	13	19	21	6
Dozer Operators	6	8	8	6	2
Grader Operators	2	2	3	3	1
Drill Operators	2	3	2	2	0
Blasters	2	2	2	2	0
Mine Support	5	6	6	6	5
Maintenance Labor	16	24	21	18	6
Sub-Total Hourly	**43**	**61**	**65**	**61**	**23**
Management					
Exec., Staff, Tech	7	7	7	7	2
Maintenance	4	4	4	4	1
Operations	4	4	4	4	1
Total Employees	**58**	**76**	**80**	**76**	**27**

1.8 RECOVERY METHODS

The ROM ore will be processed as envisioned, by crushing, reduction roasting, extracting SOP by leaching the calcine with water, solid/liquid separation, evaporation of brine, crystallization as well as drying and packaging of SOP product for markets. Provisions have been made in the process plant to conserve energy and water through treatment and reuse of effluents and disposal of residues in an environmentally sound manner.

The proposed combination of unit operations in processing alunite ore is based on test results of investigations completed in 2012 and 2013 in support of process-optimization at HRI, sedimentation and filtration studies at Pocock Industrial, Inc., in Salt Lake City, UT, and thermal processing systems modeling at FLSmidth in Bethlehem, PA.

Process design criteria for the major unit operations in the proposed integrated process plant complex are summarized below:

- ROM ore production rate is 3.4Mtpy at 2% moisture;
- Plant operation schedule is 330 days per year, 24 hours per day;
- The nominal throughput capacity of the process plant is 425 tons per hour (tph);
- Particle size of the grinding is P_{80} = 1000µm to the calciner;
- Roasting temperature at 1022ºF (550ºC) and not to exceed 1112ºF (600ºC);
- Roaster off-gases are routed as feed to a 2,090 tons per day (tpd)sulfuric acid plant;
- Water leaching of calcine: 35% solids; 176ºF; 60 minutes residence time; and 90% SOP extraction;
- Alumina/silicate leach residues conveyed at 90% solids to the tailings facility.

- Total workforce requirements to operate the processing facility will average 148 throughout the project life. Average manpower requirements for the processing plant are shown in Table 1.11 below.

Table 1.11 Average Processing Plan Workforce Requirements

Category	Y-1 – Y5	Y6-Y10	Y11-Y20	Y21-28	Y29-Y46
Process Operations	68	68	68	68	68
Maintenance	37	37	37	37	37
Electrical / Instrumentation	14	14	14	14	14
Process Assay	6	6	6	6	6
Exec. / Tech. / Staff	23	23	23	23	23
Total Employees	**148**	**148**	**148**	**148**	**148**

1.8.1 Primary Crushing

The ROM ore at minus 12in is delivered to the 400t dual loading hopper that feeds two parallel primary roll crushers, one operational and one stand by. Each roll crusher with a capacity of 425tph will crush the ROM ore to minus 3in.

The crushers are sized to operate 7,920 hours/year. The crushed ore will be conveyed to a collection bin which will feed two parallel screening/secondary crushing units.

1.8.2 Secondary Crushing

The collection bin will feed two dry crushing screens. Screen oversize will report to two secondary cage mills. The crushed product from the cage mills will be recycled back to the dry screens via bucket elevators. Screen undersize (P_{80} – 1mm) will be conveyed to the calciner feed bin. A dust collection system will be installed to manage the dust generated from dry crushing.

1.8.3 Calcining/Roasting

Crushed ore containing alunite and inert solids at approximately 98% solids (2% moisture) is dehydroxylated and roasted to decompose the alunite. The calcine produced contains a mixture of SOP (K_2SO_4) and alumina (Al_2O_3). A start-up air heater is used during the system start-up to bring the flash roaster up to the auto-ignition temperature of the fuel. A portion of the heat energy in the off-gases is used in steam generation.

The calcine produced from the roasting step is quenched with water and the slurry is pumped to the water leach circuit. Calcine particles entrained in the roaster off-gases are separated in a cyclone followed by an electrostatic separator (ESP). The dust collected in the ESPs is also recycled to the water leach circuit.

The gamma-alumina phase occurs in a porous cubic structure which can be leached with sodium hydroxide (NaOH). It reverts at high temperatures to the recalcitrant alpha form with hexagonal close-packed structures. The temperature limits on thermal processing are, therefore, required to ensure that the gamma-alumina crystals are the end-product of alunite roasting operations. Alunite decomposition reactions are, therefore, carried out at 1022^oF (550^oC). The maximum temperature in the roaster not to exceed 1112^oF (600^oC).

1.8.4 Sulfuric Acid Plant

Air-to-gas heat exchangers are used to remove heat in the roaster off-gases to ensure that the gas temperature entering the ESP is in the range of 525-575^oF (275-300^oC). Reducing conditions are created by injecting excess fuel in the roaster to convert most of the generated SO_3 from the decomposition of aluminum sulfate to usable SO_2 in the production of sulfuric acid. The conversion of SO_3 to SO_2 will be regulated based on the amount of excess fuel delivered to the roaster. An estimated 2190stpd of sulfuric acid (H_2SO_4) is manufactured at the project site from sulfur dioxide (SO_2) produced by the decomposition of alunite during the thermal processing of ROM ore.

Sulfur dioxide (SO_2) bearing off-gases from the roaster are sent to the gas cleaning section of the sulfuric acid plant for further processing and to recover 93% sulfuric acid (H_2SO_4). The highly-efficient acid plant has provisions for meeting the applicable emission limits for sulfur oxides (SO_x) in the stack gases discharged into the atmosphere. FLSmidth generally estimates the concentration of SO_2 and SO_3 in the flue gases from the thermal processing units at approximately 9.5+ vol. % as feed to the acid plant. This value is an estimate based on similar industry experience. No pilot plant testing has been completed by FLSmidth to date.

Emissions to the atmosphere from the project site sulfuric acid plant will be in compliance with the applicable Air Quality Permit requirements.

1.8.5 Calcine Water Leach

Based on bench-scale test results at HRI, it is proposed that the calcine discharged from the roaster will be leached with water at a pulp density of 35% solids. After dissolving approximately 90% of the K_2SO_4 in the calcine, the leach residue slurry at approximately 33% solids is filtered in a battery of belt filters and the filtrate (brine), containing approximately 10.5 to 11.0% dissolved K_2SO_4, is pumped to the evaporator/crystallizer circuit. The filter cake at about 90% solids, is washed with water and the washed filter cake, consisting of inert solids Al_2O_3 and undissolved K_2SO_4, is conveyed at approximately 90% solids to the tailings facility for dry stacking.

1.8.6 Solid/Liquid Separation

The leached slurry at the end of the leach cycle is filtered by using vacuum belt filters and consists of inert solids originally present in the ROM ore, Al_2O_3 produced in the roaster, an aqueous solution containing approximately 90% of the K_2SO_4 extracted during leaching, plus approximately 10% of unleached K_2SO_4 solids. Solid/liquid separation is one of the most significant unit operations at the project site because of its importance in energy conservation both in the calciner/roaster and evaporator/crystallizer circuits, water conservation through reuse of effluents as well as its impact on the size and type of water and wastewater treatment facilities.

1.8.7 Evaporation/Crystallization

Once the dissolved K_2SO_4 content of the filtrate from the calcine leach circuit has reached an estimated 10.5 to 11.0% range through recirculation of the filtrate, approximately 35% of the filtrate will be pumped as feed to the crystallizer circuit. The remaining 65% of the filtrate and cake wash water, as required, will be recycled to quench the calcine solids discharged from the roaster and to maintain the solids content of the slurry in the leach tanks at 35%.

Recovery of SOP crystalline product from the filtrate containing K_2SO_4 in the solution consists of an integrated crystallizer system followed by separation of the SOP crystals from slurry, drying, sizing and packaging of the product for shipment.

The product from the crystallizer is transferred to a thickener. The thickened slurry discharged from the thickener is then fed to a centrifuge where the crystals are separated from the liquid in the slurry. The wet crystals are transported to the dryer/cooler. The centrate from the centrifuge is collected in a tank and returned to the crystallizers. The dried product conveyed from the crystallizer system will be treated and packaged to meet market specification.

1.8.8 Power, Gas and Steam Requirements

The alternative to on-site power generation for the facility is to build a 138kV power transmission line which ties into the existing electrical grid. The only option for routing an electrical power transmission line is above ground. The routing for an electrical power transmission line will be approximately 46 miles.

The processing facility requires approximately 200,000lbs/hr of steam for use in the process. Currently no steam will be generated in the sulfuric acid plant. The steam will be provided using packaged boilers fired with natural gas.

A substantial supply of natural gas will be required for the facility. This gas will be used to power the boilers to produce the steam required for the processes. It will also be used for the calciner roaster and general heating needs throughout the facility. An emergency generating system is needed for the facility to mitigate the adverse effects of a sudden loss of power on the mineral processing system.

1.8.9 Water Conservation Measures

The Blawn Mountain project is designed to be a zero-discharge facility in regards to water. In addition, dry grinding has been selected as part ore processing which further aids in water conservation efforts. This project will make all reasonable efforts to conserve water.

1.9 PROJECT INFRASTRUCTURE

The infrastructure needed to support the mine and processing facility includes the following:

- Project access
- Water supply
- Power supply
- Natural gas supply
- Mine facilities
- Surface water management
- Tailings management
- Saleable product transportation
- Miscellaneous support

The Blawn Mountain Project will be accessed via upgrades to existing county roads Revenue Basin and Willow Springs off of State Highway 21. In addition to these road upgrades, an existing county road will be relocated west of the project area to provide a bypass for motorists and recreational users. An access road into the project site will be developed from the intersection with the county bypass road to the mine site, continuing into the processing plant location.

Water supply for this project will be provided from the Wah Wah Valley Aquifer. The water system will consist of wells, surge pond/tanks, booster pumps, pipeline, storage tanks and water treatment and distribution facilities.

This study assumes new transmission and natural gas lines will be constructed to the site to provide power and gas to the project. It is anticipated that both of these lines will be owned and operated by the respective utility company.

Facilities constructed to support the mining operation include:

- Truck shop
- Warehouse
- Administration complex (offices, training, change-house)
- Fuel depot
- Explosives storage
- Equipment ready-line
- Guard shack

Storm water controls will be located downstream of all surface disturbances. These controls will consist of diversion and collection ditches, sediment ponds, outlet control structures and a combination tailings/runoff containment structure, and a water collection pond and collection pond.

As ore is processed, tailings are produced, requiring storage. Tailings will be conveyed from the processing plant to the tailings storage area. It is anticipated that the tailings will be coarse grained sand that will yield little free draining water. A collection pond and settling pond will be constructed to collect water released from the tailings and the runoff from the site and process facilities, mine haul roads and associated areas.

Saleable products, both SOP and sulfuric acid, will be transported via over-the-road trucks to a storage and loadout facility near Milford where products will be stored then loaded onto the existing rail system. Miscellaneous supporting infrastructure, including sanitary waste treatment, firefighting capabilities and site transportation have also been considered for this PFS.

1.10 MARKETS AND CONTRACTS

SOP produced from the Blawn Mountain Project will primarily be marketed domestically. The demand for fertilizer is expected to remain robust for the foreseeable future and according to Potash Corp., the global demand for MOP is expected to average a long-term growth rate of approximately 3.0%. The majority of that increase is expected to come from Latin America, South East Asia, India and China.

SOP is a fertilizer of choice when the presence of chloride is undesirable. SOP sells at a premium over MOP due to its delivery of a high analysis potassium nutrient (K2O content at min. 50%), non-chloride formula, limited primary SOP production and the high cost to produce SOP through secondary production methods. For the period of 2000 - 2015, SOP premium over MOP in North American markets has increased from approximately $120 to $480/tonne.

The primary users of SOP are specialty crops, broadly defined as tree nuts, fruits and vegetables crop categories. The major demand market (and markets with significant production) of SOP are the US, Europe and China. According to Green Markets, total implied demand of SOP was approximately 7.5Mtpy in 2015 with an expectation to grow to 9.6Mtpy by 2026. SOP consumption in the US is largely constrained by the availability of product and substantially higher premium over MOP. Compass Minerals is currently the only producer of SOP in the US, with sales volume of approximately 400,000 tonnes of SOP in 2015-16. Fully served North American SOP market, as estimated by the North American SOP producer, is approximately 0.75Mtpy. The US market has historically been underserved. Specialty crops are best suited for using SOP and account for approximately 40% of total crop revenues. SOP is a fertilizer of choice with tree nuts, fruits and vegetables growers in California.

PRC has entered into a Memorandum of Understanding with a third party sulfuric acid marketer regarding an offtake and marketing arrangement for all of the sulfuric acid production from the Blawn Mountain Project. It is anticipated that the marketer will use this production to displace acid currently being delivered into the region from eastern North America and also potentially supply certain mining projects underway that will have a demand for sulfuric acid. PRC offers two key benefits to potential acid consumers; the security of a long-term supply of sulfuric acid and price certainty over the life of a long-term contract.

The markets for SOP and sulfuric acid as well as the product sales prices are discussed in greater detail below in Section 19. Table 1.12 summarizes average selling prices at the plant gate.

Table 1.12 Pricing Summary US$

Pricing	Unit
Average SOP Selling Price – FOB Rail at Plant	$675/ton
Average Sulfuric Acid Selling Price – FOB Rail at Plant	$115/ton

1.11 ENVIRONMENTAL AND PERMITTING

1.11.1 Regulatory Environment

Mining and processing operations in the United States must comply with all applicable federal and state regulations. Mining operations in Utah require compliance with federal as well as state mining and environmental regulations. Utah has primacy over major environmental laws applicable to the project including mining, air and water permitting. The mine and processing plant is located on SITLA-controlled mineral and surface land and is not expected to impact resources with federal oversight and as such it is not anticipated that federal site-specific approvals and permits will be required for the mine and processing plant.

1.11.2 Environmental Setting

Environmental baseline conditions have been assessed for the following resources in regards to permitting and regulatory requirements and to support permit applications:

- Air Quality;
- Archeological resources;
- Wildlife habitat including threatened, endangered, and sensitive species;
- Vegetation including threatened, endangered, and sensitive species;
- Soils;
- Surface and groundwater;
- Wetlands and waters of the US (WoUS).

The results of these surveys and evaluations are discussed in detail in Section 20.

1.11.3 Major Operating Permit and Authorizations

The major permits and approvals that need to be obtained prior to the construction and start-up of the mine and processing plant are provided in Table 1.13.

Table 1.13 Major Required Permits

Major Permits or Approvals	Issuing Agency
Exploration Permit	Utah Division of Oil, Gas and Mining
Large Mine Operation Approval	Utah Division of Oil, Gas and Mining
Water Appropriations	Utah Office of State Engineer
Groundwater Permits	Utah Division of Water Quality
Air Quality Order	Utah Division of Air Quality
General Multi-Sector Industrial Storm Water Permit	Utah Division of Water Quality
Army Corps of Engineers Jurisdictional Waters Concurrence	US Army Corps of Engineers
County Conditional Use Permit and Other Permits	Beaver County

These permits are not meant to be all-inclusive and cover only the major permits required for the mine and processing plant. The mining operation plans have been developed to meet all the regulatory requirements to operate a mine in the State of Utah. The Large Mine Operation and Discharge permits have been granted. Other significant permits including the Air Quality Approval order and Construction permit will be submitted when final design is completed. Permit details are discussed further in Section 20.

1.11.4 Social or Community Impact

A project of this scale represents a significant economic impact to Beaver County and the town of Milford and also to a lesser extent, to adjacent Iron County. Representatives of Beaver County have expressed strong support for the project. Infrastructure and public services in Beaver County and to some degree Iron County will require upgrading and expansion to support the expanded population required for the project. The Utah "School and Institutional Trust Lands Management Act" requires SITLA to manage trust lands to optimize trust land revenues and increase the value of trust land holdings consistent with the balancing of short and long-term interests so that long-term benefits are not lost in an effort to maximize short-term gains and mandates the return of not less than fair market value for the use, sale, or exchange of school and institutional trust assets. The Blawn Mountain Project will assist SITLA in meeting these objectives.

1.12 CAPITAL AND OPERATING COSTS

Capital and operating costs were developed for the following primary basis of operation:

- Whole ore processing (calcining, roasting and leaching);
- Annual ROM ore production rate of 3.4Mtpy;
- Ore cut-off grades of 3.75% K_2O for Area 1 and 3.50% K_2O for Area 2;
- Line power;
- Dry grinding.

Capital costs for the Blawn Mountain Project were developed by several parties. SNC developed capital costs for the processing facilities based on the trade-off and scaling studies mentioned earlier at an approximate level of accuracy of -30% / +30%. SNC's capital estimate for the processing facility totals $411.4M (including 25% contingency). A breakout of this estimate is presented in Section 21 as Table 21.2.

The costs for additional capital items were developed by Millcreek and RDI. Table 1.14 shows capital for project infrastructure, the processing facility, product storage, handling and indirect costs (engineering, construction management, field expenses, etc.). Mining operations will be performed by a contractor and the capital costs associated with the mining equipment are accounted for in the mine operating costs which are discussed later in the report. A contingency of 25% was added to the capital items shown in Table 1.14.

Table 1.14 Total Project Capital Estimate (US$M*)

	Year -3	Year -2	Year -1	Total Construction and Development	Sustaining Capital	Total Life of Project Capital
Project Infrastructure	$0.0	$1.3	$15.2	$16.4	$2.1	$18.5
Processing Plant & Product Handling	$0.0	$109.1	$133.4	$242.5	$47.2	$289.7
Indirect Costs	$12.1	$33.5	$64.6	$110.2	$0.0	$110.2
Contingency	$3.0	$36.0	$50.1	$89.1	$12.3	$101.4
Total	**$15.2**	**$179.8**	**$263.2**	**$458.2**	**$61.6**	**$519.8**

*Rounded

The table above does not include capital costs for the access road, power line, gas pipeline, acid plant, water supply facility or mining equipment. These items are assumed to be provided by either a build-own-operate (BOO) arrangement with a third party or financed through government programs. These items have been accounted for in the operating costs. Table 1.15 (below) presents the capital cost of each item used to formulate the basis for the operating costs. The capital costs are discussed in more detail in Section 21 of this report.

Table 1.15 Third-Party Project Capital (US$M)

	Third-Party Equipment and Infrastructure
Access Road Upgrades	$10.2
Power Transmission Line	$28.5
Natural Gas Transmission Line	$40.0
Acid Plant	$125.0
Water Supply System	$24.4
Initial Mine Capital	$14.7
Total	**$242.9**

1.12.1 Operating Costs

Average annual operating costs for the operation are shown below in Table 1.16. All costs in this report are stated in constant US 2016 dollars, unless otherwise stated.

Table 1.16 Average Annual Plant and Mine Direct Operating Costs (US$)

Direct Plant and Mine Cash Production Cost	Annual Average Cost ($) / Ton SOP	Life of Plant Annual Average (000)*
SOP Tons Sold		230
Sulfuric Acid Tons Sold		524
Mining (Contract Mine Operator Cost)	**$45**	**$10,449**
Processing		
Labor	$57	$13,131
Crushing & Grinding	$22	$4,997
Drying & Calcination	$124	$28,513
Acid Plant (Third-Party)	$42	$9,690
Leaching & Crystallization	$12	$2,755
Drying and Compaction	$2	$415
Steam Plant	$23	$5,381
Water Supply (Third-Party Operator Cost)	$6	$1,487
Other		
Tailings, Pumping, Etc.	$8	$1,862
Access Road & Power Line (Third-Party)	$5	$1,162
Product Handling & Transportation (Third-Party)	$41	$9,499
Credit for Value of Acid	**($251)**	**($57,842)**
Total Direct Operating Cost (Mining and Processing)	**$137**	**$31,498**

The approximate annual operating costs averaged over the life of the project developed for the BOO arrangements described above are presented in Table 1.17 below.

Table 1.17 Third-Party Utility, Infrastructure and Mine Operating Costs

Utilities, Infrastructure and Mining	Third-Party Average Annual Costs (US$Ms)
Access Road and Power transmission line	$1.2
Water, Acid Plant and Natural Gas Line BOO Arrangements	$16.0
Average annual contract mining cost	$10.4
Total	**$27.6**

1.13 ECONOMIC ANALYSIS

The annual average ROM ore requirements are 3.4Mtpy, which produces an average saleable SOP product of roughly 230,000t and 524,000t of sulfuric acid over the life of the project. SOP production varies from 132,400t to 277,000t while sulfuric acid ranges from 288,400t to 626,000t over the 46 year project life. Pre-production cash outflow totals roughly $458M over a three year project execution and construction period. Cashflow turns positive during year 1 and payback occurs during year 5 (2024), which is approximately 8 years after initial investment. Cashflow after payback averages $81M per year for a total net cash flow of $3.3 billion over the project life. The summary of cash flow for the project is presented in Table 1.18 in 2016 dollars and inflated dollars using an inflation rate of 2.0% beginning in Year 2 (2021).

Table 1.18 Project Cash Flow Summary

Project Cash Flow Summary	Life of Plant Annual Average (2016$M)	Life of Plant Annual Average (Inflated $M)
SOP Tons Sold	230	230
Sulfuric Acid Tons Sold	524	524
SOP revenue FOB - Plant	**$155**	**$244**
Acid revenue FOB - Plant	**$60**	**$96**
Total revenue FOB - Plant	**$216**	**$340**
Direct Plant and Mine Cash Production Costs	$89	$132
Royalties	$10	$16
Site G&A	$5	$13
Property Taxes	$3	$6
Total Cash Production Costs	$107	$167
Operating Margin	$108	$172
Income Taxes	$25	$44
Cash Flow from Operations	**$83**	**$128**

Utilizing an inflation rate of 2.0% beginning in Year 2 (2021), the after tax, inflated dollar, internal rate of return for the project is 20.1%. After tax net present values at 8%, 10%, and 12% are shown in Table 1.19.

Table 1.19 Net Present Value Results

Discount Rate	8%	10%	12%
After Tax Net Present Values	$728 million	$482 million	$315 million

The after tax, 2016 constant dollar, internal rate of return for the project is 18.4%. After tax net present values at 8%, 10%, and 12% are shown in Table 1.20.

Table 1.20 Net Present Value Results

Discount Rate	8%	10%	12%
After Tax Net Present Values	$519 million	$341 million	$216 million

1.14 OTHER RELEVANT DATA AND INFORMATION

PRC intends to secure a fixed price engineering, design and construction (EPC) contract during 2017 to facilitate the design and construction of the mine and process facility for the Blawn Mountain Project. The Blawn Mountain Project schedule has been prepared in order to begin plant commissioning in late 2019 / early 2020. A high-level summary of the schedule is provided below:

- Exploration drilling of Areas 1 and 2 in support of prefeasibility studies was completed in August 2013. Additionally, delineation drilling will likely be completed during the summer and fall of 2017.
- Project financing includes timing to cover five major categories for financing through production. Long lead items will be procured during early 2017.
- With the exception of an Air Approval Order application which requires final design emission calculations, major environmental permits (mining and water discharge permits) have already been obtained for the project.
- Environmental permitting activities will span into fourth quarter 2017. Major operating permits and those required to start civil construction are anticipated to be in hand during the first quarter of 2018.
- EPC work begins in late 2017 and is anticipated to be complete by mid-2019
- The mining contractor will be mobilized in 2018 to begin access road upgrade work. Once the access road work is completed, the mining contractor will begin mine development and civil construction work for the processing facility. Mine development and limited mining will begin in 2019. Full production is anticipated in 2021.
- Plant commissioning will occur from the third quarter 2019 through the second quarter of 2020 with full production occurring in fourth quarter 2020.

1.15 CONCLUSIONS AND RECOMMENDATIONS

Based on the results of the PFS, Millcreek and RDI provide the following conclusions and recommendations:

- There are sufficient mineable tons of ore at an average grade of 3.90% K_2O to produce an annual average of approximately 230,000t of SOP over a 46 year project life;
- No fatal flaws have been identified at this stage of project development;
- Pre-production capital costs estimated at $458 million, along with several third-party build, own and operate arrangements will be required to bring this project into production;
- Cash costs of production per ton of SOP, after sulfuric acid credits and before royalties, is estimated at $137;
- Undertake metallurgical test work to investigate the potential for creating a high quality alumina product from the leach residue.

Based on the assumptions defined in this report, the project will generate positive cash flows and achieve an after tax IRR of 20.1% using 2% inflation beginning in year 2 (2021) and an after tax IRR of 18.4% on a constant dollar basis. The overall conclusion is that the results of this study indicate positive economic results and the project should be continued to the next phase of development.

2 INTRODUCTION

Millcreek Mining Group (Millcreek) has prepared this report on the Blawn Mountain Project. This report presents information pertaining to an updated Prefeasibility Study (PFS) performed at the request of Potash Ridge Corporation (PRC). The purpose of this report is to summarize the approach, findings and ultimately the evaluation of the potential economic viability of mining and processing ore from the areas controlled by PRC. PRC controls significant alunite deposits in the Blawn Mountain area of southwestern Utah. Mineral control is directed through a State of Utah Mining Exploration Agreement with Option to Lease (Exploration/Option Agreement), administered through the State of Utah School and Institutional Trust Lands Administration (SITLA) and two state mineral leases also managed by SITLA. The property is located approximately 30 miles southwest of the town of Milford, Utah and 30 miles east of the Nevada state border.

Several firms have contributed to the creation of this document and additionally, several data sources have been utilized during the preparation of this report, including:

- Results of exploration documented in various public reports and on recent drilling campaigns;
- Processing plant design and requirements performed by SNC Lavalin (SNC) and Resource Development Inc. (RDI);
- Material balance information performed by SNC and RDI;
- Marketing information and product sales prices provided by PRC;
- SNC provided new manpower and utility requirements for the processing plant. Millcreek and RDI utilized this information in the development of certain processing plant operating costs.
- PRC's interpretation of the US Federal Tax regulations as they relate to the percentage depletion calculation used in the economic analysis;
- Mine operating and portions of the plant operating costs were primarily derived from the 2015 Western Mine and Mill Cost Estimating guide;

The Blawn Mountain Alunite deposits were explored by Earth Sciences, Inc., (ESI), a mineral exploration and development company headquartered in Denver, Colorado, in the early 1970s and 1980s. ESI, a joint venture partner in The Alumet Company (Alumet), referred to the Blawn Mountain deposits as the NG Alunite deposits. The Blawn Mountain property subsequently came under PRC control in 2011 through the initial Exploration/Option Agreement with SITLA.

Alunite is a complex mineral containing alumina (Al_2O_3), potassium (K_2O), and sulfur (SO_3) all of which have important uses in commercial markets. PRC is pursuing development of the Blawn Mountain Project primarily for the manufacture of sulfate of potash (SOP). However, following initial development, PRC anticipates multiple products from the alunite including sulfur products and potentially alumina.

Alunite is a naturally occurring mineral with the chemical composition of $KAl_3(SO_4)_2(OH)_6$. In its pure state, alunite is comprised of 11.37% K_2O, 36.92% Al_2O_3, 38.66% SO_3, and 13.05% H_2O. Under certain circumstances, sodium will replace a portion of the potassium, altering the alunite to the mineral natroalunite. This is not common in the Blawn Mountain mineral deposit as drill hole cuttings typically assay at less than 1% Na_2O. Iron can replace some of the aluminum, altering the alunite to the mineral jarosite. However, iron does not appear to occur at Blawn Mountain in significant quantities. Alunite occurs worldwide and is associated with hydrothermal alterations accompanying volcanic activity. Alunite can be present in some very large deposits (Hall, 1978) and the western United States contains some of the largest deposits known in the world. The Blawn Mountain deposit is one of these significantly large deposits (Hall, 1978).

Alunite has been mined worldwide for centuries (Hall and Bauer, 1983). Mining of alunite in the United States has historically been utilized for the production of potassium fertilizer. During War World I, alunite was mined in the Mount Baldy mining district in Utah for the production of potash fertilizer. The district was again mined during War World II, the alumina used in the production of aluminum for the war effort. When potassium prices returned to normal levels following the two wars, alunite operations were no longer economically viable in the US primarily due to the size of operations. Alunite has long been known to have value for alumina, potassium and sulfur, though three obstacles have often limited development:

- Adequate size of deposit;
- Concentrations of commercial components;
- Cost of building and operating a processing plant.

The size of most western alunite deposits were not known until the 1970s. Many of the western US alunite deposits are fairly large which makes them a cost-effective source for potassium sulfate as well as a competitive alternative to bauxite for alumina.

Compared with other types of mineral deposits containing similar chemical compounds, alunite contains less potassium than sylvinite, approximately 5% versus 20% to 35% and contains less alumina than bauxite (about 18% compared to 45%). However, alunite can be mined in the US, whereas all of the bauxite used in the US is imported. Sylvinite is more abundant and less expensive to process through solution mining and flotation.

Two of the key factors to the economics of processing alunite are that two valuable products, potash and alumina are produced and that the production of potash from alunite results in the form of potassium sulfate which sells at a premium compared to the more commonly produced potassium chloride from sylvinite. Recent changes in world mineral economics (increased demand for minerals in Asia and higher mineral prices) have led to a re-examination of alternate sources of minerals like alunite. Recent increases in the commodity prices for potash and demand growth for bauxite have led to renewed interest in the Blawn Mountain deposit.

The Blawn Mountain Project contains the four mineralized areas previously defined by ESI and Alumet as the NG Alunite Deposit. In June 2011, PRC acquired a collection of Alumet technical reports and correspondence from a third-party who had received the data as part of another business transaction with ESI.

The authors of this PFS report have visited the project site on several occasions over the last several years. The authors certify that they have supervised the work as described in this report. The report is based on and limited by circumstances and conditions referred to throughout the report and includes information available at the time of this investigation. The authors have exercised reasonable skill, care and diligence to assess the information acquired during the preparation of this report.

The accuracy of resource and reserve estimates is, in part, a function of the quality and quantity of available data and of engineering and geological interpretation and judgment. Given the data available at the time this report was prepared, the estimates presented herein are considered reasonable. However, they should be accepted with the understanding that additional data and analysis available subsequent to the date of the estimates may necessitate revision. These revisions may be material. There is no guarantee that all or any part of the estimated resources or reserves will be recoverable.

3 RELIANCE ON OTHER EXPERTS

Millcreek has prepared this report specifically for PRC. The findings and conclusions are based on information developed by Millcreek and others available at the time of preparation and data supplied by outside sources.

PRC has supplied the appropriate documentation that supports the Exploration/Option Agreement and mineral leases it holds with the State of Utah (SITLA) to be in good standing. The existence of encumbrances to the agreement have not been investigated.

Millcreek relied on RDI to prepare Sections 13, 17 and the portions of Section 1, 21, 22 and 26 related to the mineral processing, metallurgical testing and the processing plant.

Millcreek relied on transportation, marketing information and product sales prices provided by PRC. Millcreek has not independently verified the market information and sales prices. Reliance on this information applies to Sections 19, 21 and 22.

Millcreek relied on SNC's capital estimates for the processing plant that were developed from the trade-off and scaling study work performed by SNC. Additionally, manpower and utility information developed by SNC was used in developing plant operating costs. RDI and PRC also provided information on some operating cost estimates on ancillary plant equipment. Reliance on this information applies to Sections 21 and 22.

Millcreek relied on PRC's interpretation of the US Federal Tax regulations as they relate to the percentage depletion calculation used in the economic analysis. Our reliance on this information applies to Section 22.

4 PROPERTY DESCRIPTION AND LOCATION

The Blawn Mountain Project is located in the southern Wah Wah Mountains of Beaver County, Utah about 180 air miles south-southwest of Salt Lake City, Utah (Figure 4.1). The property is situated west-southwest of Milford (30 air miles to the northeast) and west-northwest of Cedar City (55 air miles to the southeast). The property is located on the Wah Wah South 100,000-scale United States Geological Survey (USGS) topographic map and straddles four 24,000-scale maps: Lamerdorf Peak, Frisco SW, The Tetons, and Blue Mountain. The property occupies Township 29 South, Range 15 West, Sections 13-16, 21-29, 32-36 and Township 30 South, Range 15 West, Section 2 along the Blawn Wash and the Willow Creek drainages that cover most of the historic NG Alunite property.

PRC controls the Blawn Mountain property through an Exploration/Option Agreement (ML 51983.0 OBA) and two mineral leases (ML 52513, and ML 52364), administered through SITLA. The agreements encompass a main tract of land that covers 14,923.72ac and six individual 80ac tracts located 3.5 to 4.5 miles northeast of the main tract of land (Figure 4.2). Table 4.1 provides a legal description of the controlled area. The combined acreage for the three leases is 15,403.72ac. The Exploration/Option Agreement was issued to Utah Alunite, LLC, a 100%-owned entity of PRC. In April 2012, Utah Alunite, LLC was merged into Utah Alunite Corporation (UAC), a 100%-owned entity of PRC and the mineral lease was assigned by Utah Alunite, LLC to UAC.

UAC added five full sections adjacent to the north of ML 51983.0 OBA through the acquisition of SITLA lease ML 52513, effective June 1, 2013. The additional leased area was acquired to expand the amount of non-mineral-bearing property that would allow for siting of process tailings and potential alumina stockpiling. Additionally, the six individual 80ac parcels to the north of the mineral property were acquired from SITLA under lease ML 52364 on January 7, 2013 and are targeted for the development of a water supply to the project.

There are two pre-existing mineral tracts consisting of a 40ac tract (ML 48699.0 MC) along the western edge of the project area and a 155ac tract (ML 48698.0 MC) within the Blawn Mountain Project area. Another mineral tract of 640ac is located approximately one mile east of the Blawn Mountain Project. Remaining lands surrounding the Blawn Mountain Project are predominantly a mix of federal lands administered by the US Bureau of Land Management (BLM) and Utah state lands administered by SITLA.

The Exploration/Option Agreement, ML 51983.0 OBA, is a combination of metalliferous minerals (including sulfur) and potash exploration and an option to a mining lease agreement with the following stipulations:

- Three year lease;
- Bonus payment of $200,000;
- $6ac each year ($69,300yr).

The initial lease terms included a provision to provide a “Positive Prefeasibility Study” to SITLA, documenting the project’s economic viability, possible markets, mining methods and potential environmental issues as well as providing evidence showing the possible continuity and grade of the ore. In May 2013, SITLA provided a letter to PRC stating that the Preliminary Economic Assessment (PEA) completed in November 2012 satisfied the Positive Prefeasibility Study requirement and that PRC could proceed with exercising the option to convert the exploration agreement to a lease.

On March 31, 2014, at the end of the agreement, an additional bonus payment of $1,020,000 was required for issuance of a combined metalliferous minerals and potash lease. The primary term of the lease will be for 10 years, with a provision to extend past the primary term, provided the lessee is either in production of leased minerals or in diligent development of leased minerals. Annual rental rate for a combined mineral lease will be $1ac as required by statute, in addition, a $4ac advanced minimum royalty, which will be increased at $1ac, commencing with the sixth lease year and each lease year thereafter. A combined lease will require a 4% gross royalty for metalliferous minerals and a 5% gross royalty for potash and associated chlorides.

“In June 2015, PRC entered into a modification of the Blawn Mountain Project mining lease agreement with SITLA. The modification cures the event of default under the lease that occurred on March 31, 2015. Under the terms of the modification, SITLA has agreed to forbear from exercising its rights and remedies resulting from PRCs failure to make lease and minimum royalty payments to SITLA under the terms of the lease. The forbearance period is from March 31, 2015 to April 1, 2017. The total amount payable to SITLA, including accrued interest, on April 1, 2017 is approximately $1.2 million.

PRC was obligated to pay accrued and unpaid interest by March 31, 2016 or when it raises US$1.5 million in new funds for the development of the Blawn Mountain project, whichever arises first.

Once PRC raises US$3 million or more of new funds for the development of the Blawn Mountain Project, then all outstanding amounts currently due under the lease, plus accrued interest, will become due.

PRC will pay interest to SITLA on unpaid lease and minimum royalty payments, which will accrue annually at a rate of SITLA’s published prime rate plus two percent (currently equivalent to 5.25%) or 6.0%, whichever is greater, with the first interest payment having been due on March 31, 2016. PRC made the required accrued interest payment to SITLA on March 30, 2016 and August 31, 2016 and is current on all lease obligations."

Mineral leases ML 52513 and ML 52364 are standard metalliferous mineral leases, each with a 10-year primary term and option to extend beyond the primary term. SITLA mineral leases carry an annual rental rate of $1ac, a gross production royalty of 8% on fissionable minerals and 4% on non-fissionable minerals. Advance royalty payments equating to three times the annual rental rate were assessed at the time of issuance.

Table 4.1 Legal Description of SITLA Property

T29S, R14W, SLB&M		**Acres**
Sec. 7	ALL	638.16
T28S, R14W, SLB&M		
Sec. 16	NW1/2NW1/4	80.00
Sec. 21	N1/2NE1/4	80.00
Sec. 22	N1/2NE1/4	80.00
Sec. 26	N1/2NE1/4	80.00
Sec. 27	N1/2NE1/4	80.00
Sec. 35	N1/2NE1/4	80.00
T29S, R15W, SLB&M		
Sec. 1	ALL	731.36
Sec. 2	ALL	725.00
Sec. 11	ALL	640.00
Sec. 12	ALL	640.00
Sec. 13	ALL	640.00
Sec. 14	ALL	640.00
Sec. 15	ALL	640.00
Sec. 16	E ½	320.00
Sec. 21	ALL	640.00
Sec. 22	ALL	640.00
Sec. 23	ALL	640.00
Sec. 24	ALL	640.00
Sec. 25	ALL	640.00
Sec. 26	ALL	640.00
Sec. 27	ALL	640.00
Sec. 28	ALL	640.00
Sec. 29	ALL	640.00
Sec. 32	ALL	640.00
Sec. 33	ALL	640.00
Sec. 34	ALL	640.00
Sec. 35	ALL	640.00
Sec. 36	W½	320.00
T30S, R15W, SLB&M		
Sec. 2	Lots 1(47.38), 2(47.32), 3(47.28), 4(47.22), S½SN½, S½(ALL)	669.20

There are four main zones of mineralization identified by PRC (Figure 4.3). Area 1 is located along a northeast trending ridgeline in the northwest portion of the property. Area 2 is located on another ridgeline, parallel to Area 1 that extends from the center of the property towards the northeast corner. Area 3 is located in the southwest corner of the property and Area 4 is located west of Area 3 and south of Area 2. Both Areas 1 and 2 are the primary focus of this report. Area 1 has been the primary focus of past exploration efforts and continues to be a key area of this report.

The two existing mineral leases located within the PRC exploration agreement area (ML 48698.0 MC and ML 48699.0 MC) are metallic mineral leases that include aluminum but not potash. PRC can explore and delineate potash resources on these leases. PRC is working to secure an agreement for the 155ac section that extends across Area 2 either through an agreement with the lessee or through an adjudication process through SITLA.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Blawn Mountain Project is located about 20 miles west of the Union Pacific (UP) Railroad route, 15 miles south of Highway 21, and 50 miles west of Interstate 15, the main north-south travel corridor through Utah. The area is reached by traveling 24 miles west from Milford on Route 21 and then turning south onto a graveled secondary road and traveling approximately 17 miles. The coordinates for the approximate center of the property are 1,420,000 feet (ft) east and 587,000ft north, Utah State Plane, NAD 27, South Zone. All coordinates given and used in maps and plans are in feet and in the above referenced coordinate system.

5.2 ARCHAEOLOGY

Berge (1974) inventoried the archeological resources of ESI's proposed alunite mine and processing plant sites and located numerous archeological sites but none were determined to be eligible for the National Register (Perry, 1977). An archeological survey of the Blawn Mountain area was completed in the summer of 2013 and is discussed in Section 20.

5.3 CLIMATE

The Blawn Mountain area is semi-arid with hot, dry, sunny summers of low humidity and cold winters. Based on climate data from the closest long-term weather station at Milford (US BLM, 1977), the climate can be described as follows: "Average mean temperatures at Milford, based on 30 years of observation, range from 25.7°F in January to 74.3°F in July. Extremes range from a record low of -34°F to a record high of 105°F. Maximum temperatures in summer frequently exceed 90°F. Cold spells in winter with temperatures below 0°F occur from time to time but seldom last for more than a few days". Temperatures at the Blawn Mountain Project would likely be cooler throughout the year than at Milford because Blawn Mountain is at a higher elevation. Average annual precipitation at Milford is 8.4in, with the wettest month being March and the driest month being July. Snow does not generally persist in the valleys but can blanket the mountains through the winter season (US BLM, 1977).

5.4 ENERGY CORRIDORS

Two energy corridors pass to the east of the Blawn Mountain Project both of which trend roughly north-south, as shown in Figure 5.1. The first, located 22 miles east of the property, contains the Utah Nevada (UNEV) gas pipeline, the Intermountain Power Project electric transmission line, and the federally designated, multimodal West-wide Energy Corridor (US Department of Energy, 2011). The second located approximately 25 miles east of the property contains the Kern River gas pipeline. The West-wide Energy Corridor follows State Highway 21 west from Milford towards Nevada, 17 miles north of the Blawn Mountain Property (US BLM, 2011[a]).

5.5 GRAZING

A grazing allotment map (US BLM, 2011[b]) shows the boundaries of cattle and sheep grazing allotments and the boundaries of wild horse Herd Management Areas (HMA) on the federal lands surrounding the Blawn Mountain Project. The entire Blawn Mountain Project is within grazing allotments administered by SITLA. The project area is not within an HMA but the four mile HMA adjoins the south boundary of the Blawn Mountain Project and covers more than 100 square miles.

5.6 LOCAL RESOURCES

Construction of a mining operation and processing plant at Blawn Mountain would require local resources of contractors, construction materials, employees and housing for employees and energy resources. The Milford area offers construction materials such as sand and gravel from several sources: crushed limestone from the Graymont Lime Plant in the Cricket Mountains north of Milford, crushed stone from a railroad ballast quarry just north of Milford and Portland cement from the Ashgrove Cement West Plant at Leamington, approximately 90 miles away. The nearby towns of Delta, Milford, Fillmore, Cedar City and Beaver could provide mine and plant workers and furnish housing for company employees. There are two nearby electrical corridors and there is sufficient electricity being supplied within the region from coal, geothermal, solar and wind power plants.

5.7 PHYSIOGRAPHY

Topographically, the Blawn Mountain Project is situated in a typical Basin and Range setting. The ranges, consisting of north-south trending mountains, are generally steep and rugged with mountaintop elevations up to 7,900ft above sea level. The ranges are separated by fault graben basins with deeply incised drainages. Pine Valley lies to the west of the Wah Wah Range and Wah Wah Valley lies to the east. The Blawn Mountain deposits occupy three of the smaller ridges in the southern Wah Wah Range. The mineral tracts include substantial low relief areas that have potential to support mine and plant facilities.

Seasonal runoff is channeled away from the Blawn Mountain Alunite deposits by two main drainages: Blawn Wash drainage carries runoff to the southeast toward Escalante Valley and Willow Creek drainage carries runoff into Wah Wah Valley to the northeast.

5.8 SEISMOLOGY

The Blawn Mountain Project area has low potential for occasional moderate earthquakes. Perry (1977) discussed the possibility of weak earthquakes in the Blawn Mountain area due to its proximity to the transition zone between the Colorado Plateau and Basin and Range physiographic provinces, an area termed the Intermountain Seismic Belt. Perry also mentions “a non-instrumented report of an earthquake with a modified Mercalli Intensity of III (nominally Richter 3.1), recorded October 26, 1885 between 0800 and 0900 hours near Frisco, about 12 miles northeast of the project area”. Pankow, Arabasz, and Berlacu (2009) refined the seismic history of the region and delineated an area of mildly anomalous seismic activity in the Escalante Valley. The most significant earthquake that is discussed for the area is the 1908 Milford earthquake of local Richter Scale magnitude (ML) 5±.

5.9 SURFACE OWNERSHIP

The Blawn Mountain Project is composed of Utah State-owned land managed by SITLA. The lands immediately around the property are predominantly federal lands managed by the BLM along with additional SITLA tracts.

5.10 TRANSPORTATION

The Blawn Mountain Project is accessed by secondary roads maintained by Beaver County and located near highway and rail transportation. State Highway 21 passes 17 miles to the north of the property, connecting Milford, Utah with Ely, Nevada to the northwest. State Highways SR-21 and SR-130 pass about 30 miles east of the property connecting Milford, Utah to Cedar City, Utah to the south. I-15 is located approximately 63 miles to the east-southeast and accessed via SR-21 and SR-130. The Union Pacific Railroad route connecting Salt Lake City, Utah to Las Vegas, Nevada passes approximately 20 miles to the east of the Blawn Mountain Project.

5.11 VEGETATION

The Blawn Mountain Project is located in the pinyon-juniper community as defined by the BLM (1977). This flora community is characterized by occurrence of Utah Juniper, single-leaf and double-leaf Pinyon Pine. Occasional patches of Mountain Mahogany, Gamble Oak, Ponderosa Pine, and Aspen occur at higher elevations with greater rainfall amounts. The valleys of the area have been extensively chained to remove Juniper and Pinyon and improve grass growth for grazing.

Vegetation in the valleys is a mixed shrub-grass community characterized by seven shrubs: Big Sagebrush, Black Sagebrush, Big Rabbitbrush, Small Rabbitbrush, Greasewood, Winterfat, and Matchweed. Galleta, Indian Ricegrass and Cheatgrass are the most common grasses across the property. A survey of the Blawn Mountain Project completed in 2013 did not identify any federally protected, threatened, or endangered (T&E) species or potential habitat. The results of this survey are discussed further in Section 20.

5.12 WATER RESOURCES

5.12.1 Surface Water

The Blawn Mountain Project area is in an arid portion of the state. Located at the headwaters of two drainages, one flows to the Wah Wah Valley and one flows to the Escalante drainage. Surface water flows are ephemeral. Runoff events from the project site are short-lived. Generally, these drainages ultimately discharge to salt lakes or playas without an outlet other than evaporation.

Discharges to the south from the project area flow into the Escalante Valley. Most of these flows infiltrate into the groundwater system. However, only a small percentage of flows from larger duration storms reach the main drainage channel of the valley. Limited surface water is available for water rights in the valley.

Flows into the Wah Wah Valley are collected first in the Wah Wah Valley Hardpan, which occupies the lower (northern) end of the Wah Wah Valley and then if there are excess flows, the discharge flows north to Sevier Lake. A few shallow stock ponds along the flanks of the Wah Wah Valley have water rights to capture periodic runoff.

Similar conditions exist in the Pine Valley located to the west of the Wah Wah Mountains. Discharges from the upper Pine Valley are collected in the playa in lower Pine Valley located to the west of the Wah Wah Mountains, the playa is referred to as the Pine Hardpan.

Sevier Lake and Pine Hardpan hold large quantities of poor quality water contained in deep evaporite and clay deposits and have occasional standing water. PRC completed a surface water monitoring program during the fall of 2013 to document the site-specific flow conditions and water quality.

5.12.2 Groundwater

The project area is on the east edge of the Basin and Range Province in which faults divide uplands from sediment-filled valleys (horst and graben structure). Deep bedrock consists of crystalline rocks (gneiss) overlain by Paleozoic carbonate rocks (limestone and dolomite). Carbonates and gneiss are commonly intermingled by faulting. Lower Tertiary volcanics, mostly tuff and andesite, cap the highlands. Quaternary alluvium filled the down-faulted valleys to depths of several hundred feet as they deepened.

The Blawn Mountain Project area has no perennial streams, indicating that near-surface groundwater in the project area is limited. PRC commissioned a spring and seep survey in the spring of 2013 to assess the occurrence of water sources in the project area. This study covered about 20 square miles and assessed surface and groundwater flows. A total of 50 spring and seep sites were identified (see Figure 5.2). Many of these sites did not have flowing water but were either damp spots, salt stain/accumulations on the surface, or phreatophytic vegetation areas. A limited number of water sources were identified which physically had water with flows ranging from 0 to 1.4 gallons per minute (gpm). For these locations, field water quality samples were collected and analyzed for pH, specific conductance, total dissolved solids (estimated), and temperature. Table 5.1 presents a summary of the data from the water sources that had water.

Table 5.1 Summary of Flows and Field Water Quality Information

Point ID	Spring Data					Fall Data				
	pH	Temp (˚C)*	SC (mS)**	TDS (ppt)***	Q (gpm)****	pH	Temp (˚C)*	SC (mS)**	TDS (ppt)***	Q (gpm)****
BWSU 0	7.55	11.60	2.17	1.12	1.41	7.96	15.20	1.82	0.90	0.79
BWSU 1	7.72	14.70	2.13	1.06	0.51	7.48	15.90	2.28	1.14	0.55
BWSU 2	7.34	14.70	2.04	1.02	1.17	7.51	14.30	1.96	0.98	0.75
BWSL0	N/A	N/A	N/A	N/A	DRY	7.41	18.00	2.11	1.05	0.04
BWSL1	N/A	N/A	N/A	N/A	DRY	7.10	15.20	2.43	1.22	0.35
BWSL5	N/A	N/A	N/A	N/A	DRY	6.87	16.00	1.96	0.98	0.99
BWT 0	7.61	20.50	3.50	1.75	1.06	7.52	16.20	3.29	1.64	0.16
RD SPRING	7.78	15.50	3.62	1.81	0.10	N/A	N/A	N/A	N/A	DRY
SEEP 02	7.09	21.00	3.54	1.79	0.00	7.57	14.90	3.40	1.70	0.36
SEEP 03	N/A	N/A	N/A	N/A	DRY	8.38	14.80	4.04	2.00	<0.01
SEEP 04	6.86	17.40	11.36	5.65	0.00	N/A	N/A	N/A	N/A	DRY
SEEP 07	7.41	14.90	3.62	1.81	0.00	N/A	N/A	N/A	N/A	DRY
SEEP 08	7.19	14.30	5.17	2.56	0.00	N/A	N/A	N/A	N/A	DRY
SEEP 10	N/A	N/A	N/A	N/A	DRY	7.68	15.50	1.89	0.94	0.06
SEEP 15	7.59	26.50	4.65	2.12	0.00	N/A	N/A	N/A	N/A	DRY
WS 0	7.10	11.50	1.97	0.90	0.59	7.04	19.10	1.61	0.80	0.00
WS 1	7.34	15.90	3.63	1.66	0.00	7.52	23.10	1.58	0.79	<0.01
WS 2	7.04	10.30	4.37	1.95	0.58	7.11	17.80	1.81	0.90	0.90
WS MAIN 1	7.98	16.20	2.62	1.21	1.34	8.04	24.70	1.96	0.98	0.90
WS MAIN 2	7.96	15.70	2.90	1.33	1.34	8.22	22.80	1.99	0.98	0.90
WS MAIN 3	7.88	19.00	3.84	1.75	0.00	7.75	22.70	1.99	0.99	<0.01
Water Well	N/A	N/A	N/A	N/A	DRY	7.43	20.40	0.58	0.29	N/A

*Temperature (Temp): degrees Celsius (˚C) **SC: Specific Conductance in milli-Siemens (mS)
***TDS: Total Dissolved Solids (estimated) parts per thousand (ppt) ****Q : Discharge in gallons per minute (gpm)

To assess the potential for groundwater issues during mining, 10 monitoring wells have been installed in the mine area by PRC. The wells are located in the valleys surrounding the project area. The zones monitored are the volcanic tuffs and andesite flows. Two of the wells were dry, the remainder encountered water. Water levels encountered in these wells are close to the surface. Flow to the wells is limited. For the wells completed in the andesite flows, the flows are in the range of 5 to 10gpm. For the wells in the volcanic tuffs, the flows are in the range of 0 to 0.5gpm.

Groundwater hydraulic parameters were gathered for eight of these wells. Six of the wells in the andesite flows were evaluated for both pumping and recovering water levels. Due to the relatively slow response of the two wells in the volcanic tuff, the analyses for these wells were limited to using recovery water levels in the wells following pumping. The transmissivity values determined from these tests are presented in Table 5.2.

Table 5.2 Summary of Aquifer Parameters

Well ID	Lithology	Transmissivity (ft^2/d)
MW-1	Andesite	15.56
MW-2	Andesite	DRY
MW-3	Tuff/Clay	2.57
MW-4	Andesite	4.74
MW-5	Tuff/Clay	6.47
MW-6	Andesite	0.13
MW-9	Andesite	1.59
MW-10	Andesite	0.72
MW-11	Andesite	DRY
MW-13	Andesite	7.31

These wells show that groundwater is limited and no extensive groundwater source is found in these zones. Water to support mining and processing will need to be produced from deeper groundwater in the adjacent areas. Three water bearing formations can be described:

- Alluvium, up to hundreds of feet thick and from the Pliocene to recent in age;
- Tertiary volcanic rocks capping the highlands in the project area;
- Carbonate rocks underlying and fault-splintered into basement gneisses.

The region is described in the hydrology literature as part of the Sevier Desert and the Great Basin, with groundwater systems dominated by coupled alluvium and carbonate rocks. Alluvium is described generally as containing some fresher water in valley sides and heads and salty water in closed valley playas. Clays usually reduce permeability factors but high yields can be obtained from deeper wells near zones where coarser sediments exist.

Volcanic rocks include tuffs and andesite flows capping highland areas; the Blawn Mountain Project's economic deposits occur in altered volcanic rocks. Volcanic rocks capping ridges receive higher recharge than the regional average due to higher elevation. Groundwater in the area shows higher salt content than expected and feeds springs at the base of these rocks and in gulches incised into them. These springs are generally documented as water-rights held for stock and wildlife, have flows less than a few gpm and some of them may be seasonal.

Carbonates include limestones and dolomites which have variable permeability and may have yields associated with solution channels. These strata typically have poorer water quality. Regionally, the carbonate strata constitutes a slow-flowing, saline aquifer which discharges to some low basins such as Sevier Lake and also on longer paths to Great Salt Lake. One potential target for water supply assessment would be the small areas of limestone located within highly faulted volcanics near the project site.

United States Geologic Survey (USGS) studies indicate substantial groundwater resources are present in the Wah Wah and nearby Pine Valley drainages (Tables 5.3 and 5.4). Stephens (1974 and 1976) evaluated the hydrology of both the Wah Wah and Pine Valleys. Based on his findings, the water balances for the Wah Wah and Pine Valleys indicate:

Table 5.3 Water Balance Wah Wah Valley

Description	Source	Estimated Quantity (acft/yr)
	Recharge	
	From precipitation in drainage basin	7,000
	Subsurface inflow from Pine Valley	3,000
	Total	**10,000**
	Discharge	
Evapotranspiration from	Stream-channel alluvium	40
	Wah Wah Springs	600
Flow and pumpage from wells and springs	Stream-channel alluvium	50
	Older alluvium	2
	Extrusive rocks	24
	Intrusive rocks	24
	Quartzite and metasedimentary rocks	10
	Wah Wah Springs	800
	Total	**1,500**

Table 5.4 Water Balance for Pine Valley

Description	Source	Estimated Quantity (acft/yr)
	Recharge	
	From precipitation in drainage basin	21,000
	Total	21,000
	Discharge	
Evapotranspiration from	Stream-channel alluvium	**5,500**
Flow and pumpage from wells and springs	Man usage	5
	Pine Valley Springs	650
	Subsurface outflow to Wah Wah Valley	3,000
	Total	**9,155**

Studies by ESI, conducted in the 1970s, demonstrated that the alluvial fill in the southern portion of the Wah Wah Valley was very productive and able to supply a majority of the water resources for that project. Pumping rates ranged from 875gpm with a stabilized drawdown of 93.5ft to 1353gpm with a drawdown of 113ft. Following pumping, the water level recovered within 14 minutes to within 3.5ft of the pre-pumping static water level. PRC continues to evaluate these resources and has secured water rights to support the Blawn Mountain Project. The USGS has also studied these water resources and is in the process of finalizing a report to describe water resources of the area.

5.13 WILDERNESS DESIGNATION

The Blawn Mountain Project has not been designated for study or inclusion for wilderness. In 1999 the BLM re-inventoried its lands for suitability for classification of US wilderness designation. Part of the Wah Wah Range north of the Blawn Mountain Project, the Central Wah Wah Mountains, met the wilderness re-inventory criteria (US BLM, 2011[c]). The southern boundary of the re-inventoried Central Wah Wah wilderness area is approximately five miles north of the northern border of the Blawn Mountain Project.

5.14 WILDLIFE

Deer, wild horses, antelope, cougars, raptors and other birds, coyotes, bobcats, and foxes are all common animals in the area. A survey of the Blawn Mountain Project completed in 2013 did not identify any (threatened and endangered) T&E species or potential habitat. The results of this survey are discussed further in Section 20. A BLM map of wildlife management areas (US BLM, 2011[d]) shows no special management areas within the Blawn Mountain Project area.

5.15 CONFLICTING DEVELOPMENT

The Blawn Mountain Project area has a long history of mineral exploration, grazing and outdoor recreation. No historical land-use conflicts are known for the property and if the property is developed for mineral extraction, no future land-use conflicts are anticipated. Recently, southwest Utah has experienced extensive conventional energy, alternative energy and energy infrastructure development in the vicinity of the Blawn Mountain Project area. The Cedar City Field Office of the BLM compiled a draft map for their Resource Management Plan (US BLM, 2011[e]) indicating that tracts with geothermal, wind power, and solar energy potential would not conflict with Blawn Mountain Project's development.

6 HISTORY

The extensive hydrothermal alteration of the southern Wah Wah Range has long been known and most of the prospecting in the area has been for metallic minerals associated with the hydrothermal alteration. Whelan (1965) was the first known geological investigation to discuss production of the Blawn Mountain Project's alunite as a commodity. In the early 1970s, ESI was simultaneously investigating deposits in Colorado, Arizona, Nevada, California and several deposits in Utah. In 1970 ESI started the first systematic exploration of Blawn Mountain, which they called the NG Alunite property. Results were encouraging. That same year ESI entered into a joint venture agreement with National Steel Corp. of Pittsburgh, Pennsylvania and the Southwire Company of Carrollton, Georgia to open an alunite mine as a source of alumina to supply the National Steel/Southwire's jointly-owned aluminum plant at Hawesville, Kentucky (Parkinson, 1974). The partnership was called The Alumet Company and was headquartered in Golden, Colorado. ESI owned 50% of the partnership and National and Southwire each owned 25%.

The NG Alunite deposit is a circular cluster of four alunite areas (Figure 6.1). These four areas were mapped, surface sampled and drilled. While ESI continued investigating other deposits, they focused most of their resources on the NG Alunite deposits. Initial results convinced ESI to further focus development on Area C, now referred to as Area 1, with the intention of investigating it as their first mine site. Additional surface sampling, drilling, and collection of bulk samples (for pilot plant testing) at Area 1, were completed before April 1974. Seven test pits (ESI, 1989) were excavated in the north end of Area 1 for samples to send to the Alumet pilot plant in Golden, Colorado. The largest sample was a 3,000t (Krahulec, 2007) sample from a pit identified as Number 5. The pilot plant (designed by HRI) had the capacity to process 12 to 18 tons per day (tpd) and operated for three years with occasional shutdowns to modify the process (ESI, 1989).

Alumet's concept was to build an integrated plant that would produce 500,000 tons per year (tpy) of alumina with by-products of 450,000tpy of sulfuric acid, 250,000tpy SOP, and aluminum fluoride (Parkinson, 1974). To achieve this level of production, Alumet planned to mine 4 million tons (Mt) of alunite per year for 25 years (Perry, 1977). Alumet acquired subsidiary mining properties and resources needed to support the alunite plant. Alumet acquired a phosphate property near Soda Springs, Idaho. Phosphate was to be mined and calcined in Idaho and shipped to the NG Alunite Plant where the by-product sulfuric acid would be used to make phosphate fertilizer.

The Soda Springs, Idaho phosphate mine was also intended to produce by-product vanadium (Parkinson, 1974). Alumet also acquired a coal property on the Wasatch Plateau, to the northeast in central Utah, to provide fuel for the alunite plant. Local water rights were acquired and water wells were drilled and tested. Local aggregate sources were evaluated for use in construction of the plant.

During this time, Alumet refined their resource calculations, commissioned feasibility and environmental studies, continued improving their metallurgical process and commissioned design of an open-pit mine on the northeast end of Area C with a plant and tailings pond adjacent to the northeast (Figure 6.2). Despite this advanced stage of development, plant construction and mining never occurred due to a challenging US economic environment in the 1980s and depressed pricing for alumina and potash.

Previous resource estimates are difficult to relate to the current assessment. Historical estimates centered on alumina as the primary product with potash as a secondary product. Cut-off grades were based on Al_2O_3 grades versus K_2O. (Previous reserve estimates for Area 1 are summarized in Table 6.1.) ESI initially carried out resource estimates in 1972 to include Areas 1 to 4. Chapman, Wood, and Griswold, Ltd. (CW&G) were retained to calculate a corresponding estimate. Pincock, Allen and Holt and Computer Associates, Inc. (PAH/CAI) calculated the resources for the north end of Area 1 in 1975. None of these studies are deemed to be NI 43-101 compliant even though reasonable methodologies were applied at the time.

Table 6.2 presents historical resource and reserve estimates for all four areas that were part of the NG Alunite project. Previous resource estimates did not specify potassium grades. Table 6.3 provides calculated K_2O and K_2SO_4 contents based on Al_2O_3 contents for the historical estimates in Table 6.2. In recent analytical work completed by PRC in a validation drilling program, a direct linear correlation was observed between K_2O and Al_2O_3 values. Based on this correlation, a multiplier of 0.2809 is applied to Al_2O_3 to derive K_2O content. Potassium sulfate, K_2SO_4, is calculated from K_2O using a factor of 1.8493.

Table 6.1 Area 1 Historical Reserve Estimates

Category	ESI		CW&G		PAH/CAI	
	Tons	%Al_2O_3	Tons	%Al_2O_3	Tons	%Al_2O_3
Proven	119,900,000	14.3	89,000,000	13	129,400,000	14
Probable	22,700,000	12.8	62,800,000	13.2	17,700,000	14.8
Inferred	36,100,000	14.1	Not estimated		18,015,000	17.1
Total	**178,700,000**	**14.1**	**151,800,000**	**13.1**	**165,185,000**	**14.4**

Table 6.2 Historical Resource and Reserve Estimates for Blawn Mountain

Deposit	Ore (000 Tons)	Alunite (%)	Al_2O_3 (%)	Al_2O_3 (000 Tons)	Inventory Classification	Reference
Area 1	129,400	38.3	14.00	18,155	Proven	Couzens, 1975
Area 1	17,770	40.3	14.80	2,626	Probable	Couzens, 1975
Area 1	18,015	46.7	17.10	3,079	Inferred	Couzens, 1975
Area 1	165,185	39.4	14.50	23,869	Total	Couzens, 1975
Area 2	54,400	38.5	14.30	7,779	Indicated	Walker, 1972
Area 2	124,900	39.5	14.60	18,235	Inferred	Walker, 1972
Area 2	25,900	41.5	15.30	3,963	High-Grade Indicated	Walker, 1972
Area 2	179,300	39	14.50	25,999	Total	Walker, 1972
Area 3	11,600	44	16.20	1,879	Indicated	Walker, 1972
Area 3	281,400	44	16.20	45,587	Inferred	Walker, 1972
Area 3	7,300	47	17.30	1,263	High-Grade Indicated	Walker, 1972
Area 3	293,000	44	16.20	47,466	Total	Walker, 1972
Area 4	51,700	36.5	13.50	6,980	Indicated	Walker, 1972
Area 4	49,200	38	14.10	6,937	Inferred	Walker, 1972
Area 4	100,900	37	13.80	13,924	Total	Walker, 1972
Total	**738,385**	**41.1**	**15.0**	**111,175**	**Grand Total**	

Table 6.3 Calculated K2O and K2SO4 Conc. - Historical Resource and Reserve Estimates

Deposit	Ore (000 Tons)	K_2O* (%)	K_2O (000 Tons)	K_2SO_4* (%)	K_2SO_4 (000 Tons)	Inventory Classification	Reference
Area 1	129,400	3.98	5,147	7.36	9,518	Proven	Couzens, 1975
Area 1	17,770	4.20	747	7.78	1,382	Probable	Couzens, 1975
Area 1	18,015	4.86	875	8.98	1,618	Inferred	Couzens, 1975
Area 1	165,185	4.12	6,804	8.98	14,840	Total	Couzens, 1975
Area 2	54,400	4.06	2,210	7.51	4,087	Indicated	Walker, 1972
Area 2	124,900	4.15	5,181	7.67	9,580	Inferred	Walker, 1972
Area 2	25,900	4.35	1,126	8.04	2,082	High-Grade Indicated	Walker, 1972
Area 2	179,300	4.12	7,386	8.04	14,412	Total	Walker, 1972
Area 3	11,600	4.60	534	8.51	987	Indicated	Walker, 1972
Area 3	281,400	4.60	12,951	8.51	23,950	Inferred	Walker, 1972
Area 3	7,300	4.91	359	9.09	663	High-Grade Indicated	Walker, 1972
Area 3	293,000	4.60	13,485	9.09	26,630	Total	Walker, 1972
Area 4	51,700	3.84	1,983	7.09	3,667	Indicated	Walker, 1972
Area 4	49,200	4.01	1,971	7.41	3,645	Inferred	Walker, 1972
Area 4	100,900	3.92	3,956	7.41	7,474	Total	Walker, 1972
Total	**38,385**	**4.29**	**31,675**	**7.93**	**58,577**		

*Calculated from Equivalent Al_2O_3 Concentrations, $3.52K_2O$=>Al_2O_3; $1.8493K_2O$=>K_2SO_4

In 2011, PRC initiated a validation drilling program in Area 1 primarily to validate the previous exploration efforts. A total of 34 holes were completed in Area 1 between October 2011 and February 2012. The drill holes were twinned to locations of previous ESI drill holes using coring and reverse circulation methods.

The 2012 Technical Report concluded that the PRC validation drilling program had adequately tested the Area 1 deposit, both spatially and in the number of twinned drilling locations. A three-dimensional geological block model (3DGBM) was constructed for Area 1 using historic and PRC validation drilling data. The 3DGBM was subsequently used for reporting of resources for Area 1 in accordance with CIM Standards on Mineral Resources and Reserves, effective date April 16, 2012 (Millcreek, April 2012). At a 1% K_2O cut-off grade, the combined measured resource plus indicated resource for Area 1 was estimated to be 162Mt carrying an average grade of 3.23% K_2O and 13.90% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade was estimated to be 5.98%. As of April 16, 2012, approximately 66% of the identified resource was classified as measured resource and 34% as indicated resource.

Between July and September, 2012, PRC completed a 90-hole drilling campaign. 18 holes were completed as infill holes on Area 1, 20 holes were drilled to test mineralization extending along the ridge southwest of Area 1, 50 holes were drilled on Area 2 to delineate potential resources, and two holes were drilled in Area 4. The results from the drilling program provided additional information to refine resource estimates on Area 1 and provided sufficient geologic and analytical data to support a resource estimate for Area 2. Resource estimates were included in a Preliminary Economic Assessment (PEA) issued by PRC in April, 2012. The measured resources plus indicated historic resources and average grades reported in the PEA, using a 1% K_2O cut-off grade, are outlined in Table 6.4.

Table 6.4 Prior Resource Estimate (November, 2012)

Area	IN Situ (Mt)	K_2O (%)	K_2SO_4 (%)	AL_2O_3 (%)
Area 1	156.3	3.22	5.96	13.90
Area 2	464.4	3.07	5.68	13.16

An open-pit conventional truck/shovel method was identified as the preferred mining method for Area 1 and Area 2 in the PEA (2012). Using a target K_2SO_4 requirement of 750,000tpy, a total of 121.6Mt ROM alunite in Area 1 and 387.9Mt of ROM alunite in Area 2 were identified as potentially extractable over a 30 year LOM.

Between January and February of 2013, a third reverse circulation drilling program was undertaken, including two holes on Area 1 and 17 holes on Area 2. The goal of these holes was to further improve resource delineation and geologic assurance.

In addition to exploration drilling, PRC completed 10 monitor wells in the alluvial/colluvial areas surrounding the alunite deposits and completed three test holes on ML 52364 to determine the groundwater potential of the Wah Wah Valley. Two holes were converted to monitor wells to further assess future development of a wellfield.

In December, 2013, a PFS was completed on the Blawn Mountain Project for PRC. Resources were once again updated to reflect the latest drilling along with mine-planning, metallurgy, processing design, infrastructure requirements and environmental assessments. Highlights from the 2013 PFS include:

- 425Mt of Proven and Probable Reserves;
- Average grade of 3.51% K_2O (6.49% K_2SO_4;
- Mine production of 10.6Mtpy after 3 year ramp-up to full production;
- 24 years of active mining plus 16 years of stockpiling for 40-year supply of ore;
- 645,000tpy production of SOP;
- 1.4Mtpy production of sulfuric acid.

In August, 2014, PRC secured its Large Mining Permit for Blawn Mountain from the Utah Division of Oil, Gas and Mining. The Large Mining Permit is the primary permit required in Utah for project development.

There has been no known production of alunite or any other mineral resource from the Blawn Mountain Property.

7 GEOLOGICAL SETTING AND MINERALIZATION

The Blawn Mountain Project is located in the southern Wah Wah Mountains of the eastern Basin and Range province in an area characterized by a thick Paleozoic sedimentary section that was:

- Thrust faulted during the Sevier orogeny;
- Buried under a thick layer of regionally distributed Oligocene volcanic rocks and locally derived volcanic rocks;
- Extended to the west by the Basin and Range event;
- Altered by H_2S rich hydrothermal alteration related to a postulated shallow laccolithic intrusive which domed, and altered the overlying calc-alkaline volcanic rock (Hofstra, 1984);
- Affected by continual erosion of the ranges contributing to colluvial and alluvial deposition in the valleys.

Blawn Mountain is located along the Blue Ribbon lineament (Rowley and others, 1978) within the Pioche mineral belt (Shawe and Stewart, 1976), a tectonic, structural, and igneous zone that contains a large number of metallic mineral mining districts with almost two dozen associated alunite vein and replacement deposits.

Figure 7.1 shows a diagrammatic cross-section through the Wah Wah Range centered on Blawn Wash and Figure 7.2 presents a diagrammatic cross-section through Area 1 at Blawn Mountain.

7.1 REGIONAL STRATIGRAPHY

Regional rock strata underlying the Wah Wah and Blawn Mountain areas are Proterozoic to Cenozoic Era in geologic age. Rock strata consist of varying types of volcanic tuffs, rhyolites, mafic flows, basalts, quartzites, limestones, dolomites, sandstones and shales. Also present are brecciated zones associated with volcanic and faulting activity.

The sedimentary and volcanic stratigraphy of the region is summarized in Table 7.1.

Table 7.1 Regional Stratigraphy

Eras	Periods	Epochs	Groups	Formations	Members
Cenozoic	Quaternary				Alluvium And Colluvium
	Tertiary	Pliocene		Steamboat Mountain	Basalt
		Miocene	Quichapa	Blawn	Rhyolite
					Tuff
					Bauers Tuff
					Mafic Flow
					Garnet Tuff
		Oligocene		Isom	Bald Hills Tuff
				Bullion Canyon Volcanics	Three Forks Tuff
			Needles Range	Lund	
				Wah Wah Springs	
				Cottonwood Wash Tuff	
				Escalante Desert	
			Tuff Of Towers Point, Volcanic Breccia		Conglomerate
		Paleocene - Eocene		Claron	
Mesozoic	Jurassic			Temple Cap	
				Navajo Sandstone	
	Triassic			Chinle	Petrified Forest
					Shinarump
				Moenkopi	
Paleozoic	Permian			Gerster Limestone	
				Pympton Limestone	
			Oquirrh	Kaibab Limestone	
				Ely Limestone	
	Pennsylvanian			Callville Limestone	
				Woodman	
	Mississippian			Gardison Limestone	
				Fitchville	
	Devonian			Pinyon Peak Limestone	
				Simonson Dolomite	
				Sevy Dolomite	
				Laketowm Dolomite	
	Silurian			Ely Springs Dolomite	
	Ordovician			Eureks Quartzite	
				Kanosh Shale	
				Juab Limestone	
				Wah Wah Limestone	
				Fillmore Limestone	
				House Limestone	
				Notch Peak	
	Cambrian			Orr	
				Wah Wah Summit	
				Trippe Limestone	
				Pierson Cove	
				Eye Of Needle Limestone	
				Swasey Limestone	
				Whirlwind	
				Dome Limestone	
				Peasley Limestone	
				Chisholm Shale	
				Howell Limestone	
				Pioche	
				Prospect Mountain Quarzite	
Protero zoic	Precambrian			Mutual	

7.2 ALUNITE OCCURRENCES

Hofstra (1984) postulates the presence of a relatively shallow laccolithic intrusion as the source of the hydrothermal fluids that created the alunite deposits, based on radial doming of the extrusive Miocene and Oligocene volcanic strata over an area of 6 miles north-south and 3 miles east-west. The laccolith may have intruded along a zone of weakness such as the Blue Mountain thrust. The high temperature H_2S rich fluid associated with the laccolith rose along the fracture zones created in the overlaying strata by the intrusion. The fluid then penetrated into the Miocene and Oligocene volcanic layers where it encountered and boiled the groundwater. With the presence of oxygen that was transported in the groundwater, the H_2S was oxidized into super-heated aqueous solutions of H_2SO_4 and the resulting solution altered the volcanic rock along fracture zones associated with normal faulting and in zones of higher porosity/permeability. The more porous the fracture zones and strata, the more mineralization occurred. The alunite alteration has been K-Ar age dated at 22.5 million years ago (Hofstra, 1984).

7.3 STRUCTURAL GEOLOGY

The Blawn Mountain Project lies within the eastern Basin and Range province. During the Late Cretaceous Sevier orogeny the region was subjected to thrust faulting and folding. Major thrust faults are the Wah Wah, Teton, Dry Canyon and Blue Mountain. The Wah Wah thrust emplaced upper Proterozoic and overlying Cambrian strata over Ordovician to Pennsylvanian strata. The Teton thrust emplaced Ordovician and Silurian strata over Silurian and Devonian carbonates and the Dry Canyon thrust emplaced Silurian and Devonian carbonates over Pennsylvanian and Mississippian strata. The Blue Mountain thrust emplaced Cambrian and younger age carbonates over Jurassic strata.

Regionally there are four sets of normal faults that relate to Basin and Range block faulting. These faults generally trend west-northwest, northeast, northwest and north-south. The Blawn Wash area is a graben bounded by west-northwest and northeast faults and the bounding volcanic ridges that host the alunite mineralization.

Within the project area are several minor normal faults that offset the alunite deposits. Figure 7.3 depicts the location of these local normal faults as well as the mapped surface geology.

7.4 PROPERTY GEOLOGY

The Wah Wah Range is partly composed of a thick section of marine, Paleozoic and Triassic quartzites and carbonates (Miller, 1966) deposited in the miogeocline of the western continental shelf. This area was covered by ocean until the Jurassic Period when it was uplifted during the Sonoma orogeny. The first major deformation of this area was during the Cretaceous/Tertiary Sevier orogeny which thrusted older basement rocks over younger rocks along both the Wah Wah and Blue Mountain thrusts, contributing to the folding of the sediments associated with the upper thrust plate (Ordovician to Pennsylvania Age strata).

Regional volcanism deposited a thick layer of calc-alkaline volcanic rocks across the area presently occupied by the southern Wah Wah Mountains. The Basin and Range extensional event created much of the current topography of the area by stretching the region about 40 miles westward, creating mountains with intervening valleys separated by range-bounding, normal faults that rotate at depth into a regional decollement. Local bimodal (calc-alkaline and basaltic) volcanism also occurred in the southern Wah Wah Mountains, associated with Basin and Range extension which began about 26 million years ago. The sedimentary and volcanic stratigraphy of Blawn Mountain is summarized in Table 7.2 below.

Table 7.2 Stratigraphy of the Blawn Mountain Area from Krahulec (2007) as Modified from Hofstra (1984) and Abbott and Others (1983)

Eras		Periods	Epochs	Groups	Formations	Members
Cenozoic		Quaternary				Alluvium And Colluvium
		Tertiary	Pliocene		Steamboat Mountain	Basalt
			Miocene	Quichapa	Blawn	Rhyolite Tuff Bauers Tuff Mafic Flow Garnet Tuff
			Oligocene		Isom Bullion Canyon Volcanics	Bald Hills Tuff Three Creeks Tuff
				Needles Range	Lund Wah Wah Springs Cottonwood Wash Tuff Escalante Desert	
						Conglomerate
Paleozoic	Upper Plate of Wah Wah Thrust	Cambrian			Orr Wah Wah Summit Trippe Limestone Pierson Cove Eye of Needle Limestone Swasey Limestone Whirlwind Dome Limestone Peasley Limestone Chisholm Shale Howell Limestone Pioche	
					Prospect Mountain Quartzite	
Proterozoic					Mutual	
Paleozoic	Lower Plate of Wah Wah Thrust	Pennsylvanian			Callville Limestone	
		Mississippian			Woodman Gardison Limestone	
		Devonian			Fitchville Pinyon Peak Limestone Simonson Dolomite Sevy Dolomite	
		Silurian			Laketown Dolomite	
		Ordovician			Ely Springs Dolomite Eureka Quartzite Kanosh Shale Juab Limestone	

7.5 MINERALIZATION

Alunite mineralization is found on four ridges that occur within the Blawn Mountain Project. Acid sulfate alteration associated with a shallow, possibly laccolithic intrusion altered the silicic-alkalic rhyolite porphyries, flows and tuffs belonging to the Miocene Blawn Formation and the Oligocene Needles Range Group. Alteration tends to be in linear bodies reflecting the role of normal faults in controlling the mineralization. Alteration is zoned away from the point of hydrothermal fluid upwelling. The mineralized ridges are erosional remnants of a once larger altered area. The alteration zoning types as described by Hofstra (1984) are summarized in Table 7.3.

Table 7.3 Mineral Alteration Zones of Acid Sulfate Alteration at Blawn Mountain Alteration Intensity Increases from Top to Bottom in the List

Zone Name	Mineral Assemblage	Rock Texture Destroyed?
Low Propylitic	chlorite-calcite ± quartz	No
High Propylitic	quartz-epidote-montmorillonite-sericite ± pyrite ± kaolinite± quartz ± calcite ± illite	No
Hematite-Clay	hematite-kaolinite-chlorite-montmorillonite ± alunite ± sericite	No
Quartz-Alunite	quartz- alunite ± kaolinite ± pyrophylite ± cristoballite ± hematite	Mostly
Silica Cap	quartz ± opal ± cristoballite ± tridymite	Yes
Quartz-Sericite-Alunite	quartz-sericite-pyrite ± alunite	Yes

(Modified from Hofstra, 1984)

Krahulec (2007) described the appearance of rocks from the silica cap and quartz-alunite zone as follows, “The Silica Cap is a zone of intense silicification believed to be the near-surface manifestation of the hydrothermal channel-ways. The silica is typically buff, dense, and massive but may be quite porous and vuggy locally and resemble a siliceous sinter… On the surface the Quartz-Alunite alteration zones are composed of white to cream to buff to gray to pink, generally fine grained, punky to dense, intermixed alunite and silica with only minor amounts of other impurities, mainly iron… Alunite also occurs locally as coarse (>0.5in.), lathy, typically pink crystals in veins. Kaolinite becomes increasingly important, at the expense of alunite, in the Quartz-Alunite zone near the boundary with the Hematite-Clay zones and also where the Quartz-Alunite zones are cut by faults (Walker, 1972). Dickite (a high-temperature member of the kaolinite group) is reported by Whelan (1965) and Thompson (1991) in the Quartz-Alunite zone”.

Figure 7.4 depicts mapping by Hofstra of the alteration facies in the Blawn Mountain area and its effect on topography. The extremely erosion-resistant Silica Cap forms the tops of peaks and the underlying highly erosion-resistant Quartz-Alunite facies forms the steepest parts of the ridges. In cross section, the alteration zones have two basic forms: a nested-cone geometry and a relatively flat-lying form, as shown in Figure 7.5.

Krahulec gives the following description of the two geometries, “The cone-shaped (narrow end at the base) zones are interpreted as the primary area of strong hydrothermal upwelling . . . and the adjoining flat-bottomed zones are recognized as permeability-controlled areas above the paleo-ground-water table where steam-heated H_2S is oxidized to H_2SO_4. Only the central portion of Area C (Area 1) at Blawn Mountain is clearly a funnel-shaped zone. The other flat bottomed alunite zones are strongly controlled by the higher porosity and permeability of the host volcanic rocks, while the hydrothermal cones are largely independent of these factors (Hofstra, 1984)”. Krahulec continues this discussion by quoting Hofstra, “…The control of permeability on the degree of alteration intensity is most important near the margins of Quartz-Alunite altered zones. Alteration is pervasive and unaffected by variations in the permeability of the host rocks”. The alteration zones tend to be thicker in cone-shaped areas than in flat-lying areas. It is possible that there were more cone-shaped feeder zones but they were eroded or are buried under valley fill.

Figure 7.6 shows the geometry of the Area 1 alunite alteration zone and Figure 7.7 shows the geometry of the Area 2 alunite alteration zone. Both figures are derived from the block model used in the resource calculations presented in this report.

8 DEPOSIT TYPES

There is no known formal industrial mineral ore deposit model for alunite. The characteristics for a model and some exploration criteria are derived from three publications: Hall (1978), Hall and Bauer (1983), and Hofstra (1984).

The local alunite deposit has been described, in the above-mentioned publications, as hydrothermal alteration of calc-alkaline volcanic rocks.

9 EXPLORATION

The Blawn Mountain area was first evaluated by ESI as part of a nationwide alunite exploration program in 1969 which included literature searches, aerial reconnaissance for the bleached alunite zones and field studies. In 1970 ESI started the first systematic exploration of the Blawn Mountain Project which they referred to as the NG Alunite property. Initial exploration focused on four separate mineralized zones located on and along three ridges. All four of these mineralized zones are completely within the current PRC lease holding. ESI conducted mapping, surface sampling and drilling before focusing its attention on the northwest trending ridge now referred to as Area 1. Figure 9.1 shows the rotary drill locations completed by ESI at Blawn Mountain.

After acquiring the mineral leases in 2011, PRC initiated a validation drilling program on Area 1 primarily to validate the previous exploration efforts. Under the guidance of NAE, a combination of 19 core holes and 16 reverse circulation holes were completed on Area 1 between October 2011 and February 2012. During the Geology QP's first site visit in February, additional recommendations were made to the validation drilling program that included the two final reverse circulation holes and some adjustments to the sample preparation procedures. All 35 drill holes were twinned to locations of previous drill holes completed by ESI.

A second drilling program was initiated by PRC in July of 2012. The drilling program included 17 additional holes on Area 1, 50 holes on Area 2, two holes on Area 4, and 21 holes on the ridgeline extending southwest of Area 1, now referred to as the Southwest Extension. A total of 90 drill holes were completed including 74 reverse circulation holes, eight HQ core holes, and eight PQ core holes. PQ core holes were completed to collect material for metallurgical testing. A total of 32,392ft were completed in the reverse circulation and core drilling program. In addition to the exploration drilling, PRC completed 10 groundwater monitor wells in valley fill material to begin baseline characterization of near-surface hydrology. The 10 monitor wells represent a total drilling footage of 2,400ft.

A third reverse circulation drilling program was conducted in January and February of 2013. The program included two holes on Area 1 and 17 holes on Area 2 for a combined total of 8,310ft. The primary purpose of the drilling was to further increase geologic assurance for resource assessment.

In addition to drilling, additional field investigations were conducted to:

- Further assess faults and fractures on Area 2 and the role they play in controlling mineralization and silicification;
- Investigate a prominent fault trending northwest from the north side of Area 1 as a possible exploration target for additional mineralization;
- Reconnaissance of Area 3 for future exploration.

10 DRILLING

10.1 HISTORICAL DRILLING

ESI company records indicate that a total of 320 drill holes were completed on their NG Alunite deposit. Within Area 1, 287 holes were completed, 18 holes at Area 2, 12 holes at Area 3, and three holes on Area 4. Six of the drill holes located in Area 2 are located within a 155ac tract where surface rights are jointly shared between SITLA and a third-party. Additionally, the rights to metallic resources for this tract are owned by a third-party, with PRC controlling the rights to potash and other minerals.

ESI used air track percussion drilling and conventional rotary drilling in its exploration efforts. Air track drilling was primarily used as a prospecting tool to test the ground where there were poor bedrock exposures. Rotary drilling was used to define subsurface geology and collect samples for analysis.

There are numerous drill site locations where multiple holes have been drilled. This was due to:

- Air track drilling being first used at several sites where there were poor surface exposures to identify sites to be followed with rotary drilling;
- Adverse drilling conditions were encountered at several sites that required abandoning a drill hole, moving over a few feet on the drill pad and making another attempt;
- Several locations where holes were re-entered or drilled a second time to collect additional information.

ESI completed its drilling in three stages:

- Reconnaissance drilling in 1971, completing 10 holes for a total of 2,650ft. Three holes were completed on Area 1, four holes at Area 2, and three holes at Area 3;
- Exploration drilling in 1972, completing an additional 42 drill holes. 16 holes were completed at Area 1 for a total of 4,438ft, 14 holes were completed at Area 2 for a total of 2,865ft, nine holes were completed at Area 3 for a total of 2,590ft and three holes were completed on Area 4 for a total of 740ft
- Development drilling in 1973 and 1974 on Area 1. Drilling was roughly aligned to a 300 (NW-SE) by 500 (NE-SW) grid pattern oriented to the ridgeline. A total of 268 air track and rotary holes were completed for a total footage of 46,267ft

ESI did not maintain complete records for most of the air track drill holes and some of the abandoned holes. Complete records were only maintained for holes with assays. PRC has geologic logs for all holes, but is missing coordinates for the air track holes and some abandoned holes.

10.2 PRC DRILLING

PRC completed a validation drilling program on Area 1 between October 2011 and February 2012. All drill sites were twinned to locations of previous drill holes completed by ESI and were oriented to provide adequate spatial representation of the deposit. Nineteen of the PRC holes were drilled using wire-line slim coring methods, continuously collecting HQ core. A total of 6,764ft of drilling was accomplished through core drilling with an average recovery of 91%. The remaining 15 drill holes were completed using RC drilling equipped with either a down-hole hammer or deep-hole bit. A total of 8,050ft were completed with RC drilling.

NAE managed logistics, logging, and sampling for the PRC program. Two different drilling contractors were used in the RC drilling. It was quickly recognized by NAE that the first drilling contractor was having difficulty recovering sufficient sample volumes in the RC drilling. After several measures were employed to improve sample returns, NAE brought in a second drilling contractor to complete the RC drilling. The second contractor did not experience the same problems and was able to deliver adequate sample volumes and complete the drilling program. The first drilling contractor completed seven RC holes for a total of 4,210ft. None of the samples from these seven holes have been used or incorporated by PRC in their evaluation of the Blawn Mountain Project. The second drilling contractor completed eight holes for a total of 3,840ft. Samples and data from these holes are being used by PRC in their evaluation of the deposit.

PRC completed a second substantial drilling program at Blawn Mountain in the summer of 2012. Drilling was accomplished using RC, wire-line coring for HQ and PQ core and conventional rotary methods. The program included a total of 90 holes in Areas 1, 2 and 4 plus 10 groundwater monitor wells.

The second drilling program accomplished several goals:

- Further delineated resources on Area 1 with nine RC holes and five HQ core holes;
- Explored and defined potential alunite resources on Area 2 with 44 RC holes and three HQ core holes;
- Explored and defined potential alunite resources on the ridge extending southwest from Area 1 with 19 RC holes and two HQ core holes. This area is referred to as the Southwest Extension;

- Completed five PQ core holes on Area 1 and three PQ core holes on Area 2. Core from the PQ holes was used to develop bulk metallurgical samples;
- Performed resource reconnaissance in Area 4;
- Completed 10 widely-spaced rotary holes in the alluvial/colluvial areas surrounding the alunite deposits to collect samples for overburden testing and observe groundwater conditions. All 10 rotary holes were subsequently converted to monitor wells to observe and sample groundwater conditions.

All core drilling was completed by a contractor using a track-mounted LF-70 core drill. A total of 2,804ft of HQ core drilling was accomplished with an average core recovery of 91% and a total of 4,054ft of PQ size core drilling with a recovery rate of 95%. All RC drilling was completed by a separate contractor utilizing up to three RC rigs simultaneously on the project area. All three RC rigs used on the program were track-mounted Schramm DLD 1000 rigs, equipped with 4in pipe and wet rotary splitters for sample collection. All RC drilling was done with foam injection to minimize dust and water consumption.

A third drilling program totaling 21 holes at 18 locations was completed by PRC in January and February of 2013. Two RC holes were completed on Area 1 and 16 RC holes were completed on Area 2. All drilling was intended to improve resource delineation and geologic assurance. Six of the drill holes on Area 2 were drilled as angle holes, specifically targeting potential resources that could not be accessed with roads and vertical holes. Difficult drilling conditions required abandoning and restarting drill holes at three of the angle hole sites. 480ft of drilling was completed on Area 1 and 7,830ft on Area 2.

A fourth drilling program was conducted in the spring/summer of 2013 in the southern end of the Wah Wah valley. Three test bores were drilled for a total of 3,340ft. This evaluation was to assess depth to bedrock and to assist in determining the presence of groundwater. In the fall of 2013, two of the bore holes were completed as observation wells to assist in assessing the potential to develop a wellfield in the area.

Throughout the second and third drilling programs, field operations and geologic logging was completed by NAE. Millcreek provided an onsite QA/QC manager who oversaw all data collection and sampling for the programs for the two drilling programs.

Table 10.1 summarizes the drilling completed by PRC from October 2011 through July 2013. Figure 10.1 shows the locations of all drilling completed by PRC on the main tract of land (excluding ML 52364).

Table 10.1 PRC Drilling Summary

Phase 1 - Area 1						
Drill Hole ID	UTM83-12			Depth (Ft)	Comp. Date	Type
	Easting	Northing	Surface			
BM1	279,764	4,240,018	2,235	230	10/25/2011	HQ
BM10	279,089	4,239,436	2,273	536	11/20/2011	HQ
BM11	279,309	4,239,629	2,239	610	11/26/2011	RC
BM11A	279,309	4,239,629	2,239	457	11/30/2011	HQ
BM12	280,161	4,239,820	2,264	300	12/4/2011	HQ
BM13	279,836	4,239,759	2,230	200	11/30/2011	HQ
BM14	279,533	4,239,655	2,237	370	11/15/2011	HQ
BM14A	279,533	4,239,655	2,237	650	11/30/2011	HQ
BM14B	279,533	4,239,655	2,237	651	12/06/2011	HQ
BM15	279,505	4,239,575	2,196	380	12/05/2011	RC
BM16	280,031	4,239,773	2,249	228	12/05/2011	HQ
BM16A	280,031	4,239,773	2,249	200	12/07/2011	RC
BM17	279,433	4,239,573	2,201	470	12/04/2011	RC
BM17A	279,433	4,239,573	2,201	820	12/07/2011	RC
BM18	279,937	4,239,817	2,259	198	12/10/2011	HQ
BM19	279,443	4,239,658	2,241	650	12/14/2011	HQ
BM2	279,270	4,239,337	2,211	146	10/26/2011	HQ
BM20	280,026	4,239,957	2,315	400	01/03/2012	RC
BM21	279,356	4,239,488	2,205	810	12/16/2011	RC
BM22	279,804	4,239,834	2,264	338	12/12/2011	HQ
BM23	279,490	4,239,746	2,272	198	12/14/2012	HQ
BM24	279,644	4,239,838	2,266	420	12/19/2011	RC
BM25	279,357	4,239,676	2,242	790	12/19/2011	RC
BM26	279,913	4,239,978	2,289	410	01/05/2012	RC
BM27	280,032	4,239,872	2,293	300	01/07/2012	RC
BM28	279,535	4,239,791	2,270	700	01/11/2012	RC
BM3	279,779	4,239,928	2,277	280	10/16/2011	HQ
BM3A	279,779	4,239,928	2,277	200	10/30/2011	RC
BM4	279,473	4,239,807	2,258	368	11/01/2011	HQ
BM5	279,498	4,239,891	2,250	440	11/03/2011	RC
BM6	279,144	4,239,376	2,258	301	11/14/2011	HQ
BM7	280,000	4,240,069	2,262	133	11/14/2011	HQ
BM8	279,392	4,239,776	2,247	800	11/10/2011	RC
BM8A	279,392	4,239,776	2,247	800	11/20/2011	RC
BM9	279,537	4,239,711	2,270	477	11/21/2011	HQ

Table 10.1 PRC Drilling Summary, Continued

Phase 2 - Area 1						
Drill Hole ID	UTM83-12			Depth (Ft)	Completion Date	Type
	Easting	Northing	Surface			
PDH 1-03	280,500	4,239,928	2,186	124	08/27/12	HQ
PDH 1-04	280,222	4,240,002	2,280	240	10/11/12	RC
PDH 1-05	280,152	4,239,948	2,330	404	08/30/12	HQ
PDH 1-06	280,089	4,239,680	2,192	520	10/07/12	RC
PDH 1-07	279,655	4,239,963	2,241	100	09/02/12	RC
PDH 1-08	279,655	4,240,028	2,220	220	09/22/12	RC
PDH 1-09	279,576	4,239,959	2,245	480	09/23/12	RC
PDH 1-10	279,086	4,239,564	2,232	600	09/25/12	RC
PDH 1-11	278,953	4,239,338	2,236	394	08/27/12	HQ
PDH 1-12	279,576	4,239,959	2,245	270	09/26/12	RC
PDH 1-13	279,086	4,239,564	2,232	200	09/19/12	RC
PDH 1-14	279,174	4,239,507	2,272	560	09/21/12	RC
PDH 1-15	278,953	4,239,338	2,236	300	10/05/12	RC
PDH 1-16	278,827	4,239,266	2,244	450	08/24/12	RC
PDH 1-17	278,773	4,239,183	2,259	324	08/24/12	RC
PDH 1-18	278,474	4,239,177	2,288	200	09/22/12	RC
PDH 1-19	278,469	4,239,037	2,298	400	09/23/12	RC
PDH 1-20	278,709	4,238,893	2,259	400	10/06/12	RC
PDH 1-22	279,040	4,238,878	2,184	140	10/08/12	RC
PDH 1-23	278,375	4,239,104	2,293	370	09/15/12	RC
PDH 1-24	278,317	4,238,842	2,281	299	08/11/12	HQ
PDH 1-25	278,592	4,238,879	2,245	350	10/04/12	RC
PDH 1-26	278,427	4,238,797	2,243	300	08/11/12	RC
PDH 1-27	278,265	4,238,676	2,293	600	09/02/12	RC
PDH 1-30	278,171	4,238,591	2,298	400	10/11/12	RC
PDH 1-31	278,050	4,238,557	2,276	400	09/13/12	RC
PDH 1-32	277,991	4,238,463	2,289	499	09/22/12	HG
PDH 1-33	278,214	4,238,548	2,288	570	08/21/12	RC
PDH 1-35	278,768	4,239,010	2,215	100	10/08/12	RC
PDH 1-36	278,584	4,238,980	2,281	340	10/06/12	RC
PDH 1-37	278,455	4,238,938	2,280	240	10/04/12	RC
PDH 1-38	277,884	4,238,365	2,280	355	08/30/12	RC
PDH 1-39	279,425	4,239,700	2,252	1,066	09/14/12	PQ
PDH 1-40	279,602	4,239,825	2,269	669	09/19/12	PQ
PDH 1-41	279,726	4,239,883	2,275	454	09/22/12	PQ
PDH 1-42	279,853	4,239,960	2,282	404	09/25/12	PQ
PDH 1-43	280,114	4,239,945	2,332	559	10/04/12	PQ
PDH 1-44	277,761	4,238,284	2,247	340	08/27/12	RC

Table 10.1 PRC Drilling Summary, Continued

Phase 2 - Area 2						
Drill Hole ID	UTM83-12			Depth (Ft)	Completion Date	Type
	Easting	Northing	Surface			
PDH-2-01	282,709	4,239,824	2,096	200	07/15/12	RC
PDH-2-03	282,578	4,239,596	2,137	220	07/14/12	RC
PDH-2-04	282,670	4,239,709	2,101	160	07/20/12	HQ
PDH-2-05	282,628	4,239,356	2,106	280	08/24/12	RC
PDH-2-06	282,680	4,239,279	2,091	620	07/19/12	RC
PDH-2-07	282,572	4,239,271	2,114	360	08/26/12	RC
PDH-2-08	282,579	4,239,134	2,097	320	07/19/12	RC
PDH-2-09	282,350	4,238,998	2,136	400	08/03/12	RC
PDH-2-10	282,485	4,238,937	2,110	400	08/01/12	RC
PDH-2-11	282,209	4,238,783	2,129	330	08/07/12	RC
PDH-2-12	282,418	4,238,809	2,106	350	08/24/12	RC
PDH-2-13	282,843	4,239,378	2,106	220	07/15/12	RC
PDH-2-14	281,725	4,239,078	2,116	220	09/03/12	RC
PDH-2-15	282,102	4,238,813	2,159	320	08/03/12	RC
PDH-2-16	282,212	4,238,893	2,133	420	07/18/12	HQ
PDH-2-17	282,318	4,238,716	2,126	300	08/21/12	RC
PDH-2-18	281,874	4,238,954	2,110	100	02/02/13	RC
PDH-2-19	282,055	4,238,578	2,150	500	08/06/12	RC
PDH-2-20	282,207	4,238,571	2,111	360	08/12/12	RC
PDH-2-22	281,729	4,238,725	2,233	220	08/08/12	RC
PDH-2-23	282,080	4,238,419	2,113	500	07/31/12	RC
PDH-2-24	281,484	4,239,032	2,124	270	09/04/12	RC
PDH-2-26	281,597	4,238,737	2,241	370	08/26/12	RC
PDH-2-27	281,791	4,238,615	2,215	400	07/20/12	RC
PDH-2-28	281,876	4,238,532	2,198	590	08/22/12	RC
PDH-2-29	281,354	4,238,905	2,130	200	09/13/12	RC
PDH-2-30	281,442	4,238,691	2,240	420	09/02/12	RC
PDH-2-31	281,595	4,238,550	2,189	160	09/02/12	RC
PDH-2-32	281,741	4,238,473	2,168	570	08/09/12	RC
PDH-2-33	282,001	4,238,334	2,111	680	07/20/12	RC
PDH-2-34	281,130	4,238,827	2,135	140	09/14/12	RC
PDH-2-36	281,429	4,238,504	2,215	504	08/01/12	HQ
PDH-2-37	281,497	4,238,341	2,163	360	08/01/12	RC
PDH-2-38	281,769	4,238,257	2,120	440	08/03/12	RC
PDH-2-39	281,023	4,238,619	2,121	200	09/15/12	RC
PDH-2-40	281,205	4,238,475	2,178	220	09/12/12	RC
PDH-2-42	281,580	4,238,162	2,157	360	07/24/12	RC
PDH-2-43	281,245	4,238,051	2,121	300	09/16/12	RC
PDH-2-44	281,428	4,237,902	2,101	520	09/16/12	RC

Table 10.1 PRC Drilling Summary, Continued

PDH-2-45	281,068	4,238,191	2,167	300	09/12/12	RC
PDH-2-46	282,288	4,239,662	2,154	355	07/12/12	RC
PDH-2-47	282,482	4,239,647	2,124	370	07/13/12	RC
PDH-2-48	281,149	4,238,596	2,182	120	09/03/12	RC
PDH-2-49	281,343	4,238,456	2,219	300	09/15/12	RC
PDH-2-50	281,912	4,238,248	2,109	620	07/17/12	RC
PDH-2-51	281,811	4,238,096	2,144	380	07/21/12	RC
PDH-2-52	280,965	4,238,058	2,149	200	09/17/12	RC
PDH-2-53	281,325	4,238,347	2,205	329	08/04/12	PQ
PDH-2-55	281,884	4,238,747	2,195	245	08/06/12	PQ
PDH-2-56	282,488	4,239,033	2,117	329	08/08/12	PQ

Phase 2 - Area 4						
Drill Hole ID	UTM83-12			Depth (Ft)	Completion Date	Type
	Easting	Northing	Surface			
PDH 4-01	281,077	4,234,846	1,977	200	09/20/12	RC
PDH 4-02	281,480	4,234,931	2,005	500	09/20/12	RC
Phase 2 - Monitor Wells and Groundwater Test Bores						
Drill Hole ID	**UTM83-12**			**Depth (Ft)**	**Completion Date**	**Type**
	Easting	**Northing**	**Surface**			
MW-1	277,499	4,238,236	2,122	240	10/06/12	ROT
MW-2	281,004	4,241,119	1,996	200	10/04/12	ROT
MW-3	282,521	4,240,855	1,937	240	10/08/12	ROT
MW-4	279,286	4,237,129	2,064	260	10/05/12	ROT
MW-5	282,223	4,237,288	2,028	260	10/07/12	ROT
MW-6	279406	4238275	2109	320	06/19/13	ROT
MW-9	280385	4239246	2051	60	06/21/13	ROT
MW-10	279859	4240683	2078	260	07/18/13	ROT
MW-11	284897	4242302	1883	280	07/28/13	ROT
MW-13	282770	4243461	1852	280	07/20/13	ROT
TB-03	290260	42491160	1632	980	03/28/13	ROT
TB-03	290184	4247778	1648	1205	04/09/13	ROT
TB-03	291235	4247638	1649	1155	04/16/13	ROT

Phase 3 - Area 1						
Drill Hole ID	UTM83-12			Depth (Ft)	Completion Date	Type
	Easting	Northing	Surface			
A1-03	280,250	4,239,817	2,254	300	02/14/13	RC
A1-04	280,356	4,240,057	2,212	180	02/13/13	RC

Table 10.1 PRC Drilling Summary, Continued

Phase 3 - Area 2						
Drill Hole ID	UTM83-12			Depth (Ft)	Completion Date	Type
	Easting	Northing	Surface			
A2-03	282,020	4,238,906	2,106	300	02/04/13	RC
A2-04	282,038	4,238,748	2,173	440	01/16/13	RC
A2-05	281,845	4,238,454	2,184	850	02/04/13	RC
A2-06	281,640	4,238,369	2,157	800	01/23/13	RC
A2-07	281,700	4,238,134	2,151	560	01/18/13	RC
A2-08	282,701	4,239,495	2,130	590	01/12/13	RC
A2-09	281,377	4,237,969	2,111	180	02/08/13	RC
A2-10	281,242	4,238,225	2,190	300	02/07/13	RC
A2-11	281,134	4,238,332	2,164	200	02/06/13	RC
A2-12	281,378	4,238,601	2,245	250	02/07/13	RC
A2-12	281,378	4,238,601	2,245	600	02/11/13	RC
A2-13	281,064	4,238,730	2,122	260	02/11/13	RC
A2-14	281,266	4,238,848	2,136	300	02/10/13	RC
A2-15	281,603	4,239,062	2,106	300	02/14/13	RC
A2-16A	281,520	4,238,719	2,243	80	01/28/13	RC
A2-16B	281,520	4,238,719	2,243	800	02/01/13	RC
A2-17A	281,673	4,238,724	2,243	680	01/21/13	RC
A2-17B	281,673	4,238,724	2,243	720	01/24/13	RC
A2-18	281,440	4,238,140	2,137	380	01/30/13	RC

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 HISTORICAL WORK

From 1969 through 1974, ESI collected samples from rotary drilling on 10ft intervals. ESI also collected extensive outcrop and trench samples. For drilled samples, the material penetrated (alunite, clay, dolomite, non-alunite) was reported in 10ft increments along with analytical results, the data column headings were: % Al_2O_3 by SO_3 determination, % soluble Al_2O_3, % Al_2O_3 by K + Na determination, % K_2O, and % Na_2O). In some drill holes, lab analysis was only performed on samples at every 30ft to 50ft or on composite samples from 4 10ft intervals. For surface samples, the alumina analysis of the sample was typically plotted by location on a resource plate.

ESI determined both the elemental and mineralogical content of a large number of samples. Some of the mineralogy was done by X-ray diffraction (XRD). The most critical analytical number for ESI was the Al_2O_3 content of the alunite and was determined by three methods simultaneously:

- Indirectly by measuring the SO_3 content through a LECO furnace determination of the sulfur content;
- By determining the soluble Al_2O_3 content, presumably by wet chemical methods;
- By indirectly determining the Na and K content.

ESI also measured K_2O and Na_2O by an unspecified method. ESI documentation provides results achieved by different techniques and different analytical laboratories. Laboratories listed were ESI, Alumet, HRI, Skyline Labs and NAS. Though ESI did evaluate their internal analytical testing with outside labs (the results are available in the documents PRC has obtained) there is little information relating to actual sample procedures or quality control methods.

11.2 SAMPLING METHOD AND APPROACH

PRC’s validation drilling program, logistics, logging and initial sample preparation was managed by NAE following recommendations made by the Geology QP. NAE maintained chain of custody for all samples from the time of collection at the drill sites, through initial sample preparation, to delivery of samples at the ALS Minerals facility in Winnemucca, NV where they have undergone further preparation for analysis.

For PRC's validation drilling program, NAE collected samples on 10ft intervals for core holes and on 5ft intervals for RC holes. Geologic logs have been maintained for all drill holes and include descriptions for lithology, alteration and recovery. In addition, core logs provide detail on fractures and orientations. Following logging, core was transported to a preparation facility set up by NAE where the core was cut longitudinally into one-half and one-quarter-core sections. Core samples submitted for analyses are comprised of 10ft quarter-core sections. Each sample weighs approximately 10 to 11 lbs. The remaining half and quarter-core sections are stored in traditional waxed cardboard core boxes in a secure storage facility in Milford. For RC drilling, samples were collected at 5ft intervals. Cuttings coming up through the central return discharge hose passed through a cyclone and then through a Jones splitter. The splitter is set to a 50/50 split with one split being retained. Samples were collected continuously at 5ft intervals. Each 5ft sample weighs approximately 18 to 24 lbs.

For the second and third drilling programs adjustments were made to the RC sampling. Sample intervals were changed from 5ft intervals to 10ft intervals based on a study of analytical variability observed in the validation drilling. Because all of the RC drilling in the second and third drilling programs were completed using foam injection, adjustments were made to collect between 18 to 24 lbs of material directly from the rotary splitter, eliminating the use of the Jones splitter.

11.3 SAMPLE PREPARATION, ANALYSES AND SECURITY

HQ core and RC samples from PRC's validation and infill drilling programs were shipped directly by NAE personnel to the ALS Minerals sample preparation facility in Winnemucca, NV. To date, NAE has delivered 944 slim core samples and 4,541 RC samples from the three drilling programs. This includes 335 blind duplicate and 150 reference samples to evaluate analytical precision.

At the ALS mineral sample preparation facility, samples are prepared using the following steps:

- Samples are initially weighed and entered into the ALS tracking system;
- Samples are completely crushed to 70% < 2mm;
- Samples are then passed through a riffle splitter to create a 1000 gram (g) representative sample;
- The 1000g samples are then pulverized to 85% < 75µm;
    - Prepared samples are then forwarded onto the ALS Minerals Laboratory in Vancouver, B.C. for geochemical analysis.

All reject material following splitting is saved and returned to PRC for potential future testing. For the validation drilling program, PRC selected two analytical packages to use on all samples. The first package is a whole rock analysis for major oxides using Ion Couple Plasma-Atomic Emission Spectroscopy (ICP-AES) following a lithium metaborate fusion. Under this procedure, determinations are made for SiO_2, Al_2O3, Fe_2O_3, CaO, MgO, K_2O, Cr_2O3, TiO_2, MnO, P_2O_5, SrO, BaO and LOI (loss on ignition). Reporting levels are to 0.01%. The second analytical package is an ICP-AES package for major, minor and trace elements using a four acid digestion. Determinations in the second analytical package include Al, Ca, Fe, K, Mg, Na, S and Ti reported to 0.01% levels and Ag, As, Ba, Be, Bi, Cd, Co, Cr, Cu, Ga, La, Mn, Mo, Ni, P, Pb, Sb, Sc, Sr, Th, Tl, U, V, W and Zn all reported in parts per million (ppm) concentrations.

For the second drilling program, completed between July and October of 2012, the ICP-AES whole rock analytical package was used on all samples from the HQ core drilling and the RC drilling. The HQ core samples were also tested for minor and trace elements using the four-acid ICP-AES procedure. Both analytical packages were determined by ALS Minerals. ALS Minerals also prepared duplicate pulps of the HQ core samples that were subsequently sent to DCM Science in Denver, CO for mineral analysis using XRD.

For the third drilling program completed in January and February of 2013, the ICP-AES whole rock analytical package was used on all samples. In addition, sulfate determination by carbonate leach and gravimetric analysis was completed on all of the drilling samples, plus a selected group of pulp samples from the previous two drilling programs.

Geotechnical logging and testing have been completed on two holes from Area 1, PDHC-1-09 and PDHC-1-11. Geotechnical work was completed by Seegmiller International, located in Salt Lake City, Utah. Geotechnical tests completed on samples from the two core holes include: Point Load Testing (axial and diametric), Uniaxial Compression, Elastic Modulus, Direct Shear and Bulk Density.

Five PQ core holes were completed during the second drilling program on Area 1 and three more were completed on Area 2. The PQ holes were drilled to collect representative material for metallurgical testing and explained in further detail in Section 13 of this report.

12 DATA VERIFICATION

Mr. Kerr, the Geology QP, has conducted numerous site visits to the Blawn Mountain property in support of the drilling and alunite resource characterization. Mr. Kerr first performed a site visit on February 9, 2012 and has made several site visits to the property since that time. Mr. Kerr's last visit to the property was on September 9, 2016. The site visits have confirmed the location and access routes of previous and current exploration activities. During the first site visit PRC's validation drilling program was still in progress with both the core and RC rigs operating. Mr. Kerr was able to observe drilling, logging and sampling procedures at the drill sites. Mr. Kerr also visited and observed the core cutting procedures and sample storage facilities being employed by NAE in Milford. At the time of the first site visit none of the drill samples had yet been shipped to ALS Minerals for sample preparation and analysis. At the request of Mr. Kerr, blind duplicate samples of core were added into the sample sequence as one step of quality control.

During subsequent site visits, the Geology QP has been able to observe and confirm both alunite and non-alunite lithologies, alterations, geologic contacts as well as observing several of the major structures that bound the alunite deposits. Throughout the second and third drilling programs an on-site QA/QC manager was provided to ensure that logging, data collection and sampling procedures were being followed in a consistent manner and that a chain of custody was maintained.

A search of the SITLA online database confirms the mining leases PRC has with the State of Utah for the Blawn Mountain Project. PRC has valid mineral control through the Exploration/Option Agreement and the two mineral leases for a combined total of 15,403.72ac.

The drill program carried out by PRC in 2011 and 2012 for Area 1 was designed to validate the previous drilling data collected by ESI between 1969 and 1974. All of PRC drill hole locations were twinned to ESI drill holes. Table 12.1 identifies the ESI holes that are twinned by the PRC holes.

Table 12.1 PRC Validation Drilling

PRC Validation Holes	Drill Type - Driller	Twin ESI Drill Hole
BM1	Core - Layne	C159
BM2	Core - Layne	C103 (C207)
BM3	Core - Layne	C12A
BM3A	RC - Gardner	C12A
BM4	Core - Layne	C178
BM5	RC - Layne*	C145
BM6	Core - Layne	C196
BM7	Core - Layne	C11
BM8	RC - Layne*	C197
BM8A	RC - Gardner	C197
BM9	Core - Layne	C125
BM10	Core - Layne	C194
BM11	RC - Layne*	C180
BM11A	Core - Layne	C180
BM12	Core - Layne	C162 (C120)
BM13	Core - Layne	C79
BM14	Core - Layne	C9
BM14B	Core - Layne	C9
BM15	RC - Layne*	C168 (C208)
BM16	Core - Layne	C13
BM16A	RC - Gardner	C13
BM17	RC - Layne*	C170A
BM17A	RC - Gardner	C170A
BM18	Core - Layne	C163
BM19	Core - Layne	C172
BM20	RC - Gardner	C7
BM21	RC - Layne*	C175
BM22	Core - Layne	C164
BM23	Core - Layne	C88
BM24	Core - Layne	C157
BM25	RC - Layne*	C171
BM26	RC - Gardner	C156
BM27	RC - Gardner	C182
BM28	RC - Gardner	C130

* Samples not used due to poor recovery

The Geology QP has examined and compared the K_2O and Al_2O_3 values from 27 of the Area 1 PRC holes with their respective twin ESI holes. The comparison covers 639 assay intervals or 6,390ft of drilling. On an interval per interval basis there is poor correlation for K_2O and Al_2O_3 concentrations between the two sets of data. However, composite intervals for each hole show that the PRC holes have concentrations that range from 9% to 19.2% higher than the ESI data.

Table 12.2 summarizes the composite values for the twinned intervals. Poor correlation between the two sets of data can be attributed to different drilling methods and most likely different analytical techniques. ESI used conventional rotary drilling methods. Rotary samples tend to be prone to dilution and wall-rock contamination compared to core and RC drilling. Though it is not specified in the ESI documents, K_2O was most likely determined by traditional spectrometry such as atomic absorption or flame photometry versus the ICP-AES analyses completed by ALS Minerals.

Table 12.2 Composite Values for Twinned Validation Drilling

PRC Drill ID	ESI Twin ID	Composite Interval		PRC		ESI	
		From	To	K_2O (%)	Al_2O_3 (%)	K_2O (%)	Al_2O_3 (%)
BM1	C159	10	230	4.84	18.39	4.11	15.62
BM2	C103 (207)	Various		1.91	15.19	2.25	12.60
BM3 (3A)	C12A	30	170	3.65	17.69	3.32	14.29
BM4	C178	90	360	3.30	14.81	2.93	12.18
BM6	C196	Various		2.58	14.28	2.53	12.41
BM7	C11	0	130	3.83	16.38	2.76	11.36
BM8 (8A)	C197	10	780	2.43	16.78	2.70	15.19
BM10	C194	Various		1.75	18.25	0.90	14.90
BM11 (11A)	C180	0	450	3.97	15.32	3.33	12.83
BM12	C162	Various		2.85	13.89	2.74	11.95
BM14 (14B)	C9	0	640	3.87	16.31	3.84	15.53
BM16 (16A)	C13	10	220	1.78	15.02	2.24	14.01
BM17 (17A)	C170A	0	810	3.12	15.04	2.86	12.25
BM18	C163	40	170	2.72	14.51	2.65	12.63
BM19	C172	0	620	5.18	19.79	5.02	18.43
BM20	C7	10	400	3.14	16.05	2.96	14.53
BM22	C164	0	330	3.72	15.81	3.57	14.70
BM24	C157	10	400	1.93	13.23	3.69	17.10
BM26	C156	0	410	3.25	15.32	3.51	14.82
BM27	C182	0	280	3.36	14.81	3.01	13.06
BM28	C130	0	650	5.01	18.89	4.18	15.45

Although there was no targeted twin-hole drilling for Area 2, Table 12.3 compares average grade data for K_2O and Al_2O_3 values from historic versus current drill holes collared less than 100ft apart. These comparisons suggest that the current versus historic K_2O and Al_2O_3 grade data is similar for Area 2 despite different drilling methods (rotary versus RC) discussed earlier and most likely different analytical techniques.

Table 12.3 Composite Values for Adjacent Area 2 Drilling

PRC Drill ID	ESI Drill ID	Distance Apart (ft)	Composite Interval		PRC		ESI	
			From	To	K_2O (%)	Al_2O_3 (%)	K_2O (%)	Al_2O_3 (%)
PDH-2-42	B14	30	10	360	2.67	10.63	2.73	12.66
PDH-2-03	B9	100	Various		2.76	12.49	3.96	14.64

A set of 12 sample pulps was forwarded to ACT Labs for comparative analysis (Table 12.4) for the Area 1 twin hole program. For this set of 12 samples there are two sets of analyses from ALS Minerals, original and duplicates, plus the one set of analyses from ACT Labs. ACT Lab analyses compare very closely to ALS Minerals for the 12 samples. Correlation between the two sets of analyses for K_2O and Al_2O_3 exceed 98%.

Table 12.4 Analytical Comparison by Laboratory for Area 1

Sample ID	Al_2O_3 (%)			K_2O (%)		
	ACT Labs	ALS Original	ALS Duplicate	ACT Labs	ALS Original	ALS Duplicate
949922	19.09	19.10	18.50	4.52	4.37	4.51
949937	23.61	23.90	24.00	6.76	6.74	7.04
949947	10.78	10.95	10.85	3.08	2.96	3.15
949957	22.87	23.20	22.90	6.38	5.84	6.49
949967	22.39	22.30	22.60	6.22	5.55	6.30
949977	19.84	19.90	20.10	5.06	4.61	5.28
949987	14.36	14.50	14.70	2.99	2.87	3.08
949992	14.10	14.15	14.35	3.21	3.10	3.34
949997	16.65	16.60	16.95	4.00	3.88	4.12
978252	16.71	16.55	17.05	4.50	4.36	4.65
978257	14.80	14.95	15.20	2.99	2.96	3.10
978262	15.41	15.30	15.00	3.22	3.11	3.28

A comparison made during the PRC validation drilling program in Area 1 was to evaluate analytical results between core and RC drilling. Two RC holes, BM3A and BM16A, are twinned to two of the core holes, BM3 and BM16. Between the two twinned locations there are 340ft of analyses to compare between the two types of drilling. There is a 75% correlation for K_2O between matched sets data between the core and RC data. Al_2O_3 has a lower correlation of 50%. Core generally returns slightly higher grades for K_2O and Al_2O_3 then drill cuttings for the respective intervals.

PRC incorporates duplicate samples in its analytical sampling program as a check to track analytical precision. To date, 274 duplicate samples have been included in the samples submitted to ALS Minerals. Figure 12.1 compares the K_2O values of duplicates to original samples. There is a strong correlation of values between the duplicate and original samples.

Beginning with the second phase of drilling in 2012, PRC began submitting two reference samples into the stream of samples being submitted to ALS Minerals. Both reference samples were prepared from bulk sample material previously collected for metallurgical testing. Original testing of the two reference samples and preparation of pulps for submittal in the drilling program was completed by ACT Labs in Ontario, Canada. Table 12.5 compares the results from the initial testing of the reference samples to results from the past two phases of drilling.

Table 12.5 Reference Sample Comparison

Sample ID	**SiO_2%**	**Al_2O_3%**	**K_2O%**			
ACT Labs Reference Sample A	25.58	25.57	7.52			
ACT Labs Reference Sample B	51.01	17.52	4.52			
Sample ID	**Assay Item**	**Number of Samples**	**Minimum**	**Maximum**	**Average**	**Standard Deviation**
ALS Labs Reference Sample A	SiO2%	64	21.30	26.90	23.93	1.35
	Al2O3 %	64	24.70	29.80	26.60	0.94
	K2O%	64	7.06	8.45	7.54	0.28
ALS Labs Reference Sample B	SiO2%	49	50.00	54.70	52.88	1.02
	Al2O3 %	49	16.75	18.75	17.67	0.44
	K2O%	49	4.22	4.79	4.52	0.12

Reference and duplicate samples show a strong continuity in the dataset without any significant anomalies. The Geology QP believes sufficient steps have been taken to validate the analytical data. The authors are of the opinion that the data used in this report adequately depicts the geology and mineral content. The data is sufficient for resource estimation.

The Geology QP is satisfied with the procedures established by NAE in data collection and sampling. The duplicate samples and comparative analyses returned favorable results that would indicate reliable analyses from ALS Minerals for the validation drilling program. While the ALS Minerals results show higher concentrations than previously indicated in the ESI drilling data, the ALS Minerals analyses confirm the presence of mineralization and indicate grades determined from the ESI drilling data will be conservative estimations.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

PRC contracted HRI from 2011 to 2013 to perform a series of bench-scale comminution, beneficiation, flotation, calcination, leaching, crystallization and solid/liquid separation tests on composites of alunite-bearing drill core and rotary drill cuttings from the exploration drilling program. A large bulk sample collected from Test Pit No. 5 in Area 1 was processed through a pilot plant at the HRI facilities in Golden, Colorado.

PRC also initiated a test program in December 2016 in which whole ore with a moisture content of 2wt% was crushed to minus 1mm and calcined at approximately 575°C, the residence time being 30 minutes and 180 minutes. Calcination operations were intended to decompose the aluminum sulfate [$Al_2(SO_4)_3$] fraction of the alunite ore into gamma-phase alumina (γ-Al_2O_3) and drive off sulfur dioxide (SO_2) for use as feed to the sulfuric acid (H_2SO_4) plant.

A block flow diagram (BFD) and a description of major unit operations in the proposed methods of processing alunite ore for the production of SOP product are given in Section 17 (Figure 17.1).

13.1.1 Product Specifications

Table 13.1 summarizes the three main parameters: purity, particle size and chloride content that determine SOP product quality.

Table 13.1 Typical Market Product Grades

Standard SOP	Low Chloride SOP	Granular SOP	Soluble SOP
Purity: 50wt% K_2O (92.5wt% K_2SO_4) Particle Size: 70 to 10 Tyler mesh Chloride Content: < 1.0%	Purity: 51wt% K_2O (94wt% K_2SO_4) Particle Size: 70 to 10 Tyler mesh Chloride Content: < 0.5%	Purity: 50wt% K_2O (92.5wt% K_2SO_4) Particle Size: 20 to 6 Tyler mesh Chloride Content: < 1.0%	Purity: 52wt% K_2O (96wt% K_2SO_4) Particle Size: 150 to 48 Tyler mesh Chloride Content: < 0.5%

13.2 RECENT AND HISTORICAL METALLURGICAL TEST RESULTS

On April 27, 2011, PRC acquired from a third-party certain historical information pertaining to the NG alunite property, including data on drilling results, resource estimates, pilot plant test-work, mine-plan and a feasibility study and engineering work performed and/or commissioned by ESI.

In 2011, PRC contracted HRI to provide conceptual metallurgical testing to support a preliminary economic assessment and subsequently in 2012 to perform the metallurgical tests for the prefeasibility study. The tests included process development-related bench-scale and pilot scale investigations, primarily to recover K_2SO_4 and secondarily, Al_2O_3 from alunite. The current proposed process flowsheet for the extraction of SOP, H_2SO_4 and alumina consists of a number of integrated unit operations for processing ROM whole ore.

The following sections describe the extent of metallurgical testing performed to date which includes size-reduction by crushing, grinding and classification, drying, calcination and roasting of whole ore to decompose the alunite, leaching the calcine to extract the soluble SOP, solid/liquid separation to recover dissolved SOP in the brine (filtrate) from the leached slurry and evaporation of brine and crystallization to recover the product in granular form for markets.

13.2.1 Recent Developments in Metallurgical Testing

Since 2011, the historical information acquired from the alunite deposit for recovering alumina as the primary product and SOP as a by-product is being supplemented and refined by PRC. These efforts by PRC, in support of the Blawn Mountain Project, consist of additional exploration, mine-planning, comprehensive metallurgical testing and hydrological and environmental studies, with a focus on recovery of SOP as the primary product, concentrated sulfuric acid as a by-product, and γ-alumina as a potential co-product.

The comminution circuit will be designed to meet the dryer/calciner/roaster system feed design criteria of P_{80} at 1000µm (1.2mm maximum). Concurrent with bench-scale and pilot plant testing at HRI, PRC has enlisted the services of several laboratories, turnkey proprietary technology suppliers and equipment vendors to assist in process-optimization through additional testing of ROM ore samples, as required and desktop process simulation and modeling studies. Equipment vendors have provided recommendations on selection of equipment based on their world-wide experience in engineering and construction of similar commercial SOP production facilities.

13.3 HISTORICAL METALLURGICAL TEST RESULTS

Results of metallurgical experiments from the 2013 test program at HRI (T. J. Salisbury, June 24, 2013) including test-work subsequent to the June report from the 2011 and 2012 HRI test program (R. J. Mellon, May 21, 2012) are summarized in the following sections.

13.4 METALLURGICAL TESTING FROM 2011 THROUGH 2013

PRC contracted HRI to perform a process development study to validate and identify process design criteria for the proposed flowsheet consisting of a number of integrated unit operations. For this effort the following were selected:

- Bulk material excavated from Test Pit No. 5 located in Area 1. This was the same test pit used by ESI in the 1970's. The first shipment of 2 tons, identified as Bulk 1 was used in experiments conducted by HRI in June 2012.
- An additional 20t of bulk material from Test Pit No. 5, identified as Bulk 2, were received in March 2013.
- Also in 2013, eight PQ size drill holes were drilled for the purpose of collecting metallurgical test samples representing the main exploration areas. Core samples from the two different regions of the deposit, designated as Area 1 and 2 and identified as PQ were shipped to HRI.

The PQ core samples were stage-crushed and master composites labeled: Master Composite A (MC-A), Master Composite B (MC-B) and Master Composite C (MC-C) composed of two to three drill cores that were prepared for bench scale and pilot plant test-work. The large-sized PQ core is required for comminution testing, including crushing and JK drop weight testing for SAG mill sizing.

HRI has performed confirmatory test-work on bulk samples collected from Test Pit No. 5 and on PQ core composite samples collected from Blawn Mountain. Test Pit No. 5 was the deepest test pit and was located near the center of the envisioned starter pit for the mine. The objectives of the test programs were to confirm the results of the original test-work by HRI and to develop new design criteria for process development.

The test programs included the following tests:

- Ore characterization;
- Particle-size analysis;
- Head sample chemical analysis;
- Comminution test-work, including JK drop-weight tests, SMC tests and bond crushing, rod and ball millwork indices and abrasion indices;
- Calcination;
- Water leach testing;
- Evaporation and crystallization;
- Solid-liquid separation;
- Alumina processing.

13.5 MATERIAL CHARACTERIZATION

Table 13.2 describes Bulk 1 and Bulk 2 surface samples and the PQ core samples used by HRI in process development work.

Table 13.2 Description of Sample Materials*

HRI Project	HRI Identification	Short Description	Use
11468	HRI 53201	Bulk 1	Scoping flotation experiments, start-up material for pilot plant
11687	HRI 53456	PQ Core samples	Flotation, comminution, variability tests, pilot plant feed material
11687	MC-A MC-B MC-C HRI 53466-01	39, 40 (Area 1, South) 41, 42, 43 (Area 1 North) 53, 55, 56 (Area 2) Bulk 2	Mai pilot plant feed material

*T. J. Salisbury, June 24, 2013

Table 13.3 provides a summary of how the cores from drill holes were combined to generate the master composites samples. The sampling interval was 8ft linear down the drill hole, which is equivalent to two core boxes per sample. The average core density was 9.4lb/ft.

Table 13.3 Descriptions of Master Composites*

Master Composite	Area	PQ Hole	Number of Samples
MC-A	Area 1 (South) Area 1 (South)	PDH1-39 PDH1-40	104 53
MC-B	Area 1 (North) Area 1 (North) Area 1 (North)	PDH1-41 PDH1-42 PDH1-43	59 52 71
MC-C	Area 2 Area 2 Area 2	PDH2-53 PDH2-55 PDH2-58	36 25 38

*T. J. Salisbury, June 24, 2013

During 2013, a head sample of each type of material described in Table 13.2 was characterized by mineralogical studies and chemical and physical assays. Table 13.4 summarizes the results of XRD studies.

Table 13.4 Results of XRD Analysis of Minerals*

Sample		Mineral Constituents (Estimate based on peak heights)			
HRI	**ID**	**Major**	**Subordinate**	**Minor**	**Trace**
53021	Bulk 1	Alunite	Quartz	Hematite	Kaolinite Muscovite
53466	Bulk 2	Quartz Alunite			Kaolinite Hematite
53456	MC-A	Quartz	Alunite		Kaolinite Hematite
53456	MC-B	Quartz	Alunite		Kaolinite Hematite
53456	MC-C	Quartz	Alunite		Kaolinite Hematite

*T. J. Salisbury, June 24, 2013

Each sample was also analyzed using HRI's Bruker S8 Tiger XRF spectrometer (WDXRF) and a sample of Bulk 1 was sent to The Mineral Lab (Golden, CO) for a comparison analysis.

Table 13.5 summarizes the analyses for major elements, typically reported as oxides. Table 13.6 summarizes the trace metal analysis for Bulk 1 sample. The composite samples were not analyzed for trace elements using WDXRF methods.

Table 13.5 XRF Results of Analysis for Major Elements*

Constituent	Analysis (wt%)				
	Bulk 1		**MC-A**	**MC-B**	**MC-C**
	The Mineral Lab	**HRI**			
Na_2O	0.28	0.15	0.04	0.14	0.30
MgO	<0.05	0.15	0.05	0.11	0.27
Al_2O_3	21.0	21.0	14.1	15.6	15.3
SiO_2	43.4	39.5	58.7	57.4	61.3
P_2O_5	0.28	0.27	0.14	0.14	0.11
S	9.02	NA	NA	NA	NA
SO_3	NA**	19.2	NA	NA	NA
Cl	<0.02	<0.01	NA	NA	NA
K_2O	5.83	5.05	3.34	4.05	3.03
CaO	0.08	0.33	0.09	0.13	0.16
TiO_2	0.59	0.54	0.58	0.48	0.39
MnO_2	<0.01	<0.02	<0.01	<0.01	<0.01
Fe_2O_3	1.32	1.48	2.75	1.62	2.33
BaO	0.07	0.07	NA	NA	NA

*T. J. Salisbury, June 24, 2013.

Table 13.6 XRF Results of Analysis for Trace Elements*

Constituent	Element (ppm)	
	The Mineral Lab	HRI
V	88	100
Cr	124	<100
Co	<10	196
Ni	<10	180
W	<10	NA
Cu	97	280
Zn	<10	60
As	23	<100
Sn	<50	NA
Pb	35	<100
Mo	<10	<100
Sr	706	<100
U	<20	NA
Th	51	60
Nb	12	<100
Zr	155	170
Rb	19	<100
Y	13	30

*T. J. Salisbury, June 24, 2013; **NA = Not Analyzed.

13.6 CHEMICAL ANALYSIS OF MASTER COMPOSITES

The first shipment of approximately two tons of Bulk 1 material in June 2012, used in the initial experiments (HRI 53021, Project 11468), was analyzed only for alunite constituents (aluminum, sulfur and potassium) and for silicon as the primary gangue mineral. Table 13.7 summarizes the results of Bulk 1 head assays.

Table 13.7 Bulk1 Head Assays*

Analysis (wt%)			
Al	K	Si	S
10.3	4.7	21.5	7.83

*T. J. Salisbury, June 24, 2013.

Chemical analysis for 28 elements was performed on the Bulk 2 sample and the three master composites using a Perkin Elmer Optima 7300 DV inductively coupled plasma-optical emission spectroscopy (ICP-OES) analyzer. These samples were also analyzed for sulfur content using a LECO carbon-sulfur analyzer. Table 13.8 presents the results of the ICP-OES analyses for the major constituents of interest such as potassium, sulfur, aluminum, sodium, iron and titanium.

Table 13.8 ICP-OES Chemical Head Assays of Major Constituent Elements*

Sample ID	Analysis (wt%)					
	K	S	Al	Na	Fe	Ti
Bulk 2	4.47	NA	10.2	0.190	0.820	0.284
MC-A	2.74	5.47	6.82	0.388	1.88	0.306
MC-B	3.31	5.47	7.50	0.221	1.13	0.245
MC-C	2.48	4.12	7.25	0.343	1.57	0.201

* Adopted from T. J. Salisbury, June 24, 2013.

To confirm sodium, potassium, iron and silicon analyses, flame atomic absorption (AA) was used. A second ICP-OES analyzer was used to confirm the aluminum assay results. Table 13.9 summarizes the confirmatory assay results.

Table 13.9 Confirmation Assays*

Sample ID	Analysis (wt%)				
	Flame AA				ICP
	Na	K	Fe	Si	Al
Bulk 2	0.141	4.46	0.790	22.0	10.0
MC-A	0.357	2.95	2.01	27.3	7.32
MC-B	0.184	3.59	1.23	28.5	7.78
MC-C	0.330	2.38	1.68	28.9	7.64

* T. J. Salisbury, June 24, 2013.

A loss on ignition (LOI) test was also conducted at 1000°C (1,830°F). Table 13.10 contains a summary of the LECO sulfur and LOI assay results.

Table 13.10 LECO Sulfur and LOI Assay Results*

Sample ID	Analysis (wt%)	
	LECO	LOI
	S	at 1000°C
Bulk 2	7.35	23.1
MC-A	5.67	17.6
MC-B	5.66	18.4
MC-C	4.30	16.3

* T. J. Salisbury, June 24, 2013

Table 13.11 summarizes the calculated elemental conversion factors from the chemical assays to determine the alunite content of a given sample using the molecular formula of alunite, $(K_2SO_4.Al_2(SO_4)_3 4Al(OH)_3$.

Table 13.11 Calculated Alunite Content of Samples from Elemental Analysis*

Sample ID	Analysis (wt%)						
	ICPA			LECO	ICPb	Flame AA	Average Alunite
	S	Al	K	S	Al	K	
Bulk 1	NA**	NA	NA	50.5	52.7	49.7	51.0
Bulk 2	NA	51.7	47.4	47.4	51.2	47.3	49.0
MC-A	35.3	34.9	29.0	36.6	37.5	31.2	34.1
MC-B	35.3	38.4	35.1	36.5	39.8	38.0	37.2
MC-C	26.2	37.1	26.3	27.2	39.1	25.2	30.3

*T. J. Salisbury, June 24, 2013; **NA = Not Analyzed; [a] Building 7 ICP-OES analyzer; and [b] Building 1 ICP-OES analyzer

Table 13.12 summarizes the silica analyses for the five samples.

Table 13.12 Silica Analyses*

Sample ID	Analysis, as %SiO2
Bulk 1	46.0
Bulk 2	47.1
MC-A	58.4
MC-B	61.0
MC-C	61.8

*T. J. Salisbury, June 24, 2013; [a]Flame AA

Table 13.13 is a summary of the average values of angle of repose for various size fractions of the Bulk 2 and MC-A, MC-B and MC-C composite samples.

Table 13.13 Average Angle of Repose Measurements of Size Fractions*

Sample HRI	Sample ID	Size Fraction	Average Measurement (°)
53466	Bulk 2	¼ in.	30.3
		6 mesh	29.3
		10 mesh	34.5
53456	MC-A	1.5in	31.8
		0.5 in.	28.3
		6 mesh	34.5
	MC-B	3 in.	43.8
		1in	32.3
		0.5 in.	29.3
		6 mesh	34.8
		10 mesh	38.8
	MC-C	3 in.	40.8
		1in	34.3
		0.5 in.	32.0
		6 mesh	27.5
		10 mesh	34.3

13.7 SAMPLE PREPARATION

During 2013, HRI crushed and sampled two shipments of ore.

On January 30, 2013, HRI received a shipment of 28 pallets (29,000lb) of PQ core boxes from the drilling campaign from the summer of 2012. The large diameter core was drilled specifically for metallurgical testing and represents the main mineralization in Area 1 and Area 2 which are PRC-designated regions at the mine site. These samples were logged in under HRI 53456. The core samples were stage-crushed to different sizes and the samples were used in comminution, flotation and variability tests.

On March 6, 2013, 20 one-ton super-sacks of ROM material taken from Test Pit No. 5 in Area 1 on Blawn Mountain were delivered to HRI and logged in under HRI 53466. This material was stage-crushed, blended, then split to generate samples used during pilot plant testing.

In 2012, HRI received 4,800lb of ore as part of a 20t sample from Pit No. 5, which was stored in Utah for testing as follows:

- First, 20 pieces of samples 2 x 3in. size were set aside for comminution testing;
- Stage-crushed the as-received material to passing 1in, cone-and-quartered the crushed material to obtain three representative lots of 200lb;
- Further stage-crushed each of the 200lb lots to obtain 0.75 x 0.50in., minus .50in., and minus 6-mesh lots for studying the effect of particle size in the calcination step.

In 1972, HRI performed tests on a core composite sample, composite NGC-101, which was prepared by coning, quartering and splitting 110 bags (2,750lb) of ore from core holes which varied in depth from 10ft to 400ft. Assay heads were prepared on the as-received and minus 65-mesh composite.

13.8 COMMINUTION TESTS AND WORK INDICES

Reduction of particle-size in mineral processing is an energy-intensive operation. Work Index relates power consumption in crushing and grinding to the feed and product size distribution. During 2013, HRI completed a comprehensive comminution testing program utilizing the PQ core composite samples which included the following tests:

- JK drop-weight index (DW_i);
- Abrasion index (A_i);
- Bond ball millwork index (BW_i);
- Bond rod millwork index (RW_i);
- Bond impact work index (CW_i).
- Semi-autogenous grinding (SAG) mill comminution (SMC) testing.

The JK drop-weight test measures the breakage parameters of a rock sample, which are required to analyze or predict SAG mill performance. Table 13.14 summarizes the drop-weight breakage evaluations.

Table 13.14 Drop-Weight Breakage Evaluations*

Parameter	MC-A	MC-B	MC-C
Specific gravity	2.54	2.48	2.36
A - Maximum breakage	66.3	64.1	63.8
b – Relation between energy and impact breakage	2.65	1.41	2.31
A x b = Overall AG-SAG hardness	175.7	90.4	147.4
T_a = Abrasion parameter	1.36	0.77	0.84
Resistance to impact breakage	Very Soft	Soft	Very Soft
Resistance to abrasion breakage	Very Soft	Very Soft	Very Soft

*T. J. Salisbury, June 24, 2013; AG = Autogenous Grinding

The abrasion index is used to determine steel media and liner wear in the comminution circuit. Table 13.15 provides a summary of test results of abrasion index and work indexes.

Table 13.15 Summary of Ai, BWi, RWi, and CWi Results*

PRC ID	Ai, g	BWi, kWh/t	RWi, kWh/t	CWi, kWh/t
MC-A	0.4057	12.0	10.6	10.3
MC-B	0.4132	14.7	13.3	10.0
MC-C	0.4838	14.5	10.9	10.2

*T. J. Salisbury, June 24, 2013.

Table 13.16 is a summary of the semi-autogenous grinding mill comminution (SMC) test results.

Table 13.16 Summary of SMC Test Results*

Parameter	MC-A	MC-B	MC-C
Specific gravity	2.60	2.49	2.32
A - Maximum Breakage	79.8	70.9	75.4
b – Relation between energy and impact breakage	1.07	1.25	1.64
A x b = Overall AG-SAG hardness	85.4	88.6	123.7
DW_i = drop-weight index, kWh/m^3	3.04	2.8	1.88
DW_i %	17	15	8
M_{ia} = Coarse particle component, kWh/t	10.7	10.5	8.2
M_{ih} = High-pressure grinding roll (HPGR) component, kWh/t	6.8	6.5	4.7
M_{ic} = Crusher component, kWh/t	3.5	3.4	2.4
T_a = Low-energy abrasion component of breakage	0.85	0.92	1.38

*T. J. Salisbury, June 24, 2013; AG = Autogenous Grinding

13.8.1 Comminution Testing - 2012

Table 13.17 presents the results of tests performed in 2012 by HRI (D. W. Gillespie, March 1, 2012) on the bulk sample from Test Pit No. 5, which was assigned Hazen number 53021, to evaluate BWi, and CWi in kilowatt hours per ton (kWh/t), and Ai in grams (g).

Table 13.17 Work Indices BWi, CWi and Ai from HRI Test Results*

Parameter	BWi, kWh/t	CWi, kWh/t	Ai, g
HRI sample # 53021	5.9	7.06	0.2391

*D. W. Gillespie, March 1, 2012

- Twenty pieces of ore ranging from 2in to 3in were set aside and used to determine $CW_{i;}$
- Two kilograms of the .75 x .50in material saved were used to determine the A_i.
- Twelve kilograms of the 6-mesh material were used to determine BW_i.

13.9 MINERALOGICAL ANALYSIS

13.9.1 Mineralogical Analyses - 2013

Table 13.18 summarizes the mineral composition of the Bulk 2 Composite and Master Composite A, B, and C samples.

Table 13.18 Mineral Composition of the Composite Samples (wt%)

Minerals	Bulk 2 Composite	Master Composite A	Master Composite B	Master Composite C
Alunite	47.1	40.8	31.7	25.8
Quartz	49.5	52.9	58.3	62.4
Hematite	1.15	3.84	2.21	1.98
Kaolinite (Clay)	0.69	0.21	2.65	3.89
Sulfide minerals	0.09	0.24	0.10	0.11
Jarosite	0.00	0.51	0.07	0.01
Rutile/Anatase	0.69	0.80	0.76	0.54
Micas	0.19	0.20	2.04	2.75
Others*	0.59	0.49	2.33	2.45
Total	**100**	**100**	**100**	**100**

Notes: *Others includes Feldspars, Fe-Al silicates, Pyroxene, Calcite, Chalcopyrite, Pyrite, and trace amounts of other minerals.

13.10 CHEMICAL ANALYSIS 2012

In January 2012, HRI received 4,800lb of alunite ore from Pit No. 5 of Area 1 which was a collection of various sizes of rocks taken from 24 super sacs (R. J. Mellon, May 21, 2012). The sample was characterized by performing chemical and physical analyses, mineralogy and thermal analyses.

Results of analyses of two pulverized head splits for the major constituents of the ore are:

- 10.6wt% Al (aluminum);
- 4.6wt% K (potassium);
- 7.6wt% S (sulfur);
- 20.5wt% Si (silicon).

The remaining components were each <1.0wt%.

The XRD analyses of the head splits confirmed the major components of the ore as Quartz (SiO_2) and two forms of alunite, $K_{0.805}Na_{0.0132}(H_2O)_{0.063}Al_3(SO_4)_2(OH)_6$ and $KAl_3(SO_4)_2(OH)_6$. Analytical results of 2012 of 400g of the 6-mesh material were pulverized to produce two head samples, (Sample A and Sample B) are given in Table 13.19.

Table 13.19 Ore Analyses*

Analyte	Analysis (wt%)		
	Sample A	Sample B	Sample B (Repeat)
Al	**10.5**	**10.5**	**10.8**
Fe	0.87	0.90	0.89
K	**4.51**	**4.65**	**4.62**
Mg	0.018	0.018	0.018
Na	0.16	0.17	0.16
Si	**21.6**	**19.6**	**20.2**
S	7.64	**NA	**NA
Ti	0.30	0.31	0.31

* R. J. Mellon, May 21, 2012; **NA = not available

HRI also determined that particle-size distribution (PSD) for the 1in material and fractions were submitted for x-ray fluorescence and semi-quantitative (XRF-SQX) analysis to determine the analyte distribution within a given size fraction for the purpose of selecting an enriched fraction after crushing.

The XRF-SQX analysis is particularly suited for a high silica sample, which is difficult to dissolve, for Inductively Coupled Plasma Mass Spectrometry (ICPMS). XRF is a more economical test for samples where composition down to 0.01% (100ppm) is desired, the major advantages being that the sample does not need to be put into solution, as in ICPMS and no standards are required for semi-quantitative (SQX) analysis.

Table 13.20 and Table 13.21 summarize the distributions of the major and minor species within each size fraction. The potassium and aluminum contents show no trend (within analytical precision) among the particles sizes.

Table 13.20 XRF Results of Major Constituents by Particle-size Distribution*

Analyte	Analysis (wt%)					
	¾ x ½ in.	½ in. x 6-mesh	6 x 20-mesh	20 x 35-mesh	35 x 65-mesh	Minus 65-mesh
Al_2O_3	**19.7**	**18.7**	**19.3**	**18.5**	**19.8**	19.7
BaO	0.07	0.07	0.07	0.07	0.07	0.07
CaO	0.05	0.06	0.07	0.22	0.22	0.24
Cl	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Fe_2O_3	1.08	1.08	1.24	1.44	1.58	1.54
K_2O	**5.65**	**5.42**	**5.58**	**5.50**	**5.86**	**5.90**
MgO	<0.05	<e	<0.05	<0.05	<0.05	<0.05
MnO_2	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Na_2O	0.23	0.22	0.22	0.22	0.23	0.22
P_2O_5	0.25	0.26	0.28	0.26	0.25	0.25
S	**8.22**	**7.96**	**8.25**	**7.93**	**8.33**	**8.30**
SiO_2	**42.2**	**44.7**	**43.8**	**45.8**	**41.5**	**39.8**
TiO_2	0.57	0.54	0.56	0.60	0.52	0.50
Distribution, wt%	**37.1**	**30.1**	**13.8**	**6.8**	**7.4**	**4.839.8**

*R. J. Mellon, May 21, 2012

Table 13.21 XRF Results of Minor Constituents by Particle-size Distribution*

Analyte	Analysis (wt%)					
	¾ x ½ in.	½ in. x 6-mesh	6 x 20-mesh	20 x 35-mesh	35 x 65-mesh	Minus 65-mesh
As	28	27	25	25	37	43
Co	<10	<10	<10	<10	<10	12
Cr	106	82	89	110	80	91
Cu	23	19	41	31	40	59
Mo	<10	<10	<10	10	14	26
Ni	<10	<10	<10	<10	<10	<10
Pb	21	31	39	38	32	35
Sn	<50	<50	<50	<50	<50	<50
Sr	691	655	697	696	662	705
U	<20	<20	<20	<20	<20	<20
W	<10	<10	<10	<10	<10	<10
Y	91	81	88	93	91	97
Zn	<10	<10	<10	<10	<10	<10

*R. J. Mellon, May 21, 2012

13.11 THERMAL ANALYSIS

HRI performed (R. J. Mellon, May 21, 2012) Thermal Gravimetric Analysis (TGA) to understand the behavior of the ore as it is heated. TGA uses a highly sensitive balance to monitor and measure the weight loss of a sample of material as a function of temperature and time in a controlled atmosphere.

In this analysis, a sample of ore placed in the instrument was heated to 1000°C at a rate of 10°C/min and held at 1000°C. Time and weight loss with temperature, weight loss and rate of weight change were determined. The results indicate:

- Initially, the weight loss commenced at 505°C and produced a weight loss of 5.6% of the sample;
- The weight loss slowed between 560 and 720°C, at which point the total loss was 14%;
- An additional 8.3% weight loss began at 735°C and continued to 780°C;
- The final weight loss at the end of analysis was 25%;
- The first weight loss occurred in a typical range when hydroxyls decompose and water is evolved;
- The weight loss at 735°C is associated with the generation of sulfur oxides, which was confirmed through the SO_2 analyzer during the batch kiln runs.

According to HRI, the TGA confirmed that the proposed lower calcining temperature of 800°C would be sufficient to drive off the water and decompose the alunite.

During drying tests performed in 1973 by HRI, the as-received ore was found to be moist and became air-dried during sample preparation. The moisture content of five 20lb bag samples averaged 0.5% H_2O when dried at 100°C before stage-crushing to 65-mesh.

Table 13.22 summarizes observed weight loss during drying tests.

Table 13.22 Sample Drying Test Results*

Drying Temperature, °C	Weight Loss %
100	0.9
200	1.2
400	1.2
600	8.0
800	18.5

* F. J. Bowen, et al. April 12, 1973.

The drying tests indicated:

- Interstitial water was lost during drying through 400°C
- At 600°C both interstitial and combined water were lost along with small quantities of sulfur in off-gases;
- At 800°C, sulfur loss in off-gases was significant.

Microscopic examination of calcines obtained from 750°C to 900°C roasting step and their water-leach residues shows:

- An amorphous, intimate mixture of alumina and dehydrated alum phase;
- Optical evidence suggests that leachable K_2SO_4 might be entrapped in this phase;
- The presence of amorphous alumina on the exterior of the alum-alumina particles could prevent diffusion of soluble K_2SO_4 from the interior of the mixture.

13.12 CALCINATION EXPERIMENTS

During 2013, in addition to flotation concentrate, HRI evaluated whole ore as feed material to the roasters.

Previous investigations have shown that high-temperature roasting at greater than 1472°F (800°C) produced calcine with potassium extraction ranging from 67 to 87% during leaching with water. However, high temperature roasting results in changing the crystal structure of Al_2O_3 to the alpha form, which is not amenable to leaching with NaOH.

A benchmark of a successful roasting experiment is a significant extraction of the SOP into the leachate along with the formation of gamma-Al_2O_3, which is amenable to extraction from the leach residue by a caustic (NaOH) leach. The goals of the roasting process, therefore, are as follows:

- Roast the alunite ore at temperatures less than 1292°F (700 °C);
- Produce a calcine material that can be leached to a target 90% K extraction;
- Produce the gamma form of alumina that can be leached with NaOH.

HRI used the 4in diameter quartz batch kiln to minimize the amount of feed needed for roasting while still generating sufficient amounts of calcine for leaching. A total of 38 batch kiln campaigns were completed using both single-stage and three-stage roasting flowsheets.

The results of the calcining and water leach testing of the calcine at particle-sizes of 80% passing 80 and 1,000 microns are presented in Table 13.23.

Table 13.23 Results of Roasting and Water Leaching Tests

			Roast Conditions				Results			
Roast ID	Type	Particle-size P_{80}, µm	Temp, °C	Gas	Type	Mass, g	Parameter	Al	K	Leachable K[a]
BK-28	Whole Ore	<80	575/575/575	Air/Natural Gas/Air	4 in	400.2	Extraction, %	0	50	85
							Accountability, %	110	108	91
BK-28	Whole Ore	<80	575/575/575	Air/Natural Gas/Air	N/A	N/A	Extraction, %	0	59	NC
							Accountability, %	113	103	NC
BK-29	Whole Ore	<1500	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	68	NC
							Accountability, %	97	100	NC
BK-30	Whole Ore	<1000	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	76	NC
							Accountability, %	96	108	NC
BK-31	Whole Ore	<500	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	71	NC
							Accountability, %	97	102	NC
BK-32	Whole Ore	<300	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	70	NC
							Accountability, %	101	105	NC
BK-33	Whole Ore	<150	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	67	NC
							Accountability, %	97	104	NC
BK-30	Whole Ore	<1000	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	71	NC
							Accountability, %	91	101	NC
BK-34	Whole Ore	<1000	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	90	100
							Accountability, %	100	104	100
BK-35	Whole Ore	<1000	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	54	92
							Accountability, %	97	103	95
BK-36	Whole Ore	<1000	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	71	NC
							Accountability, %	102	104	NC
BK-1A	Whole Ore	1mm (crush)	575	N_2/Natural Gas	None	None	Extraction, %	0	85	NC
							Accountability, %	100	106	NC
BK-2A	Whole Ore	<0	575	N_2/Natural Gas	None	None	Extraction, %	0	69	NC
							Accountability, %	97	106	NC

During single-stage roasting at the low temperature of less than 1292°F (700°C), the off-gas composition was primarily a dense white phase with low (ppm) sulfur dioxide (SO_2) concentration. The white phase was assumed to be sulfur trioxide (SO_3), which was not detected by the array of gas analyzers but was collected in the ESP. When powdered sulfur was added to the kiln, the concentration of SO_2 in the off-gas was outside the 0 to 25% analyzer range.

Potassium extraction was marginal in the initial water leaching experiments of calcine from single-stage roast tests BK-1 and BK-13 conducted in 2013. Based on literature reviews, HRI identified an alternative method of roasting the alunite in a three-stage process. The first stage of roasting, or dehydroxylation, removes the hydroxyl (OH^-) groups associated with the alunite minerals.

After the dehydroxylation, the calcine was cooled and elemental sulfur was added to the calcine to reduce the sulfate (SO_4^{2-}) groups and convert the potassium into a leachable form while maintaining the alumina in a non-alpha phase. This reaction is given by the equation:

$$4KAl(SO_4)_2 + 3S \Rightarrow 2K_2SO_4 + 2Al_2O_3 + 9SO_2(g)$$

When the reduction reaction is complete, the excess sulfur is removed as SO_2 in an oxidation stage by the reaction:

$$S + O_2 \Rightarrow SO_2$$

During the 3rd quarter of 2013, FLSmidth Minerals Pyrometallurgy group in Bethlehem, PA was contracted to evaluate the most energy-efficient method of drying, calcining and roasting either alunite flotation concentrate or whole ore filter cake with an estimated moisture content of 10wt% and P_{80} of 1000µm. FLSmidth, based on desktop simulation studies and accrued information from similar projects, identified a dryer/calciner/roaster system consisting of the following major components:

Flash Dryer Circuit: The flash dryer portion of the system supports removal of the free moisture present in the alunite cake using a combination of flash calciner off gases and additional hot gas provided by a natural gas fired air heater.

Flash Calciner Circuit: The dried material from the flash dryer is delivered to the flash calciner for further heating to remove all the chemically bonded water and to decompose the alunite to potassium and aluminum sulfates. The material produced is transferred to a flash roaster so that it serves to complete the aluminum sulfate decomposition.

Flash Roaster Circuit: The pre-calcined material from the flash calciner is delivered to the flash roaster for further heating and to decompose all aluminum sulfate to aluminum oxide and SO_3. It also serves to convert most of the generated SO_3 to SO_2 to reduced conditions.

Flash Roaster Off Gas Circuit: Air-to-gas heat exchanges cool the process gas leaving the flash roaster cyclone at a temperature level suitable for downstream ESP. The ESP removes non-condensable particulate from the final system process off gas stream and an induced fan delivers excess gases to the acid plant after some amount of gases are recycled back to the flash roaster.

Cyclone Cooling Circuit: Four cyclones operating in series transfer heat from the flash roaster circuit product to the incoming ambient air stream.

Excess Air from Cooling Circuit: The excess air from the cyclone cooling circuit is delivered to a boiler to generate steam and cool down the gases to a temperature that is suitable for handling in the downstream bag house and induced draft fan.

Preliminary process design criteria for the dryer/calciner/roaster circuit, developed by FLSmidth and based on simulation and modeling of available information, are as follows:

- Solids content of filter cake as feed to dryer = 90%;
- Moisture content of filter cake as feed to dryer = 10%;
- Number of thermal processing units = 4;
- Name plate capacity of each thermal processing unit = 330tph;
- Alunite decomposition reactions are carried out at 1022°F (550°C);
- The maximum temperature in the calciner shall not exceed 1112°F (600°C);
- The residence time in the calciner is about two seconds.

13.12.1 Roasting Studies – 2012

HRI conducted 18 calcination experiments (R. J. Mellon, May 21, 2012) in a batch quartz kiln system. The kiln, 44in overall length, consisted of a 4in diameter by 18in long quartz reactor section, four rows of lifters pressed into the sidewall at 90º intervals for mixing the charge, a direct current motor for controlling the rotation of the kiln between 1 and 5rpm, a type K thermocoupler that resided within the bed material, a programmable temperature controller that varied the furnace electrical input for a given zone and a gas handling train.

Typically, the kiln loaded with 1 to 2kg of alunite ore was placed inside an electrically-heated furnace and the thermocouple positioned into the material. In the first group of 12 experiments, the ore was ground to minus 0.50in. and the retention time (30, 60, 90, and 120 minutes) and temperature (800, 850, and 900°C) were varied to study their effects on extraction of potassium and aluminum during leaching with water.

In the remaining six kiln experiments, three were performed on the ore crushed to minus 6-mesh and three on the ore sized to 0.75 by 0.50in. to study the effect of ore particle-size on potassium extraction.

In all experiments, the O_2 concentration in the exhaust gas consistently increased or decreased as the concentration of SO_2 increased or decreased. Peak concentration of SO_2 was as high as 31vol.% (dry basis) but at the end of the experiment typically was less than 2 vol.% SO_2. It is inferred that the concentration of O_2 at 21vol.% in off-gases during the air purge of the kiln resulted in the observed peak concentration of SO_2.

The above data suggest that in all probability, sulfur trioxide (SO_3) evolved during calcination of alunite and is partially decomposed in the hot kiln to SO_2 and O_2 according to the following reaction:

$$\boxed{SO_3 \Leftrightarrow SO_2 + \tfrac{1}{2} O_2}$$

Off-gases from the kiln were cooled downstream from the furnace to remove water vapor. Downstream from the condenser, a slipstream of gas was directed to CO_2, CO, O_2 and SO_2 analyzers. Downstream of this equipment, gases passed through bubblers containing NaOH solution to remove SO_2 evolved from heating the ore. Gas analyses and thermocouple readings were continuously recorded to a digital file on a computer.

13.13 LEACHING POTASSIUM FROM THE ORE

In 1973, Hazen conducted water leach tests at 25°C on calcines produced at 800°C in a rotary Vycor Retort by agitating the slurry containing 50wt% and 17wt% solids for one hour to dissolve the potassium sulfate. The leaching cycle consisted of leaching with water as the lixiviant followed by filtration and washing the filter cake with water. The water leach tests showed that:

- About 94% of the potassium and 88% of the sulfate in the calcine can be dissolved in water under the best roasting conditions. The solubility of alumina was 3 or 4%. The leach residue assayed about 0.3%K and 1.5%SO_4;
- As much as 2.5 percentage points more of potassium sulfate was dissolved at 85°C than was dissolved at 25°C;
- No significant difference was seen in the leachability of calcine produced from 3-, 8-, 28- and 65-mesh materials.

13.13.1 Recent Leach Studies – 2012

During December 2016, tests are underway at HRI to dry-crush the run-of-mine ore and calcine at 575ºC minus 1mm size fraction, followed by leaching the quenched calcine with water at 80ºC for 30 minutes in agitated tanks.

During 2012, extraction of potassium sulfate was investigated by HRI using a two-step approach consisting of calcining the ore followed by leaching the calcine with water. The ore was calcined in a kiln under various temperature and residence time conditions to determine the effects of these variables on extraction of potassium in the subsequent leach step (Mellon, Robert J. May 21, 2012). The proposed calcination reaction at 1472 to 1652°F (800 to 900°C) is as follows:

$$\boxed{2KAl_3(SO_4)_2(OH)_6 \rightarrow K_2SO_4.3Al_2O_3 + 3SO_3 + 6H_2O}$$

Water and sulfur (as SO_3) are driven off during calcination and water-leachable K_2SO_4 is left behind. The aluminum is converted into an oxide that can be recovered in subsequent processing.

HRI performed eighteen calcination experiments during 2012, in a batch quartz kiln system in which the retention time (30, 60, and 90 minutes) and temperature (800, 850 and 900°C) were varied. Approximately 200g of samples from each of the calcines produced in the batch kiln were leached with water. The leach equipment consisted of a 2L kettle, a condenser, a heating mantle, a variable speed mixer, a thermometer, a vacuum flask, a Buchner funnel with Whatman paper (#5 or #50), and sample bottles.

The test conditions and results for three sized samples calcined and leached with water in 2012 are summarized in Table 13.24.

Table 13.24 Summary of Calcining and Leaching Conditions and Results*

Conditions	¾ by ½ in.	Minus ½ in.	Minus 6-mesh
Calcine temperature, °C	850	900	900
Residence time, minutes	60	60	90
Potassium extraction, %	87	86	85
Aluminum extraction, %	1.3	0.4	0.2
Sulfur evolution, %	63	70	70

*R. J. Mellon, May 21, 2012

HRI evaluated the effectiveness of calcination by evaluating the recovery of potassium during leaching. Potassium recovery was calculated by analyzing the primary filtrate and wash solutions. A mass balance was performed for each analyte around the calcination and leach operations and the closure of the balance is the "mass accountability," which is labeled as "Bal" in Table 13.25. Analytical precision and unavoidable material losses can cause deviations from 100% accountability. Table 13.25 summarizes the results of eighteen (18) batch kiln calcination and calcine leach experiments.

Table 13.25 Results from Batch Kiln Calcination Tests*

Expt. No.	Calcination			Calcine, %				Leach, %	
				Potassium		Aluminum		K	Al
	Ore Size	Time, Min	Temp. °C	Loss[B]	Bal[C]	Loss[B]	Bal[C]	Ext[D]	Ext[D]
1	-1/2 in.	30	800	0.1	93.4	0.01	92.7	82.7	9.3
2	-1/2 in.	30	850	0.1	97.1	0.02	91.4	84.2	7.2
3	-1/2 in.	60	800	0.9	93.1	<0.01	93.0	85.3	2.7
4	-1/2 in.	90	800	1.3	103	0.01	92.1	82.6	2.1
5	-1/2 in.	120	800	1.4	99.6	0.01	88.8	81.3	1.2
6	-1/2 in.	60	850	1.4	99.3	<0.01	102	82.4	0.6
7	-1/2 in.	90	850	1.4	97.7	<0.01	94.0	81.0	0.3
8	-1/2 in.	120	850	1.3	83.5	<0.01	94.8	86.8	0.2
9	-1/2 in.	60	900	1.3	94.3	<0.01	99.4	85.8	0.4
10	-1/2 in.	30	900	1.4	96.5	<0.01	94.8	84.9	0.4
11	-1/2 in.	120	900	1.3	89.1	<0.01	91.7	82.7	<0.01
12	-1/2 in.	90	900	1.3	93.6	<0.01	93.5	83.6	<0.01
13	¾ x ½ in.	30	800	0.5	77.5	0.01	74.7	67.4	1.9
14	- 6 mesh	30	800	0.7	98.6	0.2	101	81.8	12.4
15	- 6 mesh	60	850	0.8	100	0.3	104	83.3	1.1
16	- 6 mesh	90	900	0.7	91.8	0.01	92.3	85.0	0.2
17	¾ x ½ in.	60	850	0.5	101	0.01	102	86.6	1.3
18	¾ x ½ in.	90	900	0.5	99.9	0.01	98.6	86.3	0.2

*Mellon, Robert J. (May 21, 2012)

All leaches were conducted at 90°C; 60 minutes residence time; 20% solids; agitated. When evaluating the water leach test results, both potassium and aluminum extractions must be taken into consideration. An inspection of test results shows:

- Calcine produced at 850°C resulted in one of the highest potassium extractions of 87% and aluminum extraction of 1.3%;
- Calcining the ore at 900°C typically resulted in comparable potassium extraction in the 85 to 86% range with the added benefit of lower aluminum extraction of 0.2 to 0.4%;
- Processing the ore at a relatively high temperature (900°C) with resulting lower aluminum extraction is beneficial in the recovery of purified K_2SO_4 in downstream unit operations.

13.13.2 Recent Leach Testing 2013

During 2013, HRI completed a total of 22 leaching experiments with calcine generated from the roasting process to determine the staged roasting concept, establish operating conditions for the leaching stage and produce brine or leach liquors for the crystallization experiments.

During 2013, a series of water leach tests were completed on calcines generated using reductants such as sulfur, carbon monoxide/carbon dioxide, hydrogen sulfide and natural gas.

The results of the most recent tests run at 80 and 1000um are presented in Table 13.23. Sulfur and natural gas were found to give the best overall calcine-leach results.

The results were used to calculate the extraction of “extractable” potassium, which is associated with the alunite. Some potassium has been identified to be associated with minerals other than alunite, such as K-feldspar (Orthoclase) and kaolinite clays. It is estimated that more than 90% of the potassium associated with alunite can be extracted by leaching the calcine with water. Natural gas has been identified as the reductant of choice because of its performance over sulfur.

A series of seven calcine leach tests were performed using Master Composite B to evaluate the effect of particle-size of the feed on calcine-leach performance. The tests were performed with natural gas as the reductant. The feed size varied between 100 and 1500μm in five batch kiln tests, followed by leaching the calcine with water. The results of the seven tests were used to select the optimal feed size and a single test was then run on Master Composite A and Master Composite C to confirm the results.

Preliminary test results indicated that potassium can be leached from calcine feed with a particle-size less than 80% passing 1.0mm without a reduction in leach recovery. Master Composite B and Master Composite C have a higher percentage of potassium and aluminum associated with non-alunite minerals, including K-feldspar and kaolinite clays than master Composite A.

During September 2013, HRI completed two tests on Master Composite A and Master Composite B without the oxidation step and leaching the calcine with water at 35wt% solids. The results achieved are similar to those in the previous experiments.

13.13.3 Leaching Fresh Calcine with Recycled Leach Liquor

Process design for the commercial plant envisions recycling the filtrate, to leaching to build up the concentration of the filtrate pumped to the evaporator/crystallizer circuit, which is an effective method of conserving both energy and water. Additionally, the wash water from the filter cake wash cycle will also be reused for quenching the calcine and/or in the leach circuit.

HRI conducted an experiment in 2012 on a portion of the calcine (Table 13.34) generated from Experiment No. 4 (calcining at 800°C and 90 min. leaching) to determine the effect of recycling leachate to leach fresh calcine.

Fresh calcine was leached with the recycled leach liquor at 90°C, 20% solids and 90 min residence time to determine the changes in concentration of the ions in the freshly generated leachate. Four cycles of contacting fresh calcine with recycled leachate were performed. During each cycle, an aliquot of the leach liquor was analyzed for potassium, aluminum, titanium, phosphorus, silicon, sodium, iron, magnesium and sulfate. After the fourth cycle, potassium and sulfate concentrations increased from the initial leach by a factor 3 and 3.5-fold, respectively.

Table 13.26 summarizes the effect of recycling the leach liquor on extraction of potassium and aluminum.

Table 13.26 Potassium and Aluminum Extractions by Cycle*

Cycle	Extraction, %		K/Al Mass Ratio
	K	Al	
1	83	2.2	17.2
2	81	2.0	17.1
3	89	3.2	18.8
4	87	3.4	17.9

*Mellon, Robert J. May 21, 2012.

A ratio of potassium to aluminum is an indicator of increases in aluminum concentration as a function of the cycle. The data in Table 13.25 indicates that in successive contacts of the leachate with fresh calcine, up to four cycles, does not appear to affect the potassium extraction. Based on the mass ratio of potassium to aluminum, the concentration of aluminum is not increasing with respect to potassium over the four cycles.

The columns labeled “Ext” in Table 13.25 indicate potassium or aluminum extractions from leached calcines and are calculated by accounting for all masses of species in the solution and dividing it by the mass in solution, plus un-leached mass reporting to the leach residue.

During calcination, potassium losses ranged from 0.1 to 1.4% as dust entrained in the off-gases from the kiln.

In the FLSmidth gas suspension calciner (GSC), or in any dryer/calciner/roaster system, the off-gases will be cleaned by a train of cyclones and electrostatic precipitators (ESPs) and must be dust-free before being ducted to the sulfuric acid plant. The dust collected and containing potassium and/or alumina will be recycled to the water leach circuit, thus minimizing loss of values.

13.13.4 Effect of Roasting and Leaching Conditions on Potassium Sulfate Recovery

The 2012 HRI experiments also investigated the effect of leaching temperatures, other than 90°C on extraction. Potassium extraction was only 61% when calcine, generated at 800°C, was leached at 25°C (room temperature) for 30 minutes at 20% solids. Aluminum extraction was 4%. Comparable leaches conducted at 90ºC achieved 83% extraction for potassium and 9% for aluminum (Table 13.27).

The 1973 calcining and leaching tests results also compare favorably with those from the 2012 experiments:

- Significant increases in solubility of potassium and sulfate (SO_4) did not occur until the alunite ore was roasted at 750°C;
- The solubility of potassium peaked when roasting was at 800°C;
- At roasting temperature of 900°C, the solubility of potassium (84.7%) was about 5% less than that obtained (89.4%) at 800°C roasting temperature.

For a historical perspective, the 1973 roast/leach test results are summarized in Table 13.27. The 1973 historical test results compare favorably with those from the 2012 test results.

Table 13.27 Effect of Roasting and Leaching Conditions on Percent Extraction of Potassium and Sulfate*

Test No.	Roasting Temperature, °C	Leaching Temperature, °C	Roasting Weight Loss, %	% Dissolved	
				K	SO_4
Series 1					
T-2	510	25	5.0	4.3	2.9
T-3	750	25	12.8	47.9	30.2
T-4	800	25	18.2	89.4	87.1
T-5	850	25	19.3	86.4	85.9
Series 2					
T-14	0	85	0.0	0.5	<0.1
T-7	800	85	18.0	92.5	87.1
T-8	850	85	19.3	88.6	83.9
T-9	900	85	20.0	84.7	89.2

* F. J. Bowen, et al. April 12, 1973.

Metallurgical tests in 1973, based on filtrate analyses, suggest that the amount of alumina and sulfate dissolved is dependent on the duration of the leach cycle.

The results of influence of leaching time on percentages of dissolved alumina and sulfate in the filtrate from water leaching of high temperature calcine produced at 800°C at 50wt% solids and 80 to 90°C are given in Table 13.28. The results indicate that the alumina fraction in the calcine dissolves in water rapidly, but is re-precipitated with time.

The soluble alumina will report as alum during the subsequent step of potassium sulfate recovery from the leach liquor.

Table 13.28 Influence of Leaching Time on Alumina and Sulfate Dissolution*

Leaching Time, Minutes	% Dissolved	
	Al_2O_3	SO_4
15	5.2	80.0
60	3.3	80.5
180	2.1	60.0

* F. J. Bowen, et al. April 12, 1973.

Table 13.29 and Table 13.30 summarize the results of leaching tests at HRI in 1973 for establishing the influence of leaching parameters on percent extraction of potassium.

Table 13.29 Percent Potassium Extraction as a Function of Percent Solids, Leaching Temperature and Leaching Time*

Hazen Leaching Test Results			
Temperature, °C	Leaching Time, Minutes	Solids, %	Potassium (K) Extraction, %
Room Temperature	60	50	65.2
90	60	50	80.5
100	60	17	74

* F. J. Bowen, et al. April 12, 1973.

Table 13.30 Influence of Leaching Time on Water Leaching of High Temperature (800°C) Calcine*

Leaching Time, Minutes	Wt.% Dissolved	
	Al_2O_3	SO_4
15	5.2	80.0
60	3.3	80.5
180	2.1	60.0

* F. J. Bowen, et al. April 12, 1973.

The effects of both calcining temperature and residence time on potassium and aluminum extractions should be considered when selecting a commercial calciner and evaluated in terms of associated capital and operating costs.

13.14 LEACHING ALUMINA

Extraction of SOP is the primary objective of the Blawn Mountain Alunite Project. The tailings produced at the end of the calcine leach cycle contain alumina, a potential resource for aluminum. During 2012, HRI conducted two scoping studies to determine the extractability of aluminum from the tailings:

The hot (80°C) water-leach residue, derived from leaching material calcined at 850°C for 60 minutes, was leached in 25wt% sulfuric acid at 90°C for 60 minutes, cooled to 50°C, and filtered through a Whatman 541 filter paper. Approximately 70% of the aluminum was extracted from the tailings. H_2SO_4 as a lixiviant shows promise for extracting aluminum from alumina-bearing tailings. The tests confirm that the alumina contained in the SOP leach residue produced by low temperature calcination is soluble in hot caustic solutions.

HRI conducted a series of leach tests during 2013 to confirm that the alumina contained in the SOP leach residue is in a form that is soluble in hot sodium hydroxide solutions. The SOP leach residues were produced from low temperature calcining of master composite samples. The results of the preliminary SOP leach residue, alumina, leach tests are summarized in Table 13.31.

Table 13.31 Results of the SOP Leach Residue, Alumina, Leach Tests

Master Composite	SOP Leach Residue Sample No.	Particle-size, 80% Passing (um)	NaOH Concentration (g/L)	Temperature, Degrees C	Aluminum Recovery, %
MC-B	BK–31	500	200	60	77
MC-A	BK-34	1000	200	60	91
MC-C	BK-35	1000	200	60	63
MC-B	BK-36	1000	200	60	72

13.15 EFFECT OF PH ON LEACHING CALCINE

HRI selected minus 8-mesh calcine (F. J. Bowen, et al. April 12, 1973) produced in the 6in screw reactor during high temperature Test SR-13 to study the effect of pH of the lixiviant (leaching medium). Table 13.32 summarizes the characteristics of the calcine produced during high temperature Test SR-13.

Table 13.32 Characteristics of the Calcine from Test SR-13*

Parameter	Test SR-13
Roasting:	
Temperature, °C	850
Time, minutes	60
Purge gas	Air
Feed, mesh	8
Leaching Temperature, °C	85
Calcine analysis, wt%:	
K	4.01
SO_4	12.5
Al_2O_3	19.1
Leach weight loss, wt%	14.4
Dissolution, wt%:	
K	86.6
SO_4	76.6
Al_2O_3	12

*F. J. Bowen, et al. April 12, 1973

Three parallel leaching tests were performed by Hazen at 85°C using the calcine from high temperature test SR-13. In these leaching tests, the pH of the leach solution was raised from its normal level of pH 4.0 to pH 6.0, 8.0, and 9.0. The pH adjustment was made using ammonium hydroxide (NH_4OH). The leaching test results show that:

- When leaching the calcine with water at 17wt% solids, the potassium-rich liquor had a terminal pH of about 4.0;
- The solubility of potassium and sulfate increased by about seven percentage points when pH was raised from 4.0 to 6.0 with ammonium hydroxide;
- Little difference in the solubility of both potassium and sulfate when leaching at pH 6.0, 8.0, or 9.0;
- At all pH-adjusted levels, about 94% of the potassium and about 95% of the sulfate were dissolved.

13.15.1 Sulfate in Potassium Leach Residues

In 1973 HRI performed high (≈800°C) and low (≈600°C) temperature roasting studies on alunite composite sample NGC-101, followed by leaching the calcines with water (F. J. Bowen, et al. April 12, 1973) to determine the solubility of potassium and sulfate from the resultant calcines. The conclusions based on assays are follows:

- Water leached residues of calcines produced in the Vycor Retort contain sulfur in the sulfate form;
- The water-leached residues assayed at least 1.2% and as much as 3.4% SO_4 regardless of the potassium solubility over the range 85 to 94%;

- Approximately 18% of the SO_4 originally present in the calcine was present in the water-leached residue;
- Water-leached residues of calcines produced in the 6in screw reactor and from which about 90% of the potassium was dissolved assayed 1.8 to 2.7% SO_4, which represented about 18% of the SO_4 originally present in the calcine.

13.16 EFFECT OF LEACHING TEMPERATURE

Leaching of calcines with water experiments during 2012 had been conducted at 90°C. To evaluate the effect of leaching temperature on extraction, a sample of calcine generated at 800°C and 30 minutes residence time was leached at room temperature (25°C).

Leaching at 25°C produced a potassium extraction of only 61% and 4% for aluminum. Comparable leaches conducted at 90°C resulted in 83% extraction for potassium and 9% for aluminum.

Additional leaching experiments between 25°C and 90°C will be required to evaluate the effects of leaching temperature on extraction of potassium and aluminum as well as other impurities.

13.17 RECENT SOLID LIQUID SEPARATION TESTING 2013

Leach residue or tailings samples were generated from a series of batch calcine leach tests for solid liquid separation testing. 15kg samples of each composite were shipped to Pocock Industrial in Salt Lake City, Utah for sedimentation and filtration studies. The results of the tests are included in the appendices for this section.

13.17.1 Tests Performed on Calcine Leach Slurry

In October 2013, Pocock Industrial performed preliminary conventional thickening and rheology tests on Composite A-, Composite B-, and Composite C leach slurry samples. The results of these tests are shown on Table 13.33.

Table 13.33 Status Summary of Tests Performed

Test Performed	Sample Tested		
	Composite A Leach	Composite B Leach	Composite C Leach
Sample characterization	Yes	Yes	Yes
Flocculant screening	Yes	Yes	Yes
Static thickening tests	Yes	Yes	Yes
Dynamic thickening tests	Yes	Yes	Yes
Pulp rheology	Yes	Yes	Yes
Vacuum filtration	Yes	Yes	Yes
Pressure filtration	Yes	Yes	Yes

13.17.2 Physical Properties of Calcine Leach Slurry

Table 13.34 is a summary of the physical properties of the samples tested.

Table 13.34 Physical Properties of Samples

Property	Sample tested		
	Composite A Leach	Composite B Leach	Composite C Leach
Liquid specific gravity	1.04	1.05	1.05
Solids specific gravity	2.76	2.78	2.90
pH	5.5	5.6	5.2

The flocculant screening tests indicate Hychem AF 304, a medium-to-high molecular weight, 15% charge density, anionic polyacrylamide as the reagent of choice in solid/liquid separation operations. The tests were performed by diluting the pulp to a solids concentration likely to be encountered in the feed to the thickener. The tests were based on the amount or dosage of flocculant required to initiate pinpoint floccule formation, effectiveness in capturing fines, the stability of the floccule, and resultant quality of the supernatant.

13.17.3 Flocculant Screening Tests for Calcine Leach Slurry

Table 13.35 is a summary of the results of flocculant screening tests.

Table 13.35 Results of Flocculant Screening Tests

Parameter	Sample Tested		
	Composite A Leach	Composite B Leach	Composite C Leach
pH	5.5	5.6	5.2
Temperature, °C	20	20	20
Initial solids concentration of slurry tested, wt%	30%	20%	30%
Maximum effective dosage range, g/MT	20 – 30	25 – 35	30 – 40
Flocculant concentration*, g/L	0.1	0.1	0.1
Flocculant selected	Hychem AF 304	Hychem AF 304	Hychem AF 304

*Note: Concentration of flocculant in solution before contact with pulp.

13.17.4 Thickening Calcine Leach Slurry

Conventional (static) thickening tests of calcine leach slurry at various flocculant doses and feed solids concentration (pulp density) were performed to select the parameters for conventional-type thickener design and to determine the operating conditions for high-rate (dynamic) thickening tests. Table 13.36 summarizes the thickening test results.

Table 13.36 Thickening Test Results

Parameter	Sample Tested		
	Composite A Leach	Composite B Leach	Composite C Leach
Flocculant	Hychem AF 304	Hychem AF 304	Hychem AF 304
Flocculant dose, g/MT	20 – 30	25 – 35	30 – 40
Maximum thickener feed Solids, wt%	25 – 35 (Conventional)	30 – 40 (Conventional)	20 – 30 (Conventional)
Maximum unit area for conventional thickener sizing, m^2/MtPD	0.125	0.160 – 0.205	0.125
Estimated underflow density for standard thickener, %	69 – 73	61 – 65	69 – 73

- The static thickening test results (unit area basis) are used in the design of conventional thickeners;
- The dynamic thickening test results (hydraulic net feed loading basis) are used in the design of high-rate thickeners;
- Flocculant should be diluted to 0.1 – 0.2g/L with thickener overflow before contacting with pulp.

13.17.5 Pulp Rheology

Viscosity tests were performed in October 2013 to examine the rheological behavior of the thickened pulp across a specific shear range to correlate the relationship between apparent viscosity (Pa.sec) and shear rate (sec^{-1}), shear stress (N/m^2) and shear rate (sec^{-1}) at operating temperatures, grind size, solids concentration, residual flocculant, and pH.

Pulp viscosity data was collected using the Fann (Model 35A) Viscometer. The long-chain molecular structure of the flocculant was destroyed by shearing the underflow materials in a laboratory mixer before viscosity tests were performed. The pre-sheared data collected using the Fann Viscometer provides necessary information required to determine the maximum design underflow densities (% solids concentration) for standard conventional and high-rate thickeners.

“Pseudoplasic fluids” display decreasing apparent viscosity with increasing shear rate, or “shear-thinning” flow behavior. The thickener underflow pulps examined for the solids concentration range tested exhibited this behavior and therefore, identified as belonging to the pseudoplastic class of “non-Newtonian fluids.” In a Newtonian fluid, the relation between the shear stress and the shear rate is linear. In non-Newtonian fluids, the viscosity or the flow behavior changes with stress and is dependent on shear rate.

The maximum recommended operating underflow pulp densities for *standard thickener sizing* based on the rheology data are summarized below:

- Composite A Leach: 69% to 73%
- Composite B Leach: 61% to 65%
- Composite C Leach: 69% to 73%

13.17.6 Horizontal Belt Vacuum Filter

Vacuum filtration design criteria common to samples Composite A, Composite B and Composite C from tests performed in October 2013, are summarized below:

- Filter feed solids (total suspended solids) = 71.3%
- pH of slurry = 7.2
- Temperature of feed slurry = 20°C
- Vacuum level = 67.7 kPa (9.8psi)
- Single stage countercurrent wash

Table 13.37 summarizes the horizontal belt vacuum filter test results for the three samples under four different operating conditions.

Table 13.37 Horizontal Belt Vacuum Filter Test Results

Parameter	Sample Tested: Composite A Leach	Composite B Leach	Composite D Leach
Test #1:			
Flocculant, g/Mt	None	None	None
Cake thickness, mm	10	10	10
Cake moisture, %	17.8	17.7	17.8
Dry bulk density, kg/m^3	1834.8	1648.22	1799.39
Wet bulk density, kg/m^3	2233.29	2001.84	2190.13
Total cycle time, min.	1.48	3.51	3.74
Production rate, kg/m^{2}hr	593.7	225.39	231.02
Test #2:			
Flocculant, g/Mt	None	None	None
Cake thickness, mm	15	15	15
Cake moisture, %	18.7	18.6	19.0
Dry bulk density, kg/m^3	1834.8	1648.22	1799.39
Wet bulk density, kg/m^3	2258.21	2025.63	2222.04
Total cycle time, min.	2.71	7.27	7.79
Production rate, kg/m^{2}hr	486.89	163.18	166.41
Test #3:			
Flocculant, g/Mt	70	75	70
Cake thickness, mm	10	10	10
Cake moisture, %	20	22.1	20.9
Dry bulk density, kg/m^3	1345.68	1385.93	1357.10
Wet bulk density, kg/m^3	1682.95	1779.96	1715.05
Total cycle time, min.	0.59	0.56	0.55
Production rate, kg/m^{2}hr	1089.30	1185.09	1189.22
Test #4:			
Flocculant, g/Mt	70	75	70
Cake thickness, mm	15	15	15
Cake moisture, %	21.1	23.5	22.5
Dry bulk density, kg/m^3	1345.68	1485.93	1357.10
Wet bulk density, kg/m^3	1706.38	1810.90	1751.50
Total cycle time, min.	0.71	0.64	0.61
Production rate, kg/m^{2}hr	1363.37	1564.18	1607.67

13.17.7 Automatic Pressure Filter

Two sets of automatic pressure filter tests were performed on each of Composite A, Composite B and Composite D samples. The results are summarized for the three samples, respectively, in Tables 13.38 through 13.40.

Table 13.38 Automatic Pressure Filter—Composite A Sample Overhead Beam GHT – 2000mm – P/10

Parameter	Test #1	Test #2
Feed solids, %	69.7	69.7
pH	7.2	7.2
Air blow	Yes	Yes
Air blow & Squeeze	None	Yes
Feed temperature, °C	20	20
Feed pressure, kPa	551.6 kPa (80 psi)	551.6 kPa (80 psi)
Cake thickness, mm	60 (Full Cake)	60 (Full Cake)
Dry bulk density, kgm^3	1227.65	1472.02
Wet bulk density	1401.59	1662.40
Moisture content, %	**12.4**	**11.5**
Sizing basis, m^3/Mt	1.018 (Note #1)	0.849 (Note #1)
Filtration rate based on Total cycle time, kg/m^2hr	92.4	92.4
Filtration rate based on 30mm Cake + Dry time, $kg/m^2.hr$	463.9	509.6
Cake form time, min	0.81	1.16
Wash time, min.	0	0
Air blow/Squeeze time, min.	3.0	3.0
Miscellaneous time, min	10.0 (Note#2)	10.0 (Note #2)
Total cycle time, min.	16.0	16.0
Number of filters	1.0	1.0
No. of cycles per 20-hr day	75	75
Metric tons/day/filter	1192.0	1192.0
Metric tons/cycle required	15.9	15.90
M^3/cycle required	16.19	13.50
No. of recess plate chambers	89	74

Notes: #1 Sizing basis in m^3 of pressure filter volume per metric ton (Mt) of dry solids, including 1.25 scale-up factor.
#2 Miscellaneous time of 10 minutes for cake discharge and cloth washing.

Table 13.39 Automatic Pressure Filter—Composite B Sample Overhead Beam GHT – 2000mm – P/8

Parameter	Test #1	Test #2
Feed solids, %	69.1	69.1
pH	7.2	7.2
Air blow	Yes	Yes
Air blow & Squeeze	None	Yes
Feed temperature, °C	20	20
Feed pressure, kPa	551.6 kPa (80 psi)	551.6 kPa (80 psi)
Cake thickness, mm	60 (Full Cake)	60 (Full Cake)
Dry bulk density, kg.m^3	1760.48	2047.22
Wet bulk density	2052.71	1662.40
Moisture content, %	**14.2**	**13.1**
Sizing basis, m^3/Mt	0.710 (Note #1)	0.611 (Note #1)
Filtration rate based on Total cycle time, kg/m^2.hr	128.1	138.5
Filtration rate based on 30mm Cake + Dry time, kg/m^2.hr	386.6	377.4
Cake form time, min	3.56	4.81
Wash time, min.	0	0
Air blow/Squeeze time, min.	3.0	3.0
Miscellaneous time, min	10.0 (Note#2)	10.0 (Note #2)
Total cycle time, min.	16.56	17.81
Number of Filters	1.0	1.0
No. of cycles per 20-hr day	72.5	67.4
Metric tons/day/filter	1192.0	1192.0
Metric tons/cycle required	16.45	17.69
M^3/cycle required	11.68	10.80
No. of recess plate chambers	64	60

Notes:
#1 Sizing basis in m^3 of pressure filter volume per metric ton (Mt) of dry solids, including 1.25 scale-up factor.
#2 Miscellaneous time of 10 minutes for cake discharge and cloth washing.

Table 13.40 Automatic Pressure Filter—Composite D Sample Overhead Beam GHT – 2000mm – P/8

Parameter	Test #1	Test #2
Feed solids, %	70.7	69.1
pH	7.2	7.2
Air blow	Yes	Yes
Air blow & Squeeze	None	Yes
Feed temperature, °C	20	20
Feed pressure, kPa	551.6 kPa (80 psi)	551.6 kPa (80 psi)
Cake thickness, mm	60 (Full Cake)	60 (Full Cake)
Dry bulk density, kg.m^3	1806.96	2072.81
Wet bulk density	2052.71	2359.25
Moisture content, %	**13.2**	**12.1**
Sizing basis, m^3/Mt	0.692 (Note #1)	0.603 (Note #1)
Filtration rate based on Total cycle time, kg/m^2.hr	130.0	139.3
Filtration rate based on 30mm Cake + Dry time, kg/m^2.hr	385.6	376.3
Cake form time, min	3.75	4.93
Wash time, min.	0	0
Air blow/squeeze time, min.	3.0	3.0
Miscellaneous time, min	10.0 (Note#2)	10.0 (Note #2)
Total cycle time, min.	16.75	17.93
Number of filters	1.0	1.0
No. of cycles per 20-hr day	71.6	66.9
Metric tons/day/filter	1192.0	1192.0
Metric tons/cycle required	16.64	17.81
M^3/cycle required	11.51	10.74
No. of recess plate chambers	63	59

Notes:
#1 Sizing basis in m^3 of pressure filter volume per metric ton (Mt) of dry solids, including 1.25 scale-up factor.
#2 Miscellaneous time of 10 minutes for cake discharge and cloth washing.

In 2012, HRI performed vacuum filtration and Kynch settling experiments on slurry obtained by water leaching calcines produced at 850°C and 30min residence time.

The vacuum filtration experiments were conducted to test the dewatering aids and to estimate the best operating conditions for the filtration operations. Traditional laboratory testing, using a filter leaf or Buchner funnel, permits the multi-step process of the horizontal vacuum filter to be demonstrated in the lab as well as in the selection of production scale equipment. Leaf testing indicates the filtration rate and is used to check the feasibility of a type of filter. Proven filter leaf scale-up factors allow the leaf test results to be used to "size" vacuum filters.

- Three feed slurry loadings at an approximate pulp density of 50% were examined with the vacuum filtration;
- The cake form rate averaged 808kg/m^2/hr. over the three runs;
- The residual moisture content of the filter cake was 30%.

The vacuum filtration test results are summarized in Table 13.41.

Table 13.41 EIMCO Vacuum Filter Leaf Test Results*

Filter Leaf Area, m^2	0.0045	Test ID		
Slurry Temperature	Ambient	3136-136-1	3136-136-1	3136-136-1
Feed slurry solids, %		47	50	50
Feed slurry mass, g		183	234	275
Wet filter cake mass, g		125	164	191
Dry filter cake mass, g		86.2	118.1	136.8
Diatomaceous earth, g		5.6	6.6	7.6
Flocculant, mg		0	0	0
Filter cake solids, %		69	72	71
Vacuum, mm Hg		520	520	520
Cake form time, min.		1.35	1.92	2.42
Cake thickness, mm		10	15	20
Cake form rate, (kg/m^2)/hr		851	821	755

*Mellon, Robert J. May 21, 2012

Settling or sedimentation of a suspension is generally evaluated by a “jar test” during which a suspension is allowed to settle and the height of the clear liquid (supernatant) suspension interface is measured as a function of the settling time. Thickener design involves the application of one of a number of alternative models together with jar test data. The Kynch theory of sedimentation is based on the assumption that at any point in the suspension the settling velocity of a particle depends only on the local concentration of the suspension and that the particles are of the same size and shape.

A plot of settling rate versus concentration can be constructed from a single settling curve. Kynch developed methods for determining concentrations and fluxes of solids in the transition zone between the constant rate period and the final underflow concentration from a single batch settling test. This would allow the prediction of the unit area (ft^2/ton solids-day) for specific thickening applications. HRI used in the Kynch experiments two doses of Hychem AF-303-HH flocculant and Table 13.42 summarizes the settling test results.

Table 13.42 Kynch Settling Test Results*

Parameter	Units	Experiment 1	Experiment 2
Flocculant dosage	mg/kg	12	30
Initial settling Rate	ft/hr	242	390
Settled solids	%	48.4	50.6
Unit area	ft^2/(st/d)	0.16	0.17

*Mellon, Robert J. May 21, 2012.

The Kynch settling tests indicated:

- The flocculant cleared the supernatant and produced an underflow pulp density of 48 to 50wt% solids;
- The settling area for solids was estimated at 0.16 to 0.17ft^2/(st/day);
- Of the two flocculant dosages, the lowest dosage of 12mg/kg of solids proved sufficient in clearing the supernatant of turbidity.

13.18 CRYSTALLIZATION TEST WORK

HRI observed in the water-leach tests (Mellon, Robert J. May 21, 2012) that the concentration of potassium in the leach liquor was 12.8g/L at five minutes and 12.4g/L at the end of a 60 minute leach cycle, whereas the respective aluminum concentration declined from 0.5g/L to 0.1g/L as the leach progressed.

This suggests that minimal contact time with the water may extract the potassium from the calcines. Longer contact time with water may allow precipitation and removal of aluminum from the leachate. This would reduce the amount of aluminum in the feed to the downstream K_2SO_4 crystallizer, thus enhancing the quality of the SOP product;

HRI performed one crystallization experiment with the goal to generate crystals in several stages so that the co-precipitation of impurities can be evaluated.

- The crystallization experiment recovered 43% of the potassium in the feed solution by reducing the liquor volume 91% by evaporation and the crystals assayed 25wt%K with small amounts of other contaminants;
- The brine was generated by leaching at 90°C for one hour the calcine produced at 900°C and a residence time of 30 minutes and filtering the slurry using Whatman 541 filter paper;
- The filtrate used as feed to the crystallizer contained 11.7g/L K and 0.081g/L Al;
- The crystallizer was a “resin kettle” with an overhead agitator placed in a heating mantle and under vacuum for maintaining boiling at 70 to 75°C;
- Designated Cycle 1: about 20mg of crystals were formed after reducing the feed volume to the crystallizer from 691mL to 305mL or 55%;

- The crystals from Cycle 1 contained both potassium and aluminum at 8.0 and 13.5wt%, respectively;
- About 160mg of crystals assaying 24wt% K and less than 0.05wt% Al and indicating K_2SO_4 crystallization were precipitated between the first and second crystallization cycles by cooling the primary filtrate to room temperature;
- The slurry from the previous step was filtered and the filtrate was further heated for one hour to produce an additional crystal crop;
- The Cycle 1 filtrate as feed for the Cycle 2 experiments was evaporated from 327mL to 69mL, or an additional 79%. After washing, 200mg of crystals were collected, which assayed 25wt% K, less than 0.05% Al, and less than 0.001wt% Ti, 0.021wt% Na, 0.025wt% Fe, and less than 0.0025wt% Mg.

Leaching the calcines at higher solids content and/or recycling the leach solution to buildup the concentration of potassium should be investigated to reduce the large (91%) evaporation requirement and to improve the potassium recovery without co-crystallizing contaminants.

In the commercial-scale evaporation/crystallization operation, a limited amount of K_2SO_4 will be recovered until other salts begin to crystallize, contaminating the SOP product. A bleed from the crystallizer will be necessary to prevent contaminating the K_2SO_4 product.

Depending on the size of this bleed, which may contain considerable potassium values, an option such as membrane separation (reverse osmosis) process for recovering potassium from the bleed stream should be evaluated.

PRC contracted the services of consultants with specialized knowledge in the applied fields of thermodynamics, heat transfer, evaporators and crystallizers. Because crystallization is an energy-intensive process, the best design requires consideration of the process fluids, contaminants, maintenance and the relative cost of electricity and steam.

Pending pilot plant test results on the brine produced by leaching the calcine, tentative recommendations on the evaporator/crystallizer circuit are as follows:

- The quadruple effect crystallizer design, which uses evaporated water from the process stream to drive a second heat exchanger at a lower pressure is recommended.
- Mechanical Vapor Recompression (MVR) systems utilize mechanical fans and use less steam than the multi-effect design. However, they require large heat exchangers with cooling water. After start-up of an MVR system, no additional steam is required for operation.

- Impurity build-up should be controlled in the recirculated centrate by establishing a purge or "bleed" stream from the centrifuge dewatering circuit. Bench-scale investigations will be required to determine the amount of bleed and to identify a method of treatment to recover the contained values. After treatment, the barren solution essentially containing sodium chloride can be routed to an evaporation pond.

13.19 FLOTATION OF ALUNITE

The alunite ore evaluated by HRI in 2012 contained up to 45wt% quartz. Physical beneficiation, such as flotation, to remove the quartz fraction was not performed during the 2012 laboratory investigations.

In the 1970s, HRI investigated flotation as a method of concentrating of alunite and targeted an 80% recovery at 80% grade of alunite. In February 2013, objectives of a flotation test campaign were outlined:

- Target of 80% recovery at 80% grade;
- Depress the silica to tails;
- Float the alunite mineral;
- Use sulfur as the determinant for alunite concentration.

Flotation was identified as a candidate technology for the recovery of alunite concentrate with low levels of quartz as an alternative to whole ore processing because if this approach is technically viable, flotation has the potential to reduce the capital and operating costs, resulting in energy savings in the calcining step and reducing the equipment size in calcining and leaching operations. By mid-2013, HRI completed 71 scoping alunite flotation experiments to evaluate the process parameters.

- P_{80} of 80µm was chosen as the lower limit for fluid bed calcining/roasting operations;
- The dosage of collector was varied from 0.1 to 4.0lb/t of solids;
- Soda ash (Na_2CO_3) and potassium hydroxide (KOH) were the pH modifiers chosen to achieve the target value of pH 10.0. KOH was used to eliminate the build-up, if any, of sodium in the feed to the calciner;
- The dosage of frother W22C was approximately 0.026lb/t;
- Sodium silicate (Na_2SiO_3) at a dosage of 1.8lb/t was investigated to depress silica;
- Flotation experiments were conducted at ambient temperatures;
- The flotation experiments indicated approximately 78% recovery at 85% alunite grade was possible using oleic acid and vapor oil (a by-product of petroleum refining process) as collectors in open-circuit flotation;

- Though flotation proved to be a technical feasibility, the dosage and cost of reagents proved prohibitive. A shift was then made to whole ore processing.

Processing whole ore is the chosen technology over alunite flotation concentrate for the following reasons:

- Coarser particle-size in the 1000µm range is well suited for energy-efficient gas suspension calciner or fluid bed roaster;
- Relatively fine-ground flotation concentrate is not suitable for fluidized bed reactors;
- Moisture content of flotation concentrate filter cake is relatively high compared with coarse-ground whole ore filter cake;
- Water and energy conservation are of paramount importance at the project site.

13.19.1 Historical Flotation Test Results

In 1976, Alumet investigated the flotation of alunite from the ore and silica from a water leach residue (R. Myertons, May 27, 1976). The resulting laboratory investigations from this report are summarized below. The objective of this investigation was to produce a concentrate containing less than 5wt% silica at greater than 50wt% recovery of alumina or alunite.

- The test work on alunite ore produced concentrates containing about 8wt% silica, at 30 to 40% to alunite recovery;
- At 50% alunite recovery, under open-circuit conditions, silica content of the concentrate was about 13wt% SiO_2;
- Examination of leach residue indicated that the silica was present as porous cryptocrystalline aggregates with alunite between the quartz crystals;
- Preliminary tests were conducted with amine as the reagent for floating silica from water leach residue and alunite concentrate;
- Amine reagents were found to be more effective as collectors for alumina than silica.

The PZC for corundum or alumina (Al_2O_3) is in the pH 9.0-9.3 range and amines should not float alumina below that pH. Based on this principle, preliminary tests were conducted to develop a reverse flotation system to separate silica from alunite. These tests indicate that amines tend to float alunite in preference to quartz.

13.20 SULFURIC ACID PLANT

An estimated 2090tpd of sulfuric acid (H_2SO_4) is manufactured at the project site from sulfur dioxide (SO_2) produced by the decomposition of alunite during the drying/calcining/roasting of ROM ore. FLSmidth estimates the concentration of SO_2 and SO_3 at 9.44vol. % in the off gases as feed to the acid plant from the proposed Dryer/Calciner/Roaster systems, with a throughput capacity of 425tph of crushed ore at 2% moisture content.

Based on information developed by FLSmidth by desktop simulations for the drying, calcining and roasting of whole ore, Du Pont-MECS, a supplier of the sulfuric acid plant, estimates the composition of off-gases as feedstock to acid plant.

It is proposed that one (1) sulfuric acid plant, with a throughput capacity of 2090tpd of concentrated acid (100% H_2SO_4) be constructed, to process the off-gases from the roaster.

To improve yields and to minimize atmospheric pollution, modern day plants incorporate what is known as “interpass absorption” design, also known as” double catalysis” in which a second absorbing tower removes SO_3 formed from SO_2 before the last stage of the converter is inserted. With this design, it is possible for the yield to be 99.7% to 99.9% and for the SO_2 in the exit gas to be 100 to 350ppm. Without the inter-pass, the exit gas may contain up to 2,000ppm sulfur dioxide. Tail gas scrubbing is required to further reduce the concentration of SO_2 to less than 50ppm in stack discharges to the atmosphere.

14 MINERAL RESOURCE ESTIMATES

Resources estimates for Blawn Mountain were last determined and presented in the 2013 PFS. There have been no changes to the resource estimates since the 2013 PFS.

Four potential mine development targets have been identified within the Blawn Mountain property. Only Area 1 and Area 2 have sufficient geologic and analytical data to support resource estimation at this time. Areas 3 and 4 are defined by a limited number of historical holes and surface mapping along with only two validation holes in Area 4. Both areas are recognized as future exploration targets.

Resources have been estimated from 3DGBM's constructed in MineSight®, a software package developed by Hexagon Mining Inc. The estimate was prepared in compliance with NI 43-101 requirements for the definition of mineral resources. The 3DGBM's are based on the assays and lithologies of the current drilling database and on a series of 30 interpreted geological cross sections constructed through Area 1 and 29 cross sections constructed through Area 2.

A total of 142 exploration drill holes have been completed by PRC on the property as of the effective date of November 6, 2013. These holes include 75 twin and infill holes in Area 1 and 67 infill holes in Area 2. As discussed in Section 12, there was poor correlation observed with the twin drilling program conducted by PRC. A decision was made by PRC in 2013 to no longer use the historical data and that more reliable estimates would be achieved using only the recent (2011-2013) PRC drilling data. Holes not included in the geologic model include all pre-2011 historical holes completed by ESI. There are insufficient records for these air track holes to be used in the geologic model. Remaining historic holes were excluded from the geologic model due to lack of sufficient documentation relating to assay testing standards.

A number of criteria was established for determination of resources.

A statistical review of analytical results through the construction of a series of correlograms determined that there was no appreciable preferred orientation of grades for K_2O and Al_2O_3. Down-hole variograms were also prepared and showed that there was no significant nugget effects or directionality to the data that would require more robust kriging approaches. Correlograms and down-hole variograms of K_2O and Al_2O_3 for Area 1 and Area 2 are outlined in Figure 14.1 and Figure 14.2 respectively.

Analytical results were based on composites developed over 10ft intervals in each hole. Four lithologic domains are represented in the geologic block model: Alunite, Clay, Dolomite and Silica.

The geologic block model for Area 1 has the overall dimensions of 5,900ft west to east, 3,900ft north to south and a 1,400ft elevation range. The geologic block model for Area 2 has the overall dimensions of 8,800ft west to east, 11,960ft north to south and a 1,700ft elevation range.

The block model minimum and maximum dimensions are outlined in Table 14.1 for Area 1 and Table 14.2 for Area 2. All units outlined in Table 14.1 and Table 14.2 are in Utah State Plane – south coordinates, NAD27. Unless otherwise specified, units are reported in US customary units (feet/short tons).

Table 14.1 Area 1 Block Model Dimensions

	Minimum	Maximum	Block Size
Easting	1,418,100	1,424,000	20
Northing	591,300	595,200	20
Elevation	6,300	7,700	20

Table 14.2 Area 2 Block Model Dimensions

	Minimum	Maximum	Block Size
Easting	1,423,200	1,432,000	20
Northing	583,540	595,500	20
Elevation	6,000	7,700	20

A standard cubic block size of 20ft (X-dimension) by 20ft (Y-dimension) by 20ft (Z-dimension) was used in both the Area 1 and Area 2 block models. First pass data search radii for K_2O estimation were 350ft and Al_2O_3 were 250ft for both models. Second pass data search radii for K_2O and Al_2O_3 were 2,000ft for both models. The larger search radii for the Area 2 model was used to account for the more widely spaced drilling.

Topographic data for both Area 1 and Area 2 is derived from a detailed aerial (5ft resolution) mapping survey completed by Olympus Aerial Surveys in August, 2013. The survey covers an area of 16,300 acres. Peripheral regions within the greater project area are infilled with digital elevation model (DEM) data with a 5m (16.4ft) resolution sourced from the Utah Automated Geographic Reference Center (AGRC.)

Resource classification is based on set distances from drill hole sample intervals in 3D space. These distances were based on semi-variogram analysis of K2O sample data. Table 14.3 outlines the distance from drilling samples used to classify resource blocks within the alunite zone.

Table 14.3 Classification Criteria

Measured	Indicated	Inferred
<150ft	<350ft	<2,000ft

The assumed density of alunite and waste was established at 153.8 pounds per cubic foot (lb/ft^3) or specific gravity of 2.46 grams per cubic centimeter (g/cc), as derived from estimates used previously by ESI (1974). This bulk density factor is considered reasonable for this deposit type.

The boundaries of the deposit were defined by the applied radii of influence of drill holes or interpreted structural controls such as known bounding fault systems and alteration limits. These limits have been updated to reflect the relationship between SO_4 grade data and alunite mineralization.

As discussed in Section 12, the recent drill hole data has indicated the presence of high K_2O grades in feldspar-rich rhyolitic country rock and that there is an association between SO_4 and alunite mineralization. To better define the boundary between country rock and alunite mineralized zones, drill hole sample intervals with greater than 0.8% SO_4 were used to separate alunite mineralization from surrounding country rock. Both visual and calculated validation of model block values to posted drill assay values show strong correlation. A series of swath plots of comparing average K_2O and Al_2O_3 grades from the 3DGBM with drill hole grades are illustrated in Figure 14.3 and Figure 14.4 respectively.

Resource classification is based on the CIM Standards on Mineral Resources and Reserves, a set of definitions and guidelines established by the Canadian Institute of Mining and Metallurgy and Petroleum. Table 14.4 shows the estimated classified resource for the Area 1 Blawn Mountain Alunite deposit at increasing incremental K_2O cut-off grades.

Figures 14.5, 14.6, and 14.7 show cross sections through the block model for Area 1. The cross sections exhibit typical zoned mineralization for hydrothermal alteration also referred to as “nested cone geometry” by Krahulec. Figure 14.8 identifies the classified resource areas for the Area 1 Blawn Mountain property.

The preferred scenario for resource presentation is a 1% K_2O cut-off grade. At a 1% cut-off grade, there is a combined measured plus indicated resource of 164.8Mt of material carrying an average grade of 3.35% K_2O and 15.41% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade is 6.20%. This cut-off grade maximizes the in-place tons while providing a quantity of K_2SO_4 deemed suitable by current processing studies.

Increasing the cut-off grade to 3% K_2O reduces the combined tons of material to 106.3Mt. Average grade at a 3% K_2O cut-off is 3.94% K_2O and 16.50% Al_2O_3 with a calculated equivalent grade of 7.29% K_2SO_4. Approximately 43% of the identified resources are classified as measured, 56% as indicated resource and 1% as inferred resource.

Table 14.4 Classified Resource Estimate for the Area 1 Blawn Mountain Alunite Deposit

			IN SITU GRADES						CONTAINED RESOURCES				
RESOURCE CLASSIFICATION	CUTOFF GRADE K_2O (%)	IN SITU (TONS)	K_2O (%)	K_2SO_4 (%)	AL_2O_3 (%)	SO_4 (%)	Alunite based on K_2O (%)	Alunite based on Al_2O_3 (%)	K_2O (TONS)	K_2SO_4 (TONS)	Al_2O_3 (TONS)	Alunite based on K_2O (TONS)	Alunite based on Al_2O_3 (TONS)
MEASURED	0.00	72,400,282	3.42	6.32	15.68	2.09	30.04	42.45	2,473,700	4,574,597	11,353,088	21,746,273	30,734,994
	1.00	71,529,372	3.45	6.39	15.71	2.06	30.36	42.52	2,469,909	4,567,586	11,235,691	21,712,945	30,417,176
	2.00	64,979,040	3.64	6.73	16.19	2.09	32.01	43.83	2,366,212	4,375,819	10,520,042	20,801,341	28,479,776
	2.50	56,872,179	3.84	7.10	16.33	2.13	33.75	44.21	2,183,323	4,037,604	9,286,829	19,193,568	25,141,232
	3.00	48,362,178	4.03	7.44	16.62	2.19	35.39	45.01	1,946,916	3,600,418	8,040,067	17,115,319	21,766,008
	3.50	34,526,334	4.33	8.00	17.31	2.31	38.03	46.87	1,493,713	2,762,313	5,976,957	13,131,213	16,180,773
	4.00	19,624,648	4.78	8.84	18.71	2.55	42.01	50.64	937,725	1,734,127	3,671,046	8,243,526	9,938,226
INDICATED	0.00	93,823,555	3.26	6.03	15.19	2.03	28.65	41.12	3,057,710	5,654,601	14,250,203	26,880,292	38,578,041
	1.00	93,313,743	3.27	6.05	15.19	2.02	28.78	41.12	3,054,532	5,648,725	14,172,865	26,852,358	38,368,671
	2.00	83,493,484	3.48	6.43	15.88	2.05	30.57	42.98	2,902,985	5,368,470	13,255,676	25,520,109	35,885,665
	2.50	74,184,688	3.63	6.71	16.06	2.09	31.91	43.47	2,692,385	4,979,008	11,911,168	23,668,726	32,245,822
	3.00	57,939,557	3.87	7.15	16.40	2.14	34.01	44.40	2,241,624	4,145,419	9,503,304	19,706,088	25,727,272
	3.50	36,959,714	4.21	7.78	17.30	2.29	36.97	46.84	1,554,489	2,874,705	6,395,250	13,665,492	17,313,172
	4.00	17,565,100	4.73	8.75	19.11	2.57	41.61	51.73	831,391	1,537,486	3,356,462	7,308,752	9,086,589
MEASURED AND INDICATED	0.00	166,223,837	3.33	6.15	15.40	2.06	29.25	41.70	5,531,410	10,229,198	25,603,291	48,626,565	69,313,034
	1.00	164,843,115	3.35	6.20	15.41	2.04	29.46	41.73	5,524,441	10,216,310	25,408,555	48,565,303	68,785,847
	2.00	148,472,524	3.55	6.56	16.01	2.07	31.20	43.35	5,269,197	9,744,288	23,775,718	46,321,450	64,365,441
	2.50	131,056,867	3.72	6.88	16.17	2.11	32.71	43.79	4,875,708	9,016,612	21,197,996	42,862,294	57,387,054
	3.00	106,301,735	3.94	7.29	16.50	2.17	34.64	44.68	4,188,540	7,745,837	17,543,371	36,821,407	47,493,280
	3.50	71,486,048	4.26	7.89	17.31	2.30	37.49	46.85	3,048,201	5,637,017	12,372,207	26,796,705	33,493,946
	4.00	37,189,748	4.76	8.80	18.90	2.56	41.82	51.16	1,769,116	3,271,614	7,027,508	15,552,278	19,024,815
INFERRED	0.00	2,255,374	3.18	5.87	14.62	2.04	27.92	39.57	71,626	132,458	329,697	629,665	892,554
	1.00	2,255,374	3.18	5.87	14.62	2.04	27.92	39.57	71,626	132,458	329,697	629,665	892,554
	2.00	1,919,126	3.48	6.44	15.87	2.10	30.60	42.95	66,797	123,527	304,485	587,213	824,299
	2.50	1,793,895	3.56	6.59	15.98	2.11	31.33	43.26	63,938	118,240	286,657	562,078	776,036
	3.00	1,429,416	3.77	6.97	16.36	2.11	33.15	44.29	53,899	99,675	233,868	473,825	633,126
	3.50	665,917	4.37	8.08	18.68	2.44	38.41	50.56	29,097	53,809	124,375	255,794	336,706
	4.00	407,414	4.78	8.84	20.45	2.68	42.05	55.35	19,486	36,036	83,298	171,303	225,504

Effective Date: November 6, 2013

Table 14.5 shows the estimated classified resource for the Area 2 Blawn Mountain Alunite deposit at increasing incremental K_2O cut-off grades. Figures 14.9, 14.10, and 14.11 show cross sections through the block model for Area 2. Figure 14.12 identifies the classified resource areas for the Area 2 Blawn Mountain Property.

At a 1% cut-off grade, there is a combined measured plus indicated resource of 398.4Mt of material carrying an average grade of 2.97% K_2O and 12.99% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade is 5.50%. This cut-off grade maximizes the in-place tons while providing a quantity of K_2SO_4 deemed suitable by current processing studies.

Increasing the cut-off grade to 3% K_2O reduces the combined tons of material to 180.5Mt. Average grade at a 3% K_2O cut-off is 3.65% K_2O and 14.25% Al_2O_3 with a calculated equivalent grade of 6.76% K_2SO_4. Approximately 20% of the identified resources are classified as measured, 55% as indicated resource and 25% as inferred resource.

Table 14.5 Classified Resource Estimate for the Area 2 Blawn Mountain Alunite Deposit

			IN SITU GRADES						CONTAINED RESOURCES				
RESOURCE CLASSIFICATION	CUTOFF GRADE K_2O (%)	IN SITU (TONS)	K_2O (%)	K_2SO_4 (%)	AL_2O_3 (%)	SO_4 (%)	Alunite based on K_2O	Alunite based on Al_2O_3	K_2O (TONS)	K_2SO_4 (TONS)	Al_2O_3 (TONS)	Alunite based on K_2O (TONS)	Alunite based on Al_2O_3 (TONS)
	0.00	110,497,331	2.87	5.31	12.42	1.94	25.23	33.62	3,170,721	5,863,592	13,720,785	27,873,773	37,144,805
	1.00	104,377,825	3.02	5.58	13.05	2.04	26.54	35.33	3,150,958	5,827,044	13,622,767	27,700,035	36,879,452
	2.00	93,679,360	3.18	5.87	13.38	2.11	27.91	36.22	2,974,320	5,500,388	12,532,237	26,147,212	33,927,177
MEASURED	2.50	79,064,980	3.34	6.17	13.60	2.20	29.33	36.81	2,638,240	4,878,879	10,750,703	23,192,741	29,104,220
	3.00	50,041,863	3.68	6.80	14.26	2.43	32.34	38.60	1,841,190	3,404,900	7,134,769	16,185,883	19,315,191
	3.50	28,969,753	3.98	7.36	14.75	2.62	34.97	39.93	1,152,272	2,130,888	4,272,459	10,129,610	11,566,369
	4.00	9,150,291	4.47	8.26	15.62	2.72	39.27	42.29	408,771	755,937	1,429,550	3,593,501	3,870,067
	0.00	307,822,418	2.84	5.25	12.47	1.90	24.96	33.77	8,739,386	16,161,686	38,398,692	76,827,849	103,952,645
	1.00	293,961,004	2.96	5.47	12.97	1.97	25.98	35.12	8,688,605	16,067,777	38,135,561	76,381,434	103,240,300
	2.00	263,614,932	3.10	5.73	13.34	2.05	27.25	36.12	8,171,536	15,111,563	35,174,931	71,835,880	95,225,305
INDICATED	2.50	212,810,329	3.30	6.10	13.64	2.15	29.00	36.93	7,020,826	12,983,563	29,031,159	61,720,000	78,592,933
	3.00	130,484,506	3.64	6.74	14.25	2.37	32.04	38.57	4,755,377	8,794,086	18,592,476	41,804,470	50,333,410
	3.50	71,126,489	3.96	7.32	14.70	2.57	34.81	39.79	2,816,324	5,208,209	10,454,527	24,758,277	28,302,416
	4.00	20,689,481	4.48	8.28	15.52	2.62	39.37	42.03	926,537	1,713,438	3,211,794	8,145,177	8,694,943
	0.00	418,319,749	2.85	5.27	12.46	1.91	25.03	33.73	11,910,107	22,025,277	52,119,477	104,701,621	141,097,450
	1.00	398,338,829	2.97	5.50	12.99	1.99	26.13	35.18	11,839,563	21,894,821	51,758,329	104,081,470	140,119,752
	2.00	357,294,292	3.12	5.77	13.35	2.07	27.42	36.15	11,145,855	20,611,952	47,707,169	97,983,092	129,152,483
MEASURED AND INDICATED	2.50	291,875,309	3.31	6.12	13.63	2.17	29.09	36.90	9,659,066	17,862,442	39,781,862	84,912,741	107,697,153
	3.00	180,526,369	3.65	6.76	14.25	2.39	32.12	38.58	6,596,568	12,198,986	25,727,245	57,990,353	69,648,601
	3.50	100,096,242	3.96	7.33	14.71	2.59	34.85	39.83	3,968,596	7,339,097	14,726,986	34,887,887	39,868,784
	4.00	29,839,772	4.47	8.28	15.55	2.65	39.34	42.11	1,335,308	2,469,376	4,641,344	11,738,678	12,565,010
	0.00	150,481,703	2.64	4.88	12.08	1.67	23.18	32.70	3,967,300	7,336,699	18,175,180	34,876,487	49,203,709
	1.00	134,770,366	2.90	5.37	13.25	1.84	25.51	35.88	3,911,306	7,233,150	17,862,464	34,384,244	48,357,127
	2.00	124,717,186	2.99	5.54	13.46	1.88	26.31	36.44	3,733,035	6,903,475	16,787,557	32,817,068	45,447,146
INFERRED	2.50	94,690,184	3.23	5.97	13.70	1.97	28.37	37.10	3,055,842	5,651,146	12,974,828	26,863,870	35,125,355
	3.00	55,899,862	3.56	6.59	14.25	2.15	31.31	38.58	1,990,874	3,681,709	7,967,240	17,501,748	21,568,850
	3.50	21,879,368	4.02	7.44	14.72	2.51	35.38	39.85	880,623	1,628,529	3,220,577	7,741,544	8,718,722
	4.00	9,143,043	4.45	8.23	15.46	2.63	39.13	41.86	406,939	752,549	1,413,798	3,577,392	3,827,423

Effective Date: November 6, 2013

The accuracy of resource and reserve estimates is, in part, a function of the quality and quantity of available data and of engineering and geological interpretation and judgment. Given the data available at the time this report was prepared, the estimates presented herein are considered reasonable. However, they should be accepted with the understanding that additional data and analysis available subsequent to the date of the estimates may necessitate revision. These revisions may be material. There is no guarantee that all or any part of the estimated resources or reserves will be recoverable.

15 MINERAL RESERVE ESTIMATES

15.1 APPROACH

Millcreek has used the instrument document, the Canadian Institute of Mining, Metallurgy and Petroleum's CIM Standards on Mineral Resources and Reserves prepared by the CIM Standing Committee on Reserve Definitions, 2014, as the basis for the classification, estimation and reporting of potash resources and reserves for the Blawn Mountain property.

Based on the 3DGBM, resource areas were developed for the deposit. A mine-plan was generated as the basis of the reserve estimate as reported in this document. The mine-plan was established by first applying various criteria in selecting the method and approach to mining including an annual run-of-mine (ROM) ore production rate to be constrained by processing capabilities and maximizing economic use of the resource.

The mine-plan uses standard surface mining equipment employing truck and shovel methods to provide ore to the processing facility (see Section 16). The production level for the mine is approximately 3.4Mtpy. Active mining occurs in Area 1 and Area 2 for approximately 28 years. During this time, ore meeting nominal cut-off criteria of 3.75% K_2O for Area 1 and 3.50% K_2O for Area 2 is sent directly to the processing facility. Lower grade ores, down to 2.75% K_2O, are placed in temporary low-grade ore stockpiles. Once active mining ceases in Area 1 and Area 2, the low-grade ore stockpiles are reclaimed and sent to the processing facility. Low-grade ore stockpile reclamation occurs for roughly an additional 18 years.

The mine-plan was developed and scheduled using MineSight® software to generate a Life-of-Mine (LOM) schedule of waste and ore volumes. Equipment productivities were then applied to these volumes which allowed for fleet sizes and their associated equipment hours to be estimated. These estimates formed the basis of workforce demands and production schedules leading ultimately to estimates of capital and operating costs (see Section 21). Taking into account commodity pricing and market conditions, ore processing capital and operating costs and contract mining costs, cash flow of revenues and direct and indirect costs were developed. Ultimately, these efforts led to an estimate of project economics and valuation (see Section 22).

15.2 RESERVES ESTIMATE

The mine-plan proposed in this study, at a prefeasibility level of assurance, has been found to be of positive economic value and forms the basis of mineral reserves reported here.

Mineral reserves, by category, are summarized in Table 15.1.

Table 15.1 Mineral Reserves, by Category

Alunite Ore (ROM tons)	Grade (% K_2O)	Grade (%K2SO4)	Reserve Category Proven ('000 tons)	Reserve Category Probable ('000 tons)	Total
Direct Feed to Mill	4.32	7.99	36,882	53,917	90,799
Medium Grade Stockpile	3.49	6.47	14,916	23,158	38,074
Low-grade Stockpile	2.99	5.52	7,983	16,473	24,457
Total	**3.90**	**7.22**	**59,782**	**93,548**	**153,330**
Products					
SOP tons			**4,135**	**6,468**	**10,603**
Sulfuric Acid (tons) @ 93% Purity			**9,410**	**14,725**	**24,135**

This estimate of resources and reserves was generated using the best information available concerning issues related to environmental, permitting, legal, title, taxation, socio-economics, marketing and political factors that could have a material influence on Millcreek's findings. Millcreek is not aware of any additional factors which may affect our reserves estimate. The accuracy of resource and reserve estimates is, in part, a function of the quality and quantity of available data and of engineering and geological interpretation and judgment. Given the data available at the time this report was prepared, the estimates presented herein are considered reasonable. However, they should be accepted with the understanding that additional data and analysis available subsequent to the date of the estimates may necessitate revision. These revisions may be material. There is no guarantee that all or any part of the estimated resources or reserves will be recoverable.

15.3 CHANGE IN RESERVES ESTIMATES

The previous PFS, reported in 2013, reported mineable, economic reserves of approximately 426.0Mt. This is in contrast to currently estimated reserves of 153.3Mt. The variance in these estimates is **not** attributed to a decrease in mineable resource, or newly-identified adversities in mining conditions, operating costs or other economic factors (such as price). Rather, the current reserves estimate reflects an improved and fundamentally different approach to the project with respect to annual production rate.

15.3.1 Annual Production Rate

Due to the general challenges in commodities markets in recent years, mining projects have faced difficulties in securing financing with the result that many greenfield projects have been forced to scale back their proposed capital scope. In order to present a more realistic and accurate estimate of project value, given the market's attitude to capital investment, PRC proposes an optimum annual production rate of 3.4Mpty (down from the previously proposed 10.6Mpty).

15.3.2 Life-of-Mine Constraints

In general, Life-of-Mine projections for economic reserves are typically in the 20 to 30-year time frame, and rarely exceed approximately 46 years. The upper end of this estimate is constrained by the impact of the time value of money and decreasing confidence in items such as product pricing, market position, major sustaining capital (for example, to re-build plant or facilities) as well as a decrease in the understanding of project risk. Beyond 46 years it is generally thought that project economics and risks cannot be determined to the 'pre-feasibility' level of confidence required to state economic reserves.

15.3.3 Conclusion

Given the above, the current study proposes reducing the annual production rate while maintaining a realistic LOM. This generates an improved and more relevant forecast of economic reserves but one that is on a different basis and significantly smaller than previously proposed.

The estimate of resources available for development has not changed since the previous PFS. While economic reserves have not been estimated beyond 46 years in this study, those same abundant measured and indicated resources could support mining and processing and overall project life well beyond the proposed 46 year LOM.

16 MINING METHODS

Mining operations at Blawn Mountain will be conventional open-pit, using truck and shovel mining methods to extract ore and waste material from two mining areas, Area 1 and Area 2. The mining areas and subsequent mine-plans were developed using Lerch Grossman pit shells that optimized ore grades and their impacts to overall project economics. The mine optimization and planning work utilized the geologic model described in earlier sections of this report.

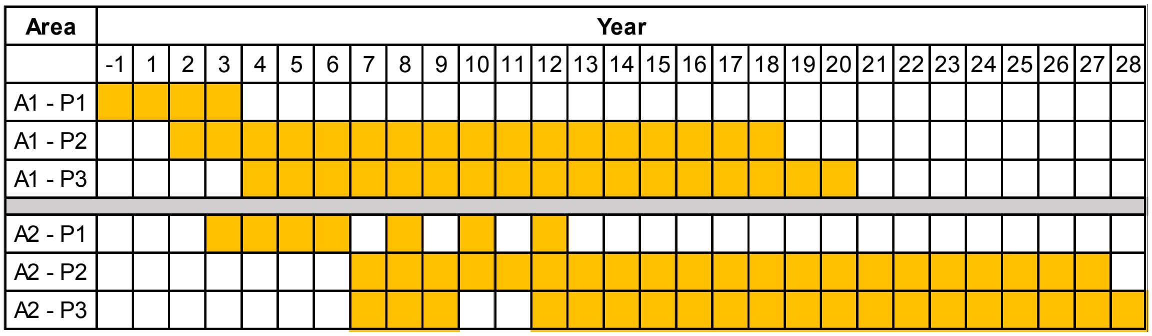
The production levels for the mine are commensurate with the capabilities of the processing facility. On a steady-state basis, the direct ROM feed to the processing facility is approximately 3.4Mtpy. Nominal ore cut-off grades of 3.75% K_2O and 3.50% K_2O were utilized for Areas 1 and 2, respectively. Ore meeting these criteria will be fed directly to the processing facility.

As lower grade ore is encountered during mining, it will be placed in temporary stockpiles that will be reclaimed and fed to the plant once active mining operations have ceased. Two low-grade ore stockpiles will be utilized, one for Area 1 and one for Area 2. The Area 1 stockpile will be constructed in such a way that ore ranging from 3.25% K_2O and 3.75% K_2O is separated from ore that ranges from 2.75% to 3.25% K_2O. Ore with grades of less than 2.75% K_2O is considered waste material and will be placed in waste dumps. This cut-off criteria was determined by analyzing ore grades and their effects to project economics. The low-grade ore stockpile for Area 2 will be handled similarly, with ore ranging from 3.25% K_2O to 3.5% K_2O being segregated from ore falling between 2.75% K_2O and 3.25% K_2O.

16.1 GENERAL MINING METHOD

Mining operations will use a conventional open pit, truck and shovel mining approach. This is a typical and standard approach for many surface mining applications and takes advantage of the flexibility of the mining equipment. For Blawn Mountain, Area 1 and Area 2 will be developed in phases that will allow for optimizing the ore grades encountered in the deposit, while providing flexibility to the operation. The mining phases for Areas 1 and 2 are illustrated below in Chart 16.1.

Chart 16.1 – Mining Phases for Area 1 and Area 2



With the exception of the first few years, mining will occur in both Area 1 and Area 2 simultaneously. Mining operations commence in Area 1 in Year 1. Year 1 is considered a construction period for stockpile area development, haul-road construction and initial mining area development. Year 1 is considered a "ramp-up" period and during this time, the mining area within Area 1 will be developed in such a way that full production can be achieved beginning in Year 2. The initial phase of mining in Area 2 is developed in Year 3 in coordination with operations in Area 1.

Ore and waste will be removed using bench mining techniques. Mining will begin at the top of the ridges in Area 1 and 2 and progress downward using multiple 20ft lifts/benches. Several mining areas and faces will be utilized throughout the mine's life to assist with mine scheduling efforts. Conventional truck/shovel mining techniques will be employed using two mid-sized front-end-loaders (FEL) to load end-dump mining trucks. FELs were chosen based on production requirements and the need to have a flexible equipment fleet. Prior to ore and waste removal, the material must be drilled and blasted.

16.2 AREA PREPARATION

Plant growth (PGM) material, also known as topsoil, encountered during mining will be salvaged and stored in temporary stockpiles. PGM stripping quantities associated with the mine plan were derived from a soil survey commissioned by PRC in 2013. PGM quantities are presented below in Table 16.1. The mining progression maps in Section 28 show the location of the temporary topsoil piles.

16.3 MINING OPERATIONS

As previously mentioned, the mining case for this PFS uses a nominal 3.75% K_2O ore grade cut-off for Area 1 and a 3.50% K_2O cut-off for Area 2. These cut-off grades were utilized in the development of the pit shells and mining areas for Area 1 and 2. The production schedule is driven by the capacity of the processing plant. The ROM ore production schedule for direct plant feed is approximately 3.4Mtpy.

Minimal waste material is encountered during operations. The average strip ratio (yd^3/ton ore) is 0.25:1. The majority of the waste material encountered in Areas 1 and 2 will be placed in out-of-pit waste dumps adjacent to the mining areas. Some of the waste material will be used to construct haul roads needed to access Areas 1 and 2. The waste dump locations as well as the progression of mining are shown in Figures 16.1 through 16.11 in Section 28.

The ROM production recognizes a leach recovery of 90%, 2% moisture. The SOP product was adjusted to account for 92.5% K_2SO_4. No mining recovery or dilution was assumed for this mine-plan. This assumption is based on the use of low-grade ore stockpiles and that a minimal amount of waste exists in the deposit by virtue of its natural formation. Low-grade ore will be stockpiled during active mining operations and this material will later be reclaimed and processed.

As previously discussed, two low-grade ore stockpiles will be developed, one for Area 1 and Area 2 respectively. Ore that is less than 2.75% K_2O is considered waste material and will be placed in the waste dumps. The Area 1 stockpile will contain ore ranging in grade from 2.75% K_2O to 3.75% K_2O and the ore in the Area 2 stockpile will range from 2.75% K_2O to 3.5% K_2O. As this material is placed in the stockpiles, it will be segregated in such a way that, as it is being reclaimed in later years of the project, the higher grades will be fed to the plant first.

Active mining begins in Year 1 and extends through Year 28. Mining operations cease in Year 28 but SOP will still be produced for an additional 18 years by reclaiming the low-grade ore stockpiles. The overall project life is approximately 46 years. Construction activities begin in Year 1 which includes construction of the stockpile pad area, Area 1 haul road and development of the initial mining phase in Area 1. Year 1 is a production ramp-up period and full production is realized in Year 2. The mining production schedule for Blawn Mountain is presented in further detail below as Table 16.1.

Table 16.1 Mining Production Schedule

Year	-1	1	2	3	4	5	6-10	11-15	16-20	21-25	26-30	31-46
PGM (kyd^3)	358	160	90	100	117	82	377	239	213	150	132	314
Waste (kyd^3)	45	103	700	873	1,237	943	4,258	7,081	11,924	7,888	861	
Waste (kt)	94	215	1,455	1,816	2,572	1,962	8,856	14,728	24,801	16,407	1,791	
Ore (kt)	218	1,684	3,365	3,405	3,367	3,362	16,830	16,809	16,813	16,849	8,098	
K_2O (%)	3.9	4.5	4.5	4.6	4.2	4.2	4.3	4.4	4.4	4.1	4.6	
Al_2O_3 (%)	15.2	16.7	17.3	17.0	15.7	15.6	16.3	17.2	16.9	14.9	16.4	
*LGO Stockpile (kt)	83	276	1,107	2,176	3,196	4,225	19,353	13,081	11,024	7,507	502	
*LGO Reclaim (kt)	0	0	0	0	0	0	0	0	0	0	8,729	53,802
K_2O (%)	3.5	3.5	3.4	3.4	3.4	3.4	3.4	3.2	3.1	3.3	3.5	3.3
Al_2O_3 (%)	14.5	14.0	14.7	14.4	14.4	14.4	14.6	14.0	12.2	12.6	14.3	13.7
Acid (kt)	34	288	600	596	545	541	2,814	2,969	2,928	2,588	2,657	7,574
SOP (kt)	15	132	268	277	251	247	1,271	1,309	1,303	1,226	1,198	3,106

*Low-grade Ore

The mining resources (equipment and manpower) needed to implement the mine-plan presented in this study will be provided by a contractor. The operating and capital costs associated with the mining operations takes this assumption into account and is presented and discussed in more detail in Section 21 of this report.

An equipment fleet has been selected to adequately execute the mine-plan and meet scheduling and production requirements. Two spreads of primary mining equipment are envisioned to remove ore and waste material from the mine. Table 16.2 presents the type, size and quantity of major mining equipment. The quantity of equipment presented in Table 16.2 is at approximate maximum levels. The quantity of end-dump trucks varies throughout the mine life as haul distances and mining areas change. Truck productivities were determined using Caterpillar's FPC, a haulage and loading simulator software. The productivity assumption used for the FEL is 1,000yd^3/hr. The density of the ore is 2.08t/yd^3.

Table 16.2 Mining Equipment

Primary Equipment		
FEL	16yd^3	2
End-Dump Truck	100t	12
Support Equipment		
Water Truck	12,000gal	1
Grader	297Hp	2
Dozer	580Hp	5
Drill	45,000lb	1

This equipment was selected as it provides the flexibility necessary to the support the multiple mining areas associated with the mine-plan. The mobility of this equipment fleet will allow for easier relocation to support the mining schedule and will assist with the various site conditions encountered during mining. The primary and support equipment will be supported by a fleet of smaller support equipment including pumps, light plants, lube and fuel trucks, mechanics trucks, pick-up trucks, etc.

The production requirements necessitate a 4 day per week schedule, operating 20 hours per day for approximately 208 days per year. The workforce needed to support the mining operation is presented below in Table 16.3. Table 16.3 shows the average personnel required in various increments throughout the entire mine life. It should be noted that after year 28, active mining is complete and low-grade ore stockpile reclamation will ensue.

Table 16.3 Average Workforce Requirements

Category	Y-1 –	Y6-Y10	Y11-Y20	Y21-28	Y29-Y46
Hourly Workers					
FEL Operators	2	3	4	3	3
Truck Drivers	8	13	19	21	6
Dozer Operators	6	8	8	6	2
Grader Operators	2	2	3	3	1
Drill Operators	2	3	2	2	0
Blasters	2	2	2	2	0
Mine Support	5	6	6	6	5
Maintenance Labor	16	24	21	18	6
Sub-Total Hourly	**43**	**61**	**65**	**61**	**23**
Management					
Exec, Staff, Tech	7	7	7	7	2
Maintenance	4	4	4	4	1
Operations	4	4	4	4	1
Total Employees	**58**	**76**	**80**	**76**	**27**

16.4 GEOTECHNICAL CONSIDERATIONS

An initial evaluation of slope stability for Blawn Mountain was performed in January 2013 by Seegmiller International of Salt Lake City, Utah (Seegmiller). This evaluation was based on data collected from the site as well as core from two holes drilled on the property. Seegmiller recommended mine slopes with overall angles of 45°. Millcreek utilized these constraints in the creation of the mining pit shells. Further geotechnical testing will be conducted prior to commencement of mining operations to provide additional data for mine design. The mine-plan assumes that waste material will be disposed of using external waste piles. The external waste piles are constructed using 2H:1V side slopes.

16.5 HYDROLOGICAL CONSIDERATIONS AND WATER MANAGEMENT

The processing plant and mine areas are located near the head of two ephemeral drainages, therefore, the runoff potential for these drainages is limited. The vast majority of surface water at the site will be controlled using a series of adequately sized berms, ditches and ponds that will collect water from the disturbance areas associated with the mine and processing plant areas. This water will be captured in sumps and ponds and allowed time for sediment to settle prior to discharge. The captured water is expected to be limited in volume, and will likely be used for watering mine haul roads, etc.

The surface water management plan is illustrated on Figure 16.12. The plan consists of five sediment ponds with collection ditches around the outside of mining Areas 1 and 2. The drainage from the area between Areas 1 and 2 and the processing plant facilities will be collected by internal ditches and conveyed to the collection pond below the tailing storage area, which is located down gradient of the mining areas and process facilities.

It is anticipated that limited groundwater will be encountered during mining and only near the maximum depths of the proposed mining pits. Groundwater was assessed using a series of groundwater monitoring wells installed in the mine area. These wells showed that limited groundwater is found in the volcanic tuffs and andesite flow zones and that flows in these zones range from 0 to 10gpm. It is anticipated that the groundwater encountered as the mine advances will not be significant and should not be a hindrance to mining. Any groundwater encountered during mining will be collected, as needed, in sumps in the pit floor. This water will then be pumped for in-pit use and dust control.

16.6 MINE RECLAMATION

Upon completion of mining, all mine facilities will be demolished and the disturbed area will be reclaimed. All utility connections and foundations will either be buried or removed. The mine areas will be left in stable configurations of no steeper than 45 degree slopes. All external waste piles will be regraded to a stable configuration and sloped to minimize safety hazards and erosion. The roads and facility pads will be regraded to transition into the adjacent terrain. Salvaged PGM will be spread over the disturbed areas per regulatory requirements and seeded. A seed mix, approved by (Utah Division of Oil, Gas and Mining) UDOGM and SITLA will be applied as necessary. Fertilization or other soil amendments will be applied if necessary. Revegetation efforts will be monitored to assure that revegetation achieves 70% of the pre-mining vegetative ground cover. Surface water drainage control will be maintained until vegetation is re-established to aid in vegetation success and prevent sediment loading.

Public safety and welfare will be considered and if necessary, signage, berms, fences and/or barriers will be installed above high-walls or other areas. A final post mining topography was not developed as part of this PFS. Reclamation planning has been considered as part of ongoing mining operations. The anticipated costs for these efforts have been included in the economic model supporting the project.

17 RECOVERY METHODS

ROM ore produced by open-pit mining methods will be processed, as envisioned, by crushing, roasting, extracting SOP by leaching the calcine with water, solid/liquid separation, evaporation of brine, crystallization and drying and preparation of SOP product for markets. Provisions have been made in the process plant to conserve energy and water through treatment and reuse of effluents and disposal of residues in an environmentally sound manner.

The proposed combination of unit operations in processing alunite ore is based on test results of investigations completed in 2012 and 2013 in support of process optimization at HRI in Golden, CO, Pocock Industrial, Inc., in Salt Lake City, UT and thermal processing systems modeling at FLSmidth in Bethlehem, PA.

Process design criteria for the major unit operations in the proposed integrated process plant complex are summarized below:

- ROM ore production rate is 3.4Mtpy at 2% moisture;
- Plant operation schedule is 330 days per year, 24 hours per day;
- The nominal throughput capacity of the process plant is 425tph;
- Particle size of dry grinding circuit product at P_{80} = 1000μm to the calciner;
- Roasting temperature at 1022°F (550°C) and not to exceed 1112°F (600°C);
- Roaster off-gases are routed as feed to a 2,090 tons per day sulfuric acid plant;
- Water leaching of calcine: 35% solids; 176°F; 60 minutes residence time; and 90% SOP extraction;
- Alumina/silicate leach residues conveyed at 90% solids to tailings impoundment for dry stacking;
- Water withdrawal from wellfield estimated at approximately 800gpm.

Total workforce requirements to operate the processing facility will average 148 throughout the project life. Average manpower requirements for the processing plant are shown in Table 17.1 below.

Table 17.1 Average Processing Plan Workforce Requirements

Category	Y-1 – Y5	Y6-Y10	Y11-Y20	Y21-28	Y29-Y46
Process Operations	68	68	68	68	68
Maintenance	37	37	37	37	37
Electrical / Instrumentation	14	14	14	14	14
Process Assay	6	6	6	6	6
Exec / Tech / Staff	23	23	23	23	23
Total Employees	**148**	**148**	**148**	**148**	**148**

The block flow diagram and general arrangement drawing for the processing plant is presented as Figures 17.2 and 17.3 in Section 28.

17.1 AREA 100-200 - PRIMARY CRUSHING AND DRY GRINDING AND CLASSIFICATION

A. Ore bulk density (lbs/ft^3) 95.0
B. Solids (sp. gr.) 2.46
C. Solids (%) 98.0
D. Moisture (%) 2.0
E. ROM top size (inches) 12
F. Crusher design tonnage (tons/hr) 425
G. Crusher nominal tonnage (tons/hr) 425
H. Crushing Bond Work Index (kWh/ton) 7.06
I. Crushing Abrasion Index (grams) 0.40

17.1.1 Design Criteria

17.1.2 General Requirements

Up to 425stph of ROM ore from the mine sized at minus 12in. will be delivered to the 400 T dual loading hopper that feeds two parallel primary roll crushers – one operational, one standby.

Each roll crusher with a capacity of 425tph will reduce ROM ore size by approximately a 4 to 1 ratio, therefore, the crushed rock will be minus 3in. The roll crushers are sized to operate 7,920 hours per year. The crushed ore will be transferred to a collection bin which will feed two parallel screening/secondary crushing units

The collection bin will feed two dry crushing screens. Screen oversize report to two secondary cage mills. The crushed product from the cage mills will be recycled back to the dry screens via bucket elevators. Screen undersize (P_{80} – 1mm) will be conveyed to the calciner feed bin. A dust collection system will be installed to manage the dust generated from dry crushing.

17.2 AREA 300 – DRYING AND CALCINATION

17.2.1 General Requirements

Crushed ore containing alunite and inert solids at approximately 98% solids (2% moisture) is dehydroxylated and roasted to decompose the alunite. The calcine product contains a mixture of leachable SOP (K_2SO_4) and alumina (Al_2O_3).

A start-up air heater is used during the system start-up to bring the flash roaster up to the auto-ignition temperature of the fuel. A portion of the heat energy in the off-gases is used in steam generation. The calcine particles are separated in a cyclone. The calcine reports to the leaching circuit where it is leached with hot water to extract SOP.

Sulfur dioxide (SO_2) bearing off-gases from the roaster are sent to the gas cleaning section of the sulfuric acid plant and for further processing to recover 93% sulfuric acid (H_2SO_4). FLSmidth generally estimates the concentration of SO_2 and SO_3 in the flues gases from the thermal processing units at 9.44 vol. % as feed to the Acid Plant. This value is an estimate based on similar industry experience. No pilot plant testing has been completed by FLSmidth to-date.

17.2.2 Chemical Reactions

When alunite is progressively heated to higher temperatures during drying and calcination, a number of temperature-dependent transitional phases occur. The stable assemblage at the termination of the calcination cycle comprises potassium sulfate and alumina with evolution of sulfur trioxide.

Dehydroxylation of alunite (Equation 1):

$$\boxed{K_2SO_4.Al_2(SO_4)_3\cdot 2Al_2O_3.6H_2O\textit{(s)} \rightarrow K_2SO_4.Al_2(SO_4)_3\textit{(s)} + 2Al_2O_3\textit{(s)} + 6H_2O\textit{(g)}}$$

Decomposition of aluminum sulfate (Equation 2):

$$\boxed{K_2SO_4.Al_2(SO_4)_3\textit{(s)} \rightarrow K_2SO_4\textit{(s)} + Al_2O_3\textit{(s)} + 3SO_3\textit{(g)}}$$

Overall Calcination Reaction (Equation 3):

$$K_2SO_4.Al_2(SO_4)_3 \cdot 2Al_2O_3.6H_2O(s) \rightarrow K_2SO_4(s) + 3Al_2O_3(s) + 6H_2O(g) + 3SO_3(g)$$

17.2.3 Thermal Processing System Components

The calciner/roaster system components offered by FLSmidth Minerals Pyrometallurgy group in Bethlehem, PA are as follows:

Flash calciner circuit: The crushed ore from the calciner feed bin is delivered to a flash calciner for heating to remove all the chemically bound water and to decompose the alunite to potassium and aluminum sulfates. The material produced is transferred to a flash roaster so that it serves to complete the aluminum sulfate decomposition.

Flash roaster circuit: The pre-calcined material from the flash calciner is delivered to the flash roaster for further heating and to decompose all aluminum sulfate to aluminum oxide and SO_3. It also serves to convert most of the generated SO_3 to SO_2 at reduced conditions.

Flash roaster off gas circuit: Air-to-gas heat exchanges cool the process gas, leaving the flash roaster cyclone to a temperature level suitable for the downstream electrostatic precipitator (ESP). The ESP removes non-condensable particulate from the final system process off gas stream and an induced fan delivers excess gases to acid plant after some amount of gases are recycled back to the flash roaster.

Cyclone cooling circuit: Three cyclones operating in series transfer heat from the flash roaster circuit product to the incoming ambient air stream.

Excess air from cooling circuit: The excess air from the cyclone cooling circuit is delivered back to the combustion circuit as pre-heated combustion/excess air.

Preliminary process design criteria for the calciner/roaster circuit, developed by FLSmidth as adapted from similar experience in the cement industry and based on desktop simulation and general modeling of available information, are as follows:

- Solids content of crushed ore as feed to calciner = 98%;
- Moisture content of crushed ore as feed to calciner = 2%;
- Number of thermal processing units =1;
- Name plate capacity of each thermal processing unit = 425tph;
- Alunite decomposition reactions are carried out at 1022^{o}F (550^{o}C);
- The maximum temperature in the roaster shall not to exceed 1112^{o}F (600^{o}C);
- The residence time in the roaster is about 2 seconds;
- Longer retention times are allowed by the inclusion of fluidized bed holding vessels.

17.2.4 Temperature Dependence of Forms of Alumina

Crushed ore containing alunite and inert solids at approximately 98% solids (2% moisture) is dehydroxylated and roasted to decompose the alunite. The calcine produced contains a mixture of SOP (K_2SO_4) and alumina (Al_2O_3).

The gamma-alumina phase occurs in a porous cubic structure which can be leached with sodium hydroxide (NaOH). It reverts at high temperatures to the recalcitrant alpha form with hexagonal close-packed structures. The temperature limits on thermal processing are, therefore, required assure that the gamma-alumina crystals are the end-product of alunite roasting operations.

17.3 AREA 400 – SULFURIC ACID PLANT

17.3.1 General Requirements

An estimated 2090tpd of sulfuric acid (H_2SO_4) is manufactured at the project site from sulfur dioxide (SO_2) produced by the decomposition of alunite during the thermal processing of ROM ore. FLSmidth estimates the concentration of SO_2 and SO_3 at 9.44 vol. % in the off gases as feed to the acid plant from the proposed calciner/roaster systems, with a throughput capacity of 425tph of crushed ore at 98% solids content.

17.3.2 Major Unit Operations

The process gas from the wet gas cleaning section has excessive water content and must be dried prior to conversion to sulfuric acid which is achieved in the drying tower by recirculating sulfuric acid.

The clean and dry gas from the drying tower is transported to the rest of the plant by the main gas blower. The conversion of SO_2 to SO_3 takes place in an MECS four-pass converter in the presence of catalyst. This conversion consumes oxygen and generates heat. In order to initiate the conversion reaction, the process gas must be heated before contacting the first pass.

Heat generated by the conversion reaction at each converter pass is recovered and used in the cold and hot heat exchangers to heat the gas entering the passes of the converter. Excess heat is recovered by two waste heat boilers to produce medium pressure saturated steam. The production of steam assumes that the condensate returns to the plant. This steam can be used in other areas of the plant. If Potash Ridge does not have uses for this steam, SO_3 coolers will replace the steam equipment.

The process gas from the third pass flows to the interpass tower where the SO_3 is absorbed into a stream of 98.5% H_2SO_4. Gas leaving the interpass tower and containing unreacted SO_2 is reheated and pass it to the fourth pass of the converter.

The acid circulating in the drying tower has a concentration of 96% H_2SO_4 which is diluted by the water removed from the process gas. The acid circulated over the interpass and final absorbing tower is strengthened by the absorption of the SO_3 gas. To counteract these strength changes, 98.5% absorbing acid is transferred to the drying acid system from the interpass tower and 96% drying acid is transferred to the absorbing system.

The heat generated from the process is removed by the acid coolers. Product acid will be produced at 93% H_2SO_4 by adding dilution water and will be continuously pumped through a product acid cooler to storage. Two storage tanks of 10,000t each are included which will provide a storage capacity of approximately 15 days.

17.3.3 Emission Limits for Sulfuric Acid Plants

Emissions to the atmosphere from the sulfuric acid plant at the project site will be in compliance with the applicable air quality permit requirements.

17.4 AREA 500 – CALCINE LEACH AND SOLID/LIQUID SEPARATION

17.4.1 General Requirements

The calcine consists of inert siliceous materials in the ROM ore and K_2SO_4 and Al_2O_3 produced by the decomposition of alunite during the roasting step. Extraction of K_2SO_4 by leaching the calcine with water as the solvent consists of dissolving potassium and sulfate values. The feed to the leach tanks consists of calcine discharged from the roaster and SOP-bearing dust recovered in the ESPs in the roaster circuit. The leachate or the potassium sulfate-rich solution is subsequently processed for the recovery of SOP in the evaporator/crystallizer system.

17.4.2 Calcine Leaching

The operating conditions of the water leach circuit are as described in US Patent 4,031,182 (K. W. Loost, July 21, 1977). Based on bench-scale test results at HRI, it is proposed that the calcine discharged from the roaster will be leached with water at a pulp density of 35% solids. After dissolving approximately 90% of the K_2SO_4 in the calcine, the leach residue slurry at approximately 33% solids is filtered through a battery of belt filters and the filtrate (brine) containing approximately 10.5 to 11.0% dissolved K_2SO_4, is pumped to the evaporator/crystallizer circuit. The filter cake, at about 90% solids, is washed with water and the washed filter cake consisting of inert solids, Al_2O_3 and undissolved K_2SO_4, is conveyed to the tailings impoundment and stacked using a radial stacker.

Process design criteria for leaching the calcine with water, based on results of bench-scale investigations at HRI, are as follows:

- Water is the leaching medium (lixiviant);
- Particle size of calcine P_{80} – 1mm (1.2mm maximum);
- Specific gravity of calcine solids (assumed) = 2.7;
- Solids content of slurry in the leach tanks = 35%;
- Specific gravity of slurry at 35% solids = 1.283;
- Average temperature of slurry during leaching = 176°F (80°C);
- Total residence time in leach circuit = 75 minutes;
- K_2SO_4 recovery during leaching = 90%;
- Recycling filtrate for K_2SO_4 to leach tanks for increased concentration;
- Number of leach tanks in series = 2;
- Residence time in each leach tank = 10 minutes;
- Leach tanks fully baffled, covered, insulated, and steam-jacketed.

17.4.3 Solid/Liquid Separation

The calcine discharged from the roaster is leached with water at a pulp density of 35% solids. After dissolving approximately 90% of the K_2SO_4 in the calcine, the percent solids content of the leach residue slurry is estimated at approximately 31%. As envisioned, the leached slurry is concentrated in a thickener and thickener overflow is pumped to the evaporator/crystallizer. The underflow from the thickener is dewatered in belt filters and the filtrate is pumped to the evaporator/crystallizer circuit. The filter cake at about 90% solids is washed with three-displacements.

Solid/liquid separation is one of the most significant unit operations at the project site because of its importance in energy conservation in the evaporator/crystallizer circuits, in water conservation through reuse of effluents and its impact on the size and type of water and wastewater treatment facilities.

The leached slurry, at the end of the leach cycle consists of inert solids originally present in the ROM ore, Al_2O_3 produced in the roaster, an aqueous solution containing approximately 90% of the K_2SO_4 extracted during leaching plus approximately 10% of unleached K_2SO_4 solids. Preliminary design criteria for the solid/liquid separation unit operations are assumed to be as follows:

- Particle size of calciner feed solids P80 = 1000µm (1.2mm maximum);
- Solids content of crushed ore as calciner feed = 98%;

- Moisture content of crushed ore as calciner feed = 2%;
- Solids content of leached solids filter cake = 90% solids.

17.4.4 Water and Energy Conservation Measures

Measures being evaluated and to be validated in bench-scale tests for conserving water use and for reducing energy consumption in the operations are as follows:

A dry crushing option was investigated in this stage of study as an improvement over wet grinding. This option eliminates the need of wet grinding and subsequent solid liquid separation, hence saving water as well as energy. It also eliminates the drying step before the calciner since the crushed ore at 2% moisture can be fed directly to the calciner without drying, resulting in natural gas consumption savings.

Increasing the pulp density of slurry from 35% to 40 or 50% solids during leaching to conserve water, reducing the size of leach tanks and downstream solid/liquid separation equipment.

17.5 AREA 600 - CRYSTALLIZATION

17.5.1 General Requirements

Once the dissolved K_2SO_4 content of the filtrate from the calcine leach circuit has reached the desktop simulation “equilibrium concentration” estimated to be in the 10.5 to 11.0% range through recirculation of the filtrate, approximately 35% of the Filtrate will be pumped as feed to the crystallizer circuit. The remaining 65% of the filtrate and cake wash water, as required, will be recycled to quench the calcine solids discharged from the roaster and to maintain the solids content of the slurry in the leach tanks at 35%.

Recovery of SOP crystalline product from the filtrate containing K_2SO_4 in solution consists of a crystallizer system followed by separation of the SOP crystals from slurry, drying, sizing, and packaging of the product for shipment.

17.5.2 Mechanical Vapor Recompression Technology

The brine from the thickener and belt filters is combined in an agitated feed tank and transferred to a crystallization circuit for SOP crystal recovery. For crystallization, mechanical vapor recompression technology was selected in order to optimize energy usage.

The product from the crystallizer is transferred to a thickener and the thickener underflow is pumped to a centrifuge to separate the SOP crystals. The thickener overflow and the centrate are recycled back to the quench tank. The centrifuge product is then conveyed to the drying area

Preliminary process design criteria for the crystallizer circuit are summarized below:

- Design capacity of SOP product produced =250,000tpy;
- Purity of SOP product produced = 92.5% K_2SO_4;
- K_2O content of product = 50%.

17.6 AREA 700 – PRODUCT DRYING AND SIZING

The dried product conveyed from the crystallizer system will be treated and packaged to meet market specifications.

17.7 AREA 800 – POWERHOUSE

17.7.1 General Requirements

The two options for providing the electrical power and steam required for the processing facility are investigated and are listed below. These options are based on a sulfuric acid plant as the preferred method of sulfur recovery. The final option selected provides the best project value for the Blawn Mountain project. In addition, both options are based on the same sized natural gas pipeline.

Option A Transmission Line: Install a new 138kV transmission line from Rocky Mountain Power's Three Peaks Substation near Cedar City, Utah to the facility and a 138/12.47 kV substation on the plant site. Install gas-fired boilers on the plant site to provide approximately 200,000lbs/hr of steam required for the processing facility in addition to any potential steam produced by the sulfuric acid plant. Currently no steam is available from the sulfuric acid plant due to the low concentration of SO_2 in the feed system. Option A also includes an emergency natural gas-fired generator.

Option B Co-Generation Facility: An evaluation of turbine vs piston plant design was undertaken to determine the best application for this project. Turbine engines are selected for their large electrical output capacities and their steam generation capabilities. The piston engines are more efficient in producing power only and in small power output configuration.

For this project application, a piston engine plant will require more installed units and will require supplementary fuel firing to achieve the necessary outputs of electricity and steam. The supplementary firing required with a piston engine in co-generation applications negates the efficiency advantage over a gas turbine engine.

A CAPEX/OPEX comparison showed that the overall cost of the turbine engine is lower due to the reduced operating expenses (predominantly gas). Hence the rest of the analysis will consider the turbine engine co-generation option.

17.7.2 Option A: Transmission Line

Transmission Line: The alternative to on-site power generation for the facility is to build a 138kV power transmission line which ties into the existing electrical grid.

Available Technology: The only option for routing an electrical power transmission line is above ground. Due to the remote location of the facility, the routing for an electrical power transmission line will be approximately 46 miles.

Boiler: The processing facility requires approximately 200,000lbs/hr of steam for use in the process. Currently no steam will be generated in the sulfuric acid plant. The steam will be provided using packaged boilers fired with natural gas.

Available Technology: Since high pressure, superheated steam is not required the configuration of the boilers becomes simpler. To achieve the necessary steam production capacity from the boilers most economically the use of packaged boilers was evaluated.

Emergency Power: An emergency generating system is needed for the facility to mitigate the adverse effects of a sudden loss of power on the mineral processing system.

Available Technology: Diesel and natural gas-fired units are available to drive an emergency generator.

17.7.3 Option B: Gas Turbine Generator

The anticipated utility demands for the processing facility are: 42 MW installed capacity and 200,000lbm/hr steam.

A co-generation facility is a combined cycle natural gas turbine facility which includes a two-train system to meet the overall demand with no back-up along with two 4.5 MW diesel generators for emergency power. The turbines selected are GE Lm2500-PR gas-turbines with HRSG with duct burners as backup for steam production. The HRSGs are located inside buildings with the gas turbines outdoors.

Two General Electric LM-2500-pr gas turbines are a good fit for the 42 MW configuration. Once again, the two engines produce the required power while considering the ambient air pressure and temperature and have exhaust energy sufficient to produce the requisite process steam. This configuration does not include complete power production redundancy; however, two of these units are required and the facility would still have half power on inadvertent shutdown of one turbine. Duct burners would provide more than half steam production from a single GTG/HRSG train.

Also included is the required demineralization plant, maintenance and administration buildings and other operational facilities, however, fire protection would be supplied by the site water.

17.7.4 Conclusion

The analysis showed that the overall cost of the co-generation facility is higher than that of the transmission line and concludes that using the transmission line is the path forward, however, this is contingent upon confirming the costs for the power supply from Rocky Mountain Power. Furthermore, over the long term (beyond 10 years), there may be a benefit to the co-generation plant due to reduced cost after the facility is paid off, however, that context has not been considered within this study.

17.8 AREA 1100 – TAILINGS AND RECLAIM

17.8.1 General Requirements

Discussed in more detail in Section 18.

17.9 AREA 1200 WATER DISTRIBUTION AND TREATMENT

17.9.1 General Requirements

Water distribution and treatment facilities at the project site may be grouped into the following major types:

Water Distribution

- Header Tank – from wells
- Fire water
- Day tank to feed water treatment and process water make-up
    - Centrate recycle feed to water treatment (may be a closed loop in the crystallizer-leach circuit)
    - Crystallization condensate to water treatment make-up as required. The balance of the condensate continues to the leach circuit.

Water Treatment

- Potable Water Treatment
- Treatment of groundwater pumped from wells to remove hardness (scale-forming constituents) and disinfection with chlorine for on-site consumptive uses.

Boiler Water Treatment

- Advanced methods of treatment of potable water, including removal of silica, addition of oxygen scavenging and corrosion control reagents, and/or demineralization for use as boiler feed water for steam generation and in the laboratory.

Industrial Effluents Treatment

- Treatment of a bleed stream from the evaporator/crystallizer circuit for build-up of sodium salts and/or impurity control and Thin Film Composite Membrane (TFCM)-type Reverse Osmosis (RO) system for reducing the total dissolved solids (TDS) in decant water from the tailings pond for reuse in the operations or reuse in the operations.

17.9.2 Preliminary Estimate of Water Demand

Based on a zero-discharge facility, preliminary design criteria for water distribution and treatment processes are as follows:

- Potable water consumption - 52gpm;
- Steam production: Steady state demand 200,000lbm/hr;
- Fresh water makeup demand for the processing facility – 232gpm;
- Total water treatment capacity - 582gpm, which consists of:
    - Potable water - 52gpm;
    - BFW to acid plant 1 62gpm;
    - Dilution water to acid plant - 44gpm;
    - Cooling water makeup to acid plant 1 462gpm;
    - Cooling water loop – 120gpm;
    - Boiler loop – 60gpm.

17.9.3 Water Quality Specifications

The Blawn Mountain area has no perennial streams. The water demand for mining and the processing plant will be sourced by extracting ground water from wells. A well-water sampling and analysis program was conducted for the wells drilled by PRC. Water quality from the aquifer of production wellfield was documented previously by ESI with additional characterization work to be completed by PRC, the results from which will be used in the feasibility study and during detailed engineering of the project facilities.

American Society of Mechanical Engineers (ASME) guidelines for water quality in modern industrial water tube boilers are summarized in Table 17.2.

Table 17.2 ASME Guidelines for Water Quality in Industrial Water Tube Boilers

Boiler Feed Water*				Boiler Water		
Drum Pressure, psi	Iron, mg/L Fe	Copper, mg/L Cu	Total Hardness, mg/L $CaCO_3$	Silica, mg/L SiO_2	Total Alkalinity, mg/L $CaCO_3$	Specific Conductance, µS/cm (not neutralized)
0 – 300	0.100	0.050	0.300	150	700	7000
301 – 450	0.050	0.025	0.300	90	600	6000
451 – 600	0.030	0.020	0.200	40	500	5000
601 – 750	0.025	0.020	0.200	30	400	4000
751 – 900	0.020	0.015	0.100	20	300	3000
901 – 1000	0.020	0.015	0.050	8	200	2000
1001 – 1500	0.010	0.010	0.0	2	0	150
1501 – 2000	0.010	0.010	0.0	1	0	100

*Feed Water = Make-up water + Return condensate

Table 17.3 summarizes water quality guidelines for cooling towers recommended by Marley SPX Cooling Technologies. A comprehensive water treatment program, including treatment with biocides to minimize the growth of bacteria (including Legionella Pneumophilia) is required in the operation and maintenance of evaporative-type cooling towers.

Table 17.3 Cooling Tower Water Quality Guidelines

Parameter	Guideline
pH Maximum water temperature Langelier saturation index M-Alkalinity (total alkalinity) Silica Iron Manganese Oil and Grease Sulfides Ammonia Chlorine Organic solvents Total Dissolved Solids (TDS)*	6.5 to 9.0 120ºF 0 to 1.0 100 to 500 ppm as $CaCO_3$ 150 ppm as SiO_2 3 ppm ppm 10 ppm for splash-filled towers None allowed for film-filled towers 1 ppm 50 ppm if copper alloys are present 1 ppm free residual intermittently (shock) Or 0.4 ppm continuously None allowed Over 5,000 ppm can affect thermal performance
Cations: Calcium Magnesium Sodium	 800 ppm as $CaCO_3$ Depends on pH and silica level No limit
Anions: Chlorides Sulfates Nitrates Carbonates/Bicarbonates	 750 ppm as NaCl, 455 ppm as Cl^- 800 ppm as $CaCO_3$ 300 ppm (nutrient for bacteria) 300 ppm as $CaCO_3$ (maximum for wood)
Biological/Bacteria Film-Type Fills: MC75 MCR 12/16	 Aerobic bacteria Count: Less than 10,000 CFU/mL Less than 1,000,000 CFU/mL* when TSS<25 ppm and less than 100,000 CFU/mL when TSS>25 ppm
Total Suspended Solids (TSS)* Film-Type Fill Splash-Type Fill	 Less than 25 ppm No specific limit
Other nutrients	For film fill, avoid fats, glycols, alcohols, sugars, and phosphates

***Notes:** CFU/mL = Colony-forming units per milliliter; TDS = Total Dissolved Solids; TSS = Total Suspended Solids

17.10 AREA 1000 LOAD OUT

17.10.1 General Requirements

The load out associated with the processing facility has been designed to facilitate batch loading of the saleable SOP and Sulfuric Acid products into over the road trucks. All necessary weigh-scales as well as accommodations for ingress and egress traffic have been included. Section 18 discusses saleable product transportation in more detail below.

18 PROJECT INFRASTRUCTURE

The infrastructure needed to support the mine and processing facility includes the following:

- Project access
- Water supply
- Power supply
- Natural gas supply
- Mine facilities
- Surface water management
- Tailings management
- Saleable product transportation
- Miscellaneous support

The aforementioned are discussed in more detail below.

18.1 PROJECT ACCESS

Primary access for the Blawn Mountain Project will utilize existing county roads, Revenue Basin and Willow Springs from State Highway 21 (between milepost 53 and 54). These existing roads reside on land currently administered by both BLM and SITLA. Beaver County has obtained a ROW grant from the BLM to upgrade these existing roads in order to accommodate the transportation needs associated with the project. In addition to these road upgrades, a bypass road will be developed west of project area to provide a safe travel route around the project site for motorists and recreational users. The finished roadway surface will be chip-sealed gravel but is approved for an asphalt pavement surface if needed in the future. The roadway will have two 14ft lanes with 6ft shoulders (total width 40ft) and berms and ditches to manage storm water flows.

An access road into the project site will be developed from the intersection with the county bypass road to the mine site, continuing to the processing plant location. A guard shack will be placed at the intersection of the mine access road and the upgraded county roads discussed above. The location and alignment of the upgraded county roads, bypass road and site access road is illustrated in Figures 16.1 through 16.11 in Section 28. The access road will be constructed using the same general constraints as the upgraded county road segments (total width 40ft).

18.2 WATER SUPPLY

Water sourcing for this project will be provided from the Wah Wah Valley aquifer. The water system will consist of wells, surge pond/tanks, booster pumps, pipeline, storage tanks and water treatment facilities.

18.3 POWER SUPPLY

This study assumes that a new transmission line will be installed. The new line will run to the project from Rocky Mountain Power's Three Peaks Substation near Cedar City, Utah or possibly from the Milford Area where the Rocky Mountain Power infrastructure has recently been upgraded.

18.4 NATURAL GAS SUPPLY

Natural gas will be required for several areas of the processing facility. PRC has been in discussions with utility companies that can supply natural gas for the project. The economics associated with this PFS assume that the natural gas line will be owned and operated by the utility company.

18.5 MINE FACILITIES

Facilities to support the mining operation include:

- Truck shop
- Warehouse
- Administration complex (offices, training, change-house)
- Fuel depot
- Explosives storage
- Equipment ready-line
- Guard shack

The truck shop consists of three repair bays and one wash bay for maintenance of heavy equipment. A tool room, light equipment repair bay, lubrication storage and meeting area are also included in the truck shop. The mezzanine level provides office space for the maintenance manager, planner and foreman. The mine truck shop is shown in Figure 18.1. The mine warehouse provides indoor and outdoor storage space for the material, parts and supplies that support the mining operation. Figure 18.2 shows the layout of the mine warehouse.

The administration complex provides office space for mine management and engineering staff. Within the administration complex are areas for mine employees (both male and female) to change prior to and after shifts. It also provides lockers for storage of personal clothing and items and restroom facilities. Storage for first aid and safety material, an ambulance bay and training rooms will also reside in the administration complex.

A fuel depot will be placed in a location convenient for mine operations. The fuel depot provides fuel and fluids for the mining equipment. This facility will be equipped with rapid fueling systems to minimize equipment downtime. An equipment ready line will be used as a muster and temporary storage location for the mining equipment. It will be equipped with power outlets that supply power for the engine block heaters during cold weather periods. Additionally, this area will also include lighting, housekeeping items and compressed air to assist in preparing the mining equipment for operation.

18.6 SURFACE WATER MANAGEMENT SYSTEM

As previously mentioned in Section 16, storm water controls will be located downstream of all surface disturbances. These controls will consist of diversion ditches, sediment ponds, outlet control structures and a combination tailings storage area/runoff containment structure and a settlement pond. The sediment ponds and diversion ditches will collect and clarify water from the periphery of the site. The drainage from the plant and facilities area will drain to the tailings storage area/collection pond (discussed below), where it will be collected and clarified by providing retention time for sediments to settle out.

18.7 TAILINGS MANAGEMENT

As ore is processed, tailings are produced, requiring storage. Tailings will be transferred to the storage area via overland conveyor. The tailings material is anticipated to be a relatively dry, coarse grained sand that will be roughly 85 to 90% solid content as it exits the processing facility. Tailings will be deposited in the storage area utilizing a combination of portable conveyance equipment and radial stackers. Over the life of the project, approximately 89M yd^3 of tailings will be produced. Based on current testing results and characterization, the tailings material will consist of non-hazardous, non-toxic materials. As such, no liner for the tailings storage area has been proposed.

Table 18.1 shows the amount of tailings released over the life of the project. Figures 16.1 through 16.11 in Section 28 provide illustration of the tailings progression.

Table 18.1 Tailings Storage Volumes (Cumulative)

Year	1	2	3	10	20	30	46
Tailings (Myd^3)	1.1	3.1	5.0	18.7	38.3	57.8	89.1

A collection pond will be constructed below the tailings storage area to collect any drainage associated with the tailings and also any runoff from the site, process facilities, mine areas, etc. A settlement pond will be installed downstream of the collection pond. The location of these water management structures are shown on Figure 16.12. Runoff and drainage water will initially be captured by the collected pond and then transferred to the settlement pond where it will be allowed to clarify. Water will then be pumped back up to the processing plant water tank. These water management facilities will be constructed prior to commencement of operations.

The majority of the tailings area is anticipated to be reclaimed concurrently as operations progress. As soon as practical, the disturbed area behind the discharge conveyor will be reclaimed. This process will be repeated until the full development of the tailings area is completed. The costs for these activities are considered part of the normal operation which is included in the economic analysis.

Final reclamation will include the removal of the water management structure, reestablishment of the drainage channel, spreading of PGM material and seeding with an approved seed mix and covering the steeper slope surfaces with erosion control blankets. Fertilization or other soil amendments will be applied if necessary.

18.8 SALEABLE PRODUCT TRANSPORTATION

Two saleable products will be produced from the Blawn Mountain Project: SOP and sulfuric acid. Different product transportation options have been considered throughout the various courses of study associated with the project; including over the road trucking of SOP and sulfuric acid, a sulfuric acid pipeline and rail haulage of both products. Ultimately, over-the-road trucking was chosen and utilized in this PFS. PRC has been in discussions with transportation contractors that have the ability to facilitate all saleable product transportation and trans-loading requirements. The products will be trucked from the project site to a trans-loading facility that is anticipated to be located near Milford, Utah. The economics associated with this PFS assume that the trans-loading facility will be installed, owned and operated by the product transportation contractor.

18.9 MISCELLANEOUS SUPPORT INFRASTRUCTURE

Some additional support infrastructure has been addressed with allowances common for this level of study. Millcreek has made reasonable assumptions to provide for the required services listed below:

- Helipad
- Sanitary waste treatment
- Fire fighting
- Site transportation

Ambulance service is available to two small hospitals within driving distance of the operation in either Milford or Beaver, UT. Life flight for critical care or trauma care exists to either Dixie Medical Center in St. George, UT or to Salt Lake City, UT. For these reasons, Millcreek has not accounted for a helipad on site. A waste water treatment facility septic tank and absorption leach field to support the operation has been accounted for in project economics. Sludge will be removed and disposed of in a nearby municipal facility in Milford.

Fire suppression for the processing and mine facilities will be managed by a fire water network and portable units. Fire water will consist of a network of fire hydrants around the site. The anticipated fire flow requirements were estimated and this water storage volume was included in the header tank.

Transportation to and from work for full-time staff via high capacity crew busses has been taken into consideration in project economics. In this manner, the traffic associated with the project site will be minimized resulting in less congestions in the area.

19 MARKETS AND CONTRACTS

Millcreek understands that PRC has conducted several market and pricing studies relative to this project and several other studies and forecasts are publicly available. Millcreek has not independently reviewed this information. The information presented below was provided to Millcreek from PRC.

19.1 MARKETS

19.1.1 Agricultural Fertilizer Marker

Fertilizers, a large component of the global chemicals industry, consist of essential plant nutrients that are applied to farmed crops in order to achieve favorable quality and yield. They replace the nutrients that crops remove from the soil, thereby sustaining the quality of crops and are considered the most effective means for growers to increase yield.

Growers adjust the types, quantities and proportions of fertilizer to apply depending on crops, soil quality, weather conditions, regional farming practices, fertilizer products and crop prices. High populations, urbanization, increasing per capita income, changing dietary habits and environmental risks will require continued intensive crop production in a sustainable manner. Demand for higher crop yields will continue to translate into robust demand for agricultural inputs. However, with environmental sustainability taking center stage, input products that deliver positive net economic impact without negatively impacting environment will be receiving better market acceptance with higher demand.

The key nutrients of fertilizer products are classified by three main nutrient category groups: primary macronutrients (referred to as NPK complex), secondary macronutrients and micronutrients as shown below in Table 19.1.

Table 19.1 NPK Complex Nutrient Categories

Primary Macronutrients	Secondary Macronutrients	Micronutrients
Nitrogen (N)	Sulfur (S)	Boron (B)
Phosphorous (P)	Magnesium (Mg)	Copper (Cu)
Potassium (K)	Calcium (Ca)	Iron (Fe)
		Chloride (Cl)
		Zinc (Zn)
		Manganese (Mn)

While the benefits of primary macronutrients (NPK) have been extensively researched and documented, the benefits and yield impact of the secondary macronutrients and micronutrients have recently received significant attention. As the agricultural cycle of growing and harvesting crops depletes soil of all three groups of nutrients, the need to resupply all nutrients or the soil to remain fertile has become one of the main strategies in sustainable agricultural practices.

Sulfur, specifically, has gained significant attention in agriculture over the past several years due to its agronomic benefits and a rapidly developing sulfur deficiency in most of the key crop producing regions. Sulfur is necessary for the production of protein and the development of enzymes and vitamins. In addition to these benefits, sulfur also improves root growth and seed production, aids in the creation of chlorophyll and increases resistance to cold temperatures. The importance of sulfur in crop production, (especially sulfate of potash (SOP)'s form of sulfate-sulfur, the only form of sulfur the plant can utilize) and developing sulfur deficiency has recently positioned sulfur as a "fourth" macro-nutrient in the NPKS complex.

19.1.2 Fertilizer Demand Drivers

Global fertilizer demand is expected to remain robust due to continued high levels of agricultural production in the developed markets as well as the adoption of more intensive agricultural practices in the developing markets. This trend will also be accompanied by the limited ability to increase arable land, changes in diet which will increase production of a broader variety of speciality crops and the consumption of alternative fuels which use crops as feedstock.

As disposable income grows, consumers decrease consumption of staple foods like rice, oats and cereals and increase consumption of premium foodstuffs, such as fruits and vegetables as well as adopting diversified protein-rich diets.

According to FAO's (Food and Agriculture Organization) forecast of global cereal crops (as shown in Table 19.2) production in 2015-2030, a crop yield increase will remain the engine for output growth. Better agronomic practices, genetic seeds, pest control, host resistance and refined nutrient management will be the cornerstone of yield-enhancing strategies.

A more intensive agriculture, in combination with ever-increasing yields, translates into increased use and more intensive management of agricultural inputs like crop nutrients. Nutrient use efficiency such as 4Rs stewardship programs, proliferation of speciality fertilizers, site-specific nutrient management practices, soil testing, fertigation, drip and micro-sprinkler irrigation, controlled release fertilizer products, and other related crop nutrient management efficiency processes, applications and practices will be the key drivers of a fertilizer demand profile going forward.

Forecasted improved nutrient use efficiency will result in increased crop production with the same or smaller quantities of fertilizer usage. FAO estimates global fertilizer use will remain at a 0.6% compound annual growth between 2015 and 2030. However, fertilizer use will be dictated by crop types and geographic regions as shown in Table 19.2.

Table 19.2 FAO Global Crop Types*

	Fertilizer Use			% Growth	
Crop	**1995-97**	**2015**	**2030E**	**2015**	**2030E**
Barley	4.1	4.6	4.9	0.5	0.5
Sugar cane	3.6	5.9	6.9	2.5	1.9
Cotton	4.6	5.8	7.0	1.1	1.2
Maize	19.3	22.0	24.6	0.7	0.7
Other cereals	5.0	4.7	4.9	0.3	0
Rice	21.3	23.0	23.8	0.4	0.3
Soybean	3.8	4.1	4.7	0.3	0.6
Vegetables	4.8	4.2	4.1	0.7	0.5
Wheat	24.7	28.8	31.6	0.8	0.7
Others	36.4	41.1	45.5	0.6	0.6
Subtotal	127.8	144.2	158.1	0.6	0.6
Total	**133.9**	**151.2**	**165.7**	**0.6**	**0.6**

* Note: quantities are in million nutrient tons of N, P2O5, and K2O. Percentage growth rates (annual) are compounded.

North America and East Asia will remain the largest fertilizer markets with Latin America and South Asia markets exhibiting the fastest growth rates in 2015-2030 as shown in Table 19.3.

Table 19.3 FAO Geographical Area Growth Estimation 2015 – 2030*

Crop	Fertilizer Use			% Growth	
	1995-97	2015	**2030E**	**2015**	**2030E**
East Asia	45.5	49.2	52.5	0.4	0.4
East Europe	3.0	4.3	4.8	1.9	1.4
FSU	3.9	4.6	5.1	0.8	0.8
LA	9.7	12.4	14.4	1.3	1.1
East and N. Africa	3.0	3.9	4.7	1.2	1.2
NA	22.6	24.4	26.4	0.4	0.4
Oceania	2.6	2.6	2.7	0.1	0.2
South Asia	18.1	22.2	25.5	1.0	1.0
Sub-Sahara	1.4	1.8	2.2	1.5	1.4
Western Europe	17.9	18.8	19.9	0.2	0.3
Sub-total:	127.8	144.2	158.1	0.6	0.6
Total	**133.9**	**151.2**	**165.7**	**0.6**	**0.6**

* Note: quantities are in million nutrient tons of N, P2O5, and K2O. Percentage growth rates (annual) are compounded.

19.1.3 Potash Market Overview

Potash is a generic term that refers to a group of potassium (K) bearing minerals, naturally occurring potassium salts and the products produced from those salts. Plants depend on potassium for water retention as well as the production, transport and accumulation of sugar. Potassium also supports plant hardiness and resistance to water-stress and disease. Plants deficient in potassium are less resistant to pests and disease and have smaller size, bleak color, poor taste and reduced shelf life.

The amount of potassium contained in potash varies. The industry has established a common standard of measurement in defining a product's potassium content in terms of equivalent percentages of potassium oxide (K_2O). Sylvite, potassium chloride (KCl), is one of the common natural mineral forms of potassium. In combination with halite (NaCl) it forms Sylvinite, the most common and prolific mineral for the production of Muriate of Potash (MOP) fertilizer product. Typical MOP contains approximately 60% of K_2O. Approximately 95% of all potash produced is used in agriculture, mostly in the form of MOP. Potash has no commercially viable substitute as a potassium fertilizer source.

In its processed state, potash appears as a granular mineral of varying sizes and ranges in colour from white to reddish, depending on the presence of trace elements, notably iron, which remains after processing. There are four principal potash grades: granular, coarse, standard and soluble. Granular, coarse and standard grades are differentiated principally by the size of the particles, with granular being the largest and standard the smallest.

19.1.4 Potash Demand Drivers

At the macro level, potash demand depends on the demand for fertilizer, which is expected to remain robust in the coming years. At the micro level, the demand is based on total planted crop acreage, crop mix, fertilizer application rates and farmer economics. Each of these factors is affected by current and projected stocks and prices of crops, governmental agricultural policies, improvements in efficiency, fertilizer application and weather.

According to Potash Corp., global demand of MOP is expected to grow to 62-63M tonnes in 2017 from approximately 58-60M tonnes in 2015, and up to 70M tonnes in 2020, averaging to approximately 2.5-3.0% long-term growth rate. The majority of that increase is expected to come from Latin America, South East Asia, India and China.

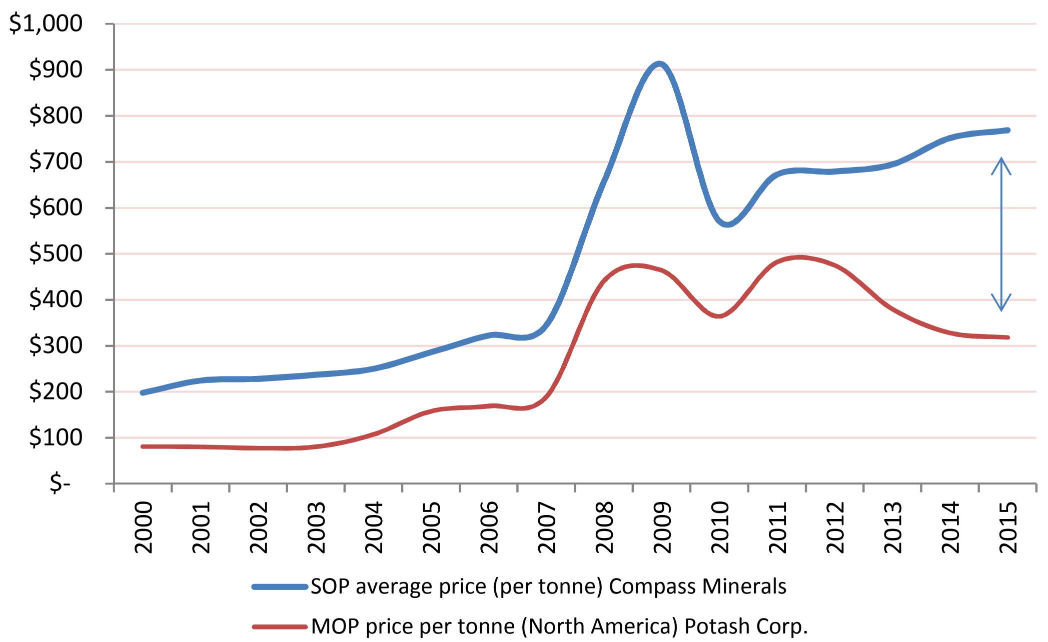
North American MOP prices decreased substantially over the last few years (from $400-$500/tonne in 2011 to approximately $200/tonne in 2016, or down over 50% in absolute terms). The price drop was a result of declining soft commodity prices and collapse of a marketing and distribution partnership between two majors, Uralkali and Belaruskali. The entailing “sales volume and market share grab over price discipline” strategy adopted by the potash industry exacerbated the price decline.

Historically, the price of MOP has been a referencing point to all sulfate of potash (SOP) producers, as SOP, a potassium chloride-free, sulfate-sulfur containing product has historically traded at a consistent premium over MOP. Over the past five years, however, the premium spread of SOP over MOP has expanded significantly in North America as the price decline of MOP has not translated to declines in the price of SOP.

This dynamic largely reflects the constrained supply of SOP in North America (only one North American producer) and beneficial farm economics of a specialty crops grower, the main user of SOP, as opposed to a row crops grower, that primary uses MOP.

Chart 19.1 illustrates the premium SOP commands over MOP for the last several years.

Chart 19.1 Average Realized MOP and SOP Prices in North America



Source: Company Reports, Northern Shoreline Corp., Potash Ridge Corp.

19.1.5 Potash Supply Environment

Although potash is a commonly occurring mineral, the geological conditions needed for its economic extraction can only be found in a few regions of the world. Moreover, many of the currently unexploited potash reserves that have been identified are located in politically unstable and/or remote locations such as the Congo, Ethiopia, Laos, Uzbekistan and the Rio Colorado region of Argentina, which generally require significant new infrastructure to be built to facilitate mine development. Greenfield construction in such locations can be prohibitively expensive. Canada accounts for almost 46% of the world's potash reserves. The majority of remaining reserves are found in Russia and Belarus. There are currently more than 12 countries with active capacity for MOP production, with Canada, Russia and Belarus accounting for almost two thirds of current capacity. Together with Germany, Israel and China, these countries account for approximately 90% of the world's MOP production.

According to Potash Corp, the world MOP industry had operational capability (estimated annual achievable production level from existing operations) of approximately 65M tonnes in 2015. Over 90% of global capacity is held by 10 companies. The recent collapse in MOP prices caused seven MOP producers to curtail production at mines with high operating costs (approximately 7.0M tonnes of global capacity is expected to be closed by 2020). However, a few brownfield and greenfield expansions in North America, Russia and East Asia are expected to come online in 2017-2020, bringing operational capacity to 74M tonnes.

19.1.6 Chloride-Free Potassium Fertilizer – Potassium Sulfate (SOP) Market Overview

While MOP is widely used in all types of farming, the chloride ion within it can be detrimental to some plants, especially tree nut, fruits and vegetables crops. Chloride-free potash is priced at a premium to MOP, due to limited supply, higher production costs, and due to premium end-user markets (specialty crop grower of fruits, nuts and vegetables). There are three main chloride-free forms of potash:

- SOP (Potassium Sulfate)
- SOPM (Potassium Sulfate Magnesia)
- NOP (Potassium Nitrate)

Of these chloride-free products, SOP has the highest potassium content, at 50% K_2O.

SOP is the next most commonly used potassium based product after MOP, comprising approximately 10% of total potash consumption. While SOP can be used in every application that MOP is used, SOP performs particularly well with crops that have a low tolerance to the chloride, such as fruits, vegetables and tree nuts and in arid, saline and heavily cultivated soils. SOP is sold as a standard, granular, and premium soluble grade product.

19.1.7 SOP Specific Demand Drivers

SOP is a fertilizer of choice when the presence of chloride is undesirable. SOP sells at a premium over MOP due its delivery of a high analysis potassium nutrient (K2O content at min. 50%), non-chloride formula, limited primary SOP production and the high cost to produce SOP through secondary production methods.

For the period of 2000 - 2015, SOP premium over MOP in North American markets has increased from approximately $120 to $480/tonne.

The primary users of SOP are specialty crops, broadly defined as tree nuts, fruits, and vegetables crop categories. The major demand market (and markets with significant production) of SOP are the US, Europe and China.

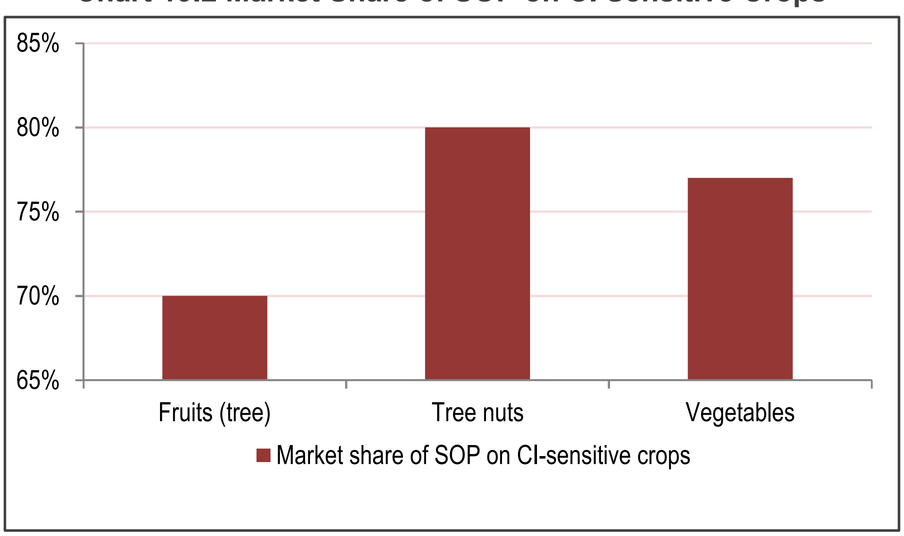
According to Green Markets, total implied demand of SOP was approximately 7.5Mtpy in 2015 with expectation to grow to 9.6Mtpy by 2026. China (with a population of 1.35 billion and the world's largest producer of tobacco, fruits and vegetables) is the largest consumer of SOP, accounting for approximately 56% of global demand in 2015. Over the past 20 years, the demand for SOP in China has experienced significant growth, from approximately 0.5Mtpy in the early 1990s to 4.2 million tonnes in 2015. According to Green Markets, SOP demand growth in China is expected to increase to 4.6Mtpy by 2026. According to CRU, SOP consumption in Europe is approximately 1Mtpy and staying relatively flat through year 2020.

Fully served North American SOP market, as estimated by the North American SOP producer, is approximately 0.75Mtpy. The US market has historically been underserved. Specialty crops are best suited for using SOP and account for approximately 40% of total crop revenues.

SOP actual consumption in the United States is approximately 0.5Mtpy tonnes per annum, (shown below in Chart 19.2) with over 60% of this demand coming from California. California is the number one state in cash farm receipts, growing 58% of US-grown non-citrus fruits, nuts and vegetables and 100% of US almond production (the second highest commodity in value after milk).

SOP is a fertilizer of choice with tree nuts, fruits and vegetables growers in California.

Chart 19.2 Market Share of SOP on Cl Sensitive Crops



SOP consumption in the US is largely constrained by the availability of product and substantially higher premium over MOP. Compass Minerals is currently the only producer of SOP in the US, with sales volume of approximately 400,000 tonnes of SOP in 2015-16.

The Compass' realized SOP sale price in 2015 averaged US$769 per metric tonne, a 140% premium over the average realized MOP price for the same period as reported by Potash Corp. While the company's realized average SOP sale price (averaged over a reported three quarters in 2016) decreased approximately 12% yoy, the SOP premium increased to over 200% due to even greater decrease in MOP prices in the same period.

India represents a sizeable potential market of SOP demand. However, SOP consumption is constrained by the lack of locally manufactured product and SOP pricing premiums not typically suitable for developing and fragmented agricultural markets.

Latin America is the world's second largest consumer of potash, consuming approximately 10-12M tonnes of MOP fertilizer. However, SOP consumption in Brazil is very low, at approximately only 40,000 tonnes. Notwithstanding this fact, Brazil is the largest grower of premium crops, such as citrus fruits and coffee in the world and has 20% of its planted land dedicated to premium crops.

19.1.8 SOP Specific Supply Environment

According to Green Markets, the total worldwide production capacity of SOP was 8.6Mtpy in 2015, with approximately 49% located in China, 20% in continental Europe, 10% in the Americas, with the remaining in other countries. Average capacity utilization rates within SOP industry can vary greatly (60-80%), subject to the producer type (utilization rates are higher for integrated producers), region, market conditions and other factors.

The largest SOP producer is SDIC Xinjiang Luobupo Potash Co., Ltd. (Luobupo) in Western China. The international export market is supplied mainly by two European companies, K+S and Tessenderlo Chemie NV.

Production capacity in China in 2015 was approximately 4.2M tonnes. Luobupo represented approximately 1.6M tonnes of this capacity with Qingshang, the second largest SOP producer in China, at 0.9M tonnes. The rest of China's SOP production capacity consists of smaller, primarily synthetic, SOP manufacturers (Mannheim process). Most production from Europe comes from a conversion process similar to the Mannheim process. In the United States, Compass Minerals produces approximately 400,000 tonnes, through an evaporation process in the Great Salt Lake in Utah and approximately 36,000 tonnes in Saskatchewan Canada.

Of total worldwide SOP production, approximately 15% comes from the evaporation of salt lakes, with the remaining 85% coming from Mannheim Furnaces or similar conversion processes. Opportunities for new production from evaporation are limited and the high cost of Mannheim Furnaces and similar conversion processes makes additional investment to increase capacity using these technologies unlikely.

In recent years, increases in Chinese demand has largely been absorbed through additional supply coming from the newly-commissioned Luobupo project. While the average premium of SOP over MOP has decreased slightly in China, many of the high-cost Mannheim Furnaces remain competitive due to their location and quality/consistency of product compared with Luobupo. Exports from China are negligible and are not expected to increase in the future, as domestic production is subject to export tariffs.

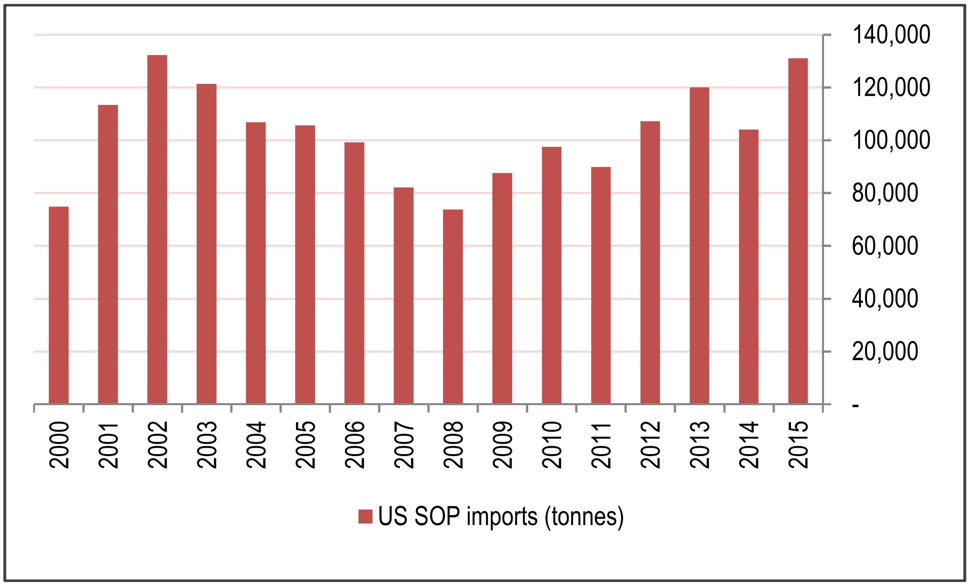
Outside of China, no new major sources of SOP have been developed since the 1990s. The profitability of producers using Mannheim Furnaces or similar conversion processes is largely dependent on the price of MOP, sulfur and energy. Several high cost operations using Mannheim Furnaces or similar conversion processes have curtailed operations due to their high operating cost and the limited market for hydrochloric acid produced as a by-product of the Mannheim Furnace production process.

There are a handful of new primary SOP development projects currently under evaluation, although production from these projects is not expected until 2020 at the earliest. The only new potential SOP production is PRC's Valleyfield Project near Montreal, Quebec which contemplates producing 44,000t per year of SOP using Mannheim Technology within the next 18 months.

19.1.9 PRC SOP Marketing Strategy

PRC intends to target its SOP production primarily to the US market. According to the United States Geological Survey (USGS), the US SOP imports (as per Chart 19.3) were 131,000 tonnes in 2015. US SOP imports have been steadily increasing in the last seven years, growing on average 9% a year.

Chart 19.3 US SOP Imports



Studies undertaken by PRC, based on utilization rates for certain key crops, indicate that there is potential incremental SOP demand of up to 485,000 tonnes per year in the United States, a certain portion of which can be supplied to the Premium SOP Market, thereby maintaining existing premiums within that market.

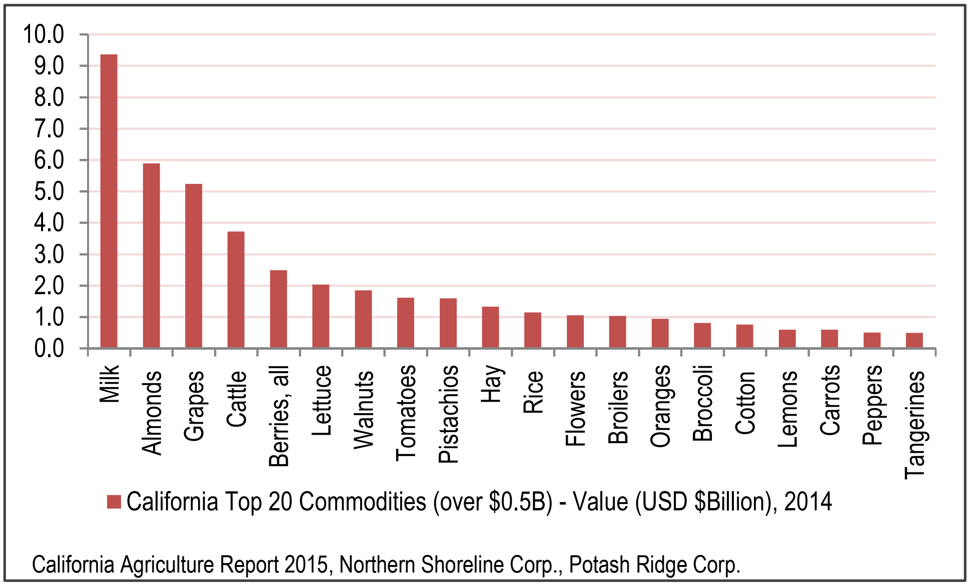
PRC intends to focus its marketing efforts on the economic value of SOP to growers of Premium Value Crops. California will be a key market given its large base agricultural base of premium crops. Florida, Georgia, South Carolina, Texas, New Mexico and the Great Lakes region will be other key targets. Currently, over 100,000 tonnes per annum of SOP is imported into the US markets from Europe, which can be potentially economically displaced by the local US supply.

California: Specialty Crops

California is the North American agricultural powerhouse. Despite the ongoing drought conditions throughout the state, the farm gate value of California growers reached US$54 billion or 13% of the total US crop cash receipts in 2014 crop year (next closest is Iowa at $31 billion). Of that, $21 billion or 40% was attributed to the state’s growing agricultural exports.

California’s top 20 crop and livestock commodities (see Chart 19.4) accounted for more than $43 billion in value in 2014. Ten speciality crops each exceeded $1.0 billion in value in 2014.

Chart 19.4 California Top 20 Crop and Livestock Commodities

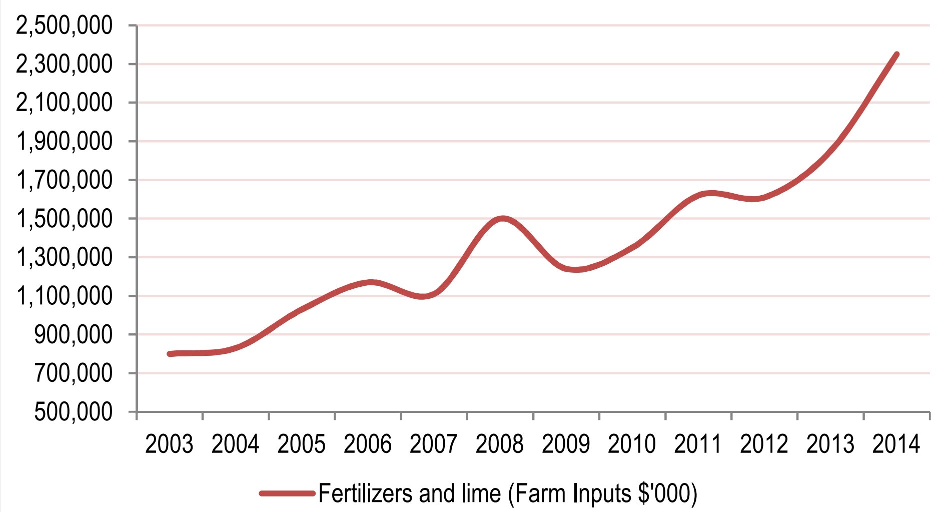


California continues to be the US largest agricultural producer and is the nation's sole exporter of many agricultural commodities, supplying almost 100% of table grapes, raisins, plums, kiwi, dates, olives and olive, oil, figs, almonds, walnuts, pistachios, garlic, and artichokes.

California - Fertilizer Purchases Growth

California's market of speciality crops and high-quality agricultural products translated into a consistently growing demand for fertilizers. The state's growers tripled its budget allocated to fertilizers and soil amendments. California farm inputs (fertilizers and lime) purchases grew to over $2.3 billion in 2014 (see Chart 19.5).

Chart 19.5 California Fertilizer and Lime Statistics



Source: California Agriculture Reports, Northern Shoreline Corp., Potash Ridge Corp.

California: Soil Salinity and Soluble Product Development

California's agriculture depends on water for all aspects of crop production. As the State's has not fully recovered from a 5-year drought and the state's climate become less predictable, there is a heavy reliance on irrigation water and concerns over the quantity and quality of that resource will become more prominent. Current crop production systems heavily rely on irrigation, which necessitates extreme need to improve irrigation systems and nutrient use efficiency of California's crop production processes. Growers in the recent years increased investments in drip and micro-sprinkler irrigation platforms, the proliferation of which, in turn increased demand for soluble fertilizer products.

Soil salinity is an important stressor affecting irrigated crops (see Table 19.4) in many agricultural production areas in California. The stress is often exacerbated during drought periods. While all crops can tolerate a certain level of salts without sufficient leaching, accumulated salts will eventually reach levels that damage crops. Fertilizers that contribute to the accumulation of chloride (Cl-), sodium (Na+) and/or boron (B) can substantially supress crop's growth.

Table 19.4 Salinity in Sensitive Crops

Salinity – Highly Sensitive Crops				Salinity – High to Moderately Sensitive Crops		
Almonds	Limes	Walnuts		Peanuts	Pecans	Watermelon
Apples	Mandarins	Rice		Sugarcane	Pomegranates	Artichokes
Avocados	Tangerines			Flax	Plums	
Bananas	Mangos			Corn	Potatoes	
Blackberries	Oranges			Chick peas	Pumpkins	
Strawberries	Passion Fruit			Alfalfa	Radishes	
Cherries	Peaches			Beans	Spinach	
Grapefruit	Persimmons			Grapes	Sweet potatoes	
Lemons	Raspberries			Macadamias	Tomatoes	

Source: UCANR, Northern Shoreline Corp., Potash Ridge Corp.

Recent improvements in the manufacturing of nitrogen products (N) have created clean, very soluble and reliable N solution products. Potassium (K) dry forms, however, typically contain impurities that can contribute to plugging emitters in drip and micro-sprinkler systems. While potassium chloride is soluble and lends itself to fertigation, the product has a high salt index and is not suitable for California dry soils and/or specialty crops.

Potassium nitrate (NOP) and soluble SOP products, such as Compass Minerals' Protassium+® Soluble Fines are gaining market momentum as a growing speciality soluble products demand.

The "soluble fertilizer" market development is typically based on the key following factors:

- Nutrient requirements of the specific crop;
- Soil and environmental site considerations;
- Timing of application (season);
- Water controls within the drip irrigation system designed to avoid leaching of soluble nutrients below the root zone.

The water controls within the drip irrigation system, soluble product quality and application strategies are becoming of paramount importance as they directly relate to both environmental stewardship, such as the Nutrient Stewardship Principle recently adopted by the fertilizer industry 4Rs, as well as specialty crop grower's economic return.

19.1.10 Sulfuric Acid Market Overview

Sulfuric acid is a colorless, viscous liquid; it is hazardous and corrosive in nature. Sulfur is the major raw material used in the production of sulfuric acid, and around 90 percent of global sulfur production is used in the sulfuric acid production. The global sulfuric acid market is expected to be worth US$85.4 billion by the end of 2023 as against US$67.9 billion in 2014, rising at a 2.6% CAGR between 2015 and 2023.

Sulfuric acid is the major raw material employed in processing of phosphate fertilizers such as calcium dihydrogen phosphate, diammonium phosphate, and triple superphosphate. Processing of phosphate fertilizers mainly require raw materials rich in sulfur content due to which sulfuric acid is most preferably used in this application. Hence rising utilization of fertilizers for agricultural activities in key geographies such as Latin America, Asia-Pacific, and North America are expected to boost the consumption of sulfuric acid by 2023.

Recent developments of fertilizers, chemical manufacturing, and wastewater treatment industries in Asia Pacific especially in China and Japan are anticipated to boost the demand for sulfuric acid as well as its end-products during the forecast period.

In terms of volume, fertilizers were the largest application segment, accounting for more than 60% share of the global sulfuric acid market in 2014. However, chemical synthesis is projected to be the fastest-growing application of the global sulfuric acid market by 2023. This application is estimated to boost sulfuric acid market demand by 2023, owing to the wide utility of sulfuric acid for synthesizing other industrial chemicals.

A significant amount of fertilizer production in the US requires sulfuric acid. Historically, the greatest annual demand for sulfuric acid in the United States is derived from the production of phosphoric acid at approximately 23 million tonnes, followed by industrial uses at 6.7 million tonnes, ammonium sulphate production at 2.2 million tonnes and copper production at 1.6 million tonnes.

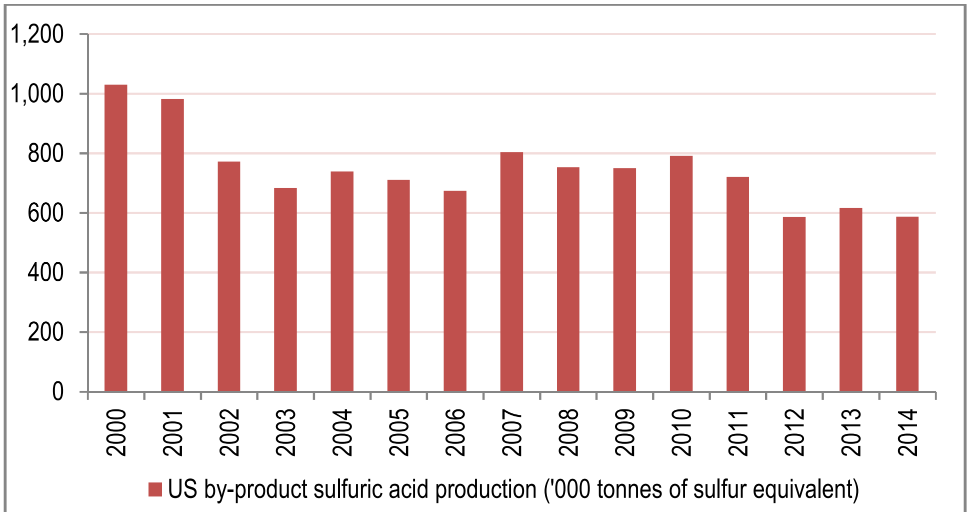
Domestic production of sulfuric acid in the United States has historically been approximately 32 million tonnes per year with historic domestic demand of approximately 34 million tonnes per year. The majority of this supply deficit is met through imports from Canada, Mexico, Western Europe, Japan and South Korea, primarily through the Gulf of Mexico.

Although statistics usually report sulfur and sulfuric acid production and consumption numbers separately, over 80% of sulfur is eventually used for conversion into sulfuric acid. According to USGS, in 2015 recovered elemental sulfur and by-product sulfuric acid were produced at 103 operations in 27 States.

As of 2014, sulfuric acid by-product production at copper, lead, molybdenum, and zinc smelters accounted for about 6% of total domestic production of sulfur in all forms and totaled equivalent of 587,000 tonnes of elemental sulfur. The three largest by-product sulfuric acid plants, in terms of size and capacity, were copper smelters and accounted for the majority (93%) of all by-product sulfuric acid output in the US.

The major copper producers, with each operating its own sulfuric by-product plant, are ASARCO, Rio Tinto Kennecott and Freeport-McMoRan Copper & Gold. All facilities are located in Western US. US by-product sulfuric acid production has been steadily declining by about 3% over the last decade. As of 2014, by-product sulfuric acid production decreased to 587,000 tonnes of sulfur content equivalent, or 43% decrease from 2000. (See Chart 19.6).

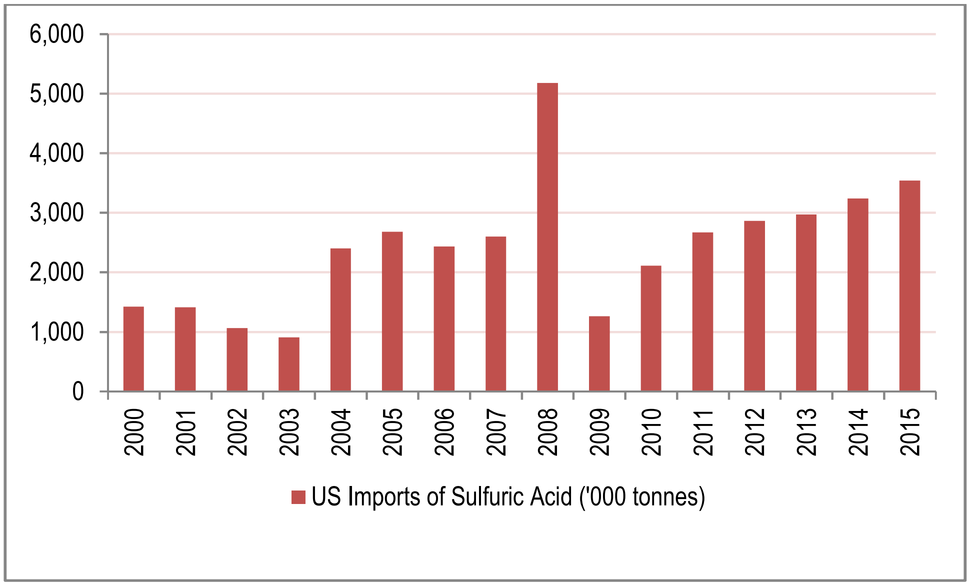
Chart 19.6 US By-Product Sulfuric Acid Production



Source: USGS, Northern Shoreline Corp., Potash Ridge Corp.

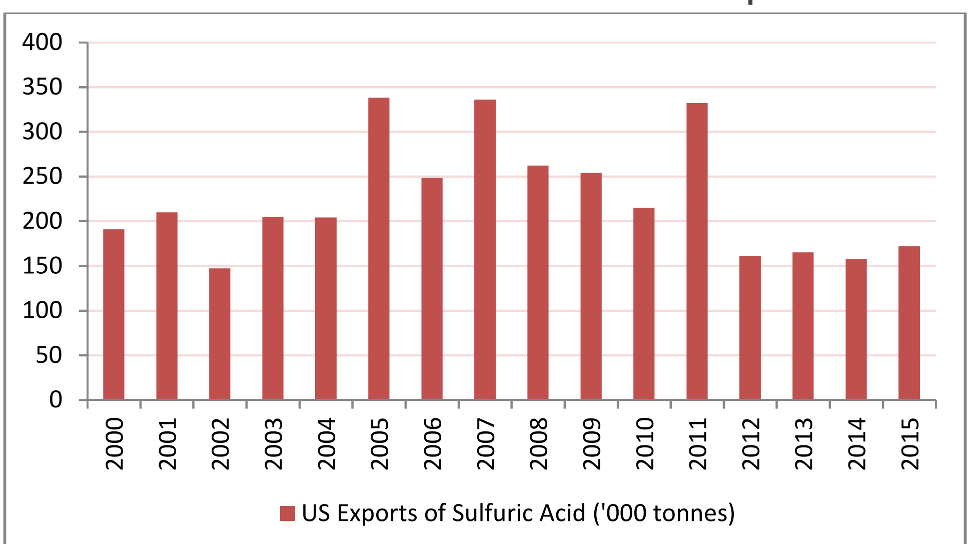
The US both imports and exports sulfuric acid; however, exports historically substantially exceeded export volumes. US imports of sulfuric acid increased significantly over the last decade, and grew on average 19% YOY between 2000 and 2015. As of 2015, acid imports increased to 3.5 million tonnes, or 150% in absolute terms from 2000. (See Chart 19.7).

Chart 19.7 US Sulfuric Acid Imports



US exports of sulfuric acid remained relatively flat in the same period. As of 2015, acid exports were at 172,000 tonnes. (See Chart 19.8).

Chart 19.8 US Sulfuric Acid Exports



Source: USGS, Northern Shoreline Corp., Potash Ridge Corp.

19.1.11 PRC Sulfuric Acid Marketing Strategy

The United States has a significant amount of fertilizer production that requires sulfuric acid consumption. The majority of United States acid consumption in the eastern portion of the US, is largely supplied by dedicated sulfur-burning sulfuric acid plants. The western

United States sulfuric acid consumption is primarily in copper oxide leach and the fertilizer and industrial sectors. Historical demand by phosphate producers is estimated to be approximately 3.0 million tonnes per year, while copper producers in the region historically consume approximately 2.1 million tonnes per year.

In October 2014, PRC entered into a Memorandum of Understanding with a third party sulfuric acid marketer regarding an offtake and marketing arrangement for 100% of PRC's sulfuric acid production from the Blawn Mountain Project. It is anticipated that the marketer will use Blawn Mountain's sulfuric acid to displace sulfuric acid currently being delivered into the region from eastern North America and to potentially supply certain mining projects underway that will have a demand for sulfuric acid.

PRC offers two key benefits to potential acid consumers. First, a long-term security of supply of acid that could make mine development and expansions viable under long term fixed price contracts. Second, PRC can offer price certainty over the life of a long-term contract, with a fixed price linked to either the price of acid or to the price of the commodity the potential customer is processing; thereby reducing the input commodity risk to the consumer.

19.2 PRICING

19.2.1 Potash Price Environment

MOP prices refer to the delivered cost of potash and are usually negotiated as delivery contracts (typically free-on-board, (FOB), or cost-and-freight (CFR)) between suppliers and their customers. By their nature, such contracts contain terms which vary depending on the suppliers' and consumers' geographic locations. The contracts are typically structured as either large, fixed-price sales contracts, monthly contracts with annual minimums or "spot" purchases. Premium MOP grades include coarse and granular material with larger particle sizes (1-4mm) and soluble industrial products generally purer than 98% KCl. Granular and coarse potash is generally priced at a premium.

MOP prices were relatively stable prior to 2007. In 2007, escalating prices helped by fertilizer demand from Brazil, China and India, pushed producer "operating rates" over 90%. The resulting supply/demand imbalance caused MOP prices to continue rising in 2008, reaching a peak of over US$800 per tonne (more than three times the highest price realized between 2001 and 2007). High commodity prices declined sharply due to the economic crisis in 2008 and falling grain prices, this in conjunction with high fertilizer costs squeezed growers' margins. The margin squeeze and tightening of global credit negatively affected growers' ability to pay for fertilizer, which led to a decline in potash demand.

By 2015, the disintegration of a marketing and distribution partnership between two majors, Uralkali and Belaruskali, (a partnership that controlled over a third of the global market MOP) a record crop production in North and South America, and currency depreciations in major crop producing regions all contributed to a significant correction in crop and MOP prices. As of Q3/16 Potash Corp. reported its realized MOP (North America markets) of $155/tonne vs $330/tonne in Q3/2013.

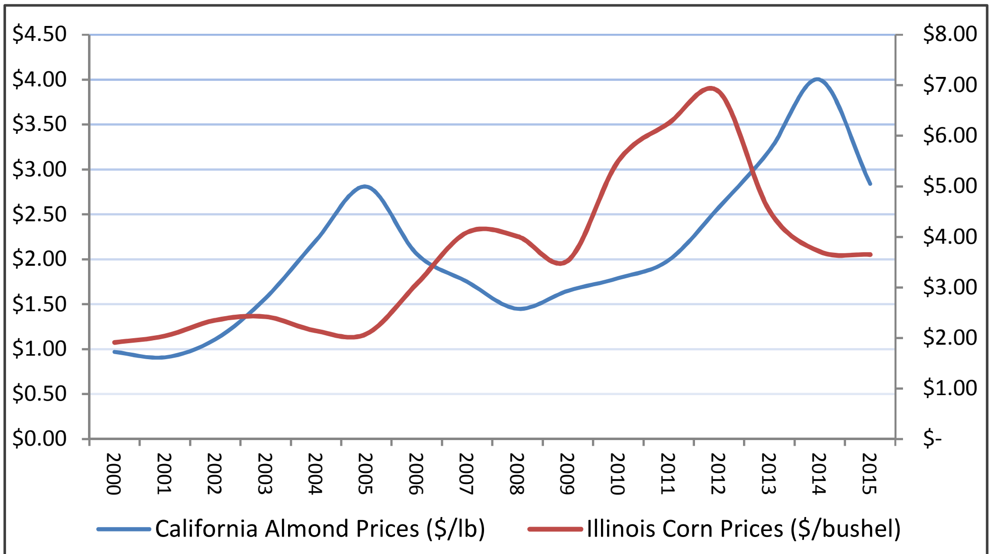
19.2.2 SOP Specific Price Environment

SOP has historically attracted a premium over MOP prices due to its limited availability, cost structure, specialty crop end user markets and the fact that 85% of SOP production comes from high operating costs manufacturing processes that use MOP as a primary feedstock.

Between 2000 and 2011, SOP prices have followed the path of MOP and traded at a consistent premium. However, beginning in 2011, the price trends between SOP and MOP decoupled and the premium increased from approximately $190/tonne to $450/tonne by 2015. (See Chart 19.9).

This price and premium dynamic was primarily a result of collapsing MOP prices due to softening prices of row crops, the main users of MOP, and stable or increasing prices of speciality crops, (almonds, fruits and vegetables), the main users of SOP.

Chart 19.9 Row Crop and Specialty Crop Prices



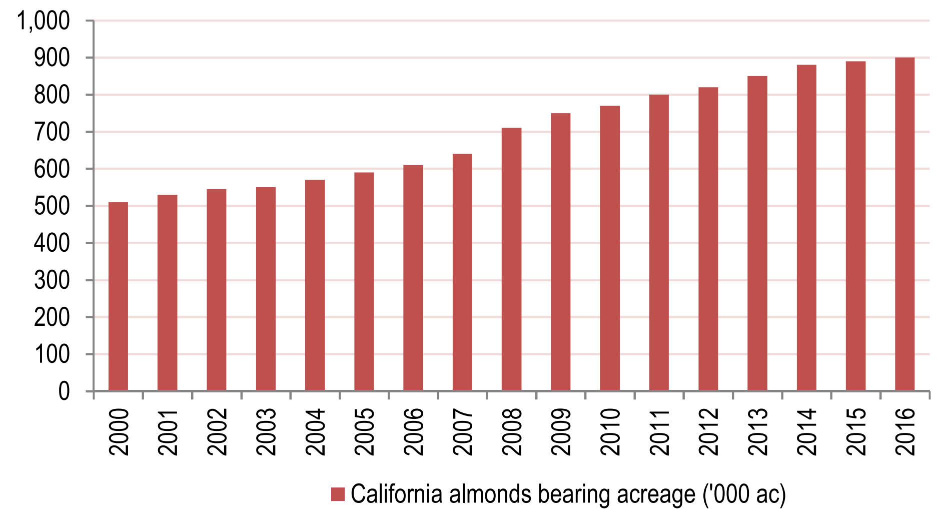
Source: USDA, CAB, Northern Shoreline Corp, Potash Ridge Corp.

The best approximation for the short to medium term forecast of SOP prices will be the price dynamics of specialty crops (tree crops, fruits and vegetables), the main users of SOP, as opposed to MOP prices. Long-term SOP price forecast will depend on speciality crop prices and potential SOP production capacity additions.

The International Nut and Dried Fruit Council (INC) forecasts world tree nut production to reach 4.02Mt in 2016/2017 crop year, a 5.8% annual growth increase. The world production of dried fruit is expected to reach 2.9Mt in 2016/17, a 4.4% production growth. Dried apricots and sweetened dried cranberries are expected to register a 12% increase in the same period with the US being the world's largest producer of both.

The top three producers of global tree nuts and dried fruits are the US, Turkey and China, with California being one of the major key global markets (as per Chart 19.10). California's bearing acreage grew on average 4% yoy and increased 76% in absolute terms from 510k to 900k bearing acres from 2000 to 2016.

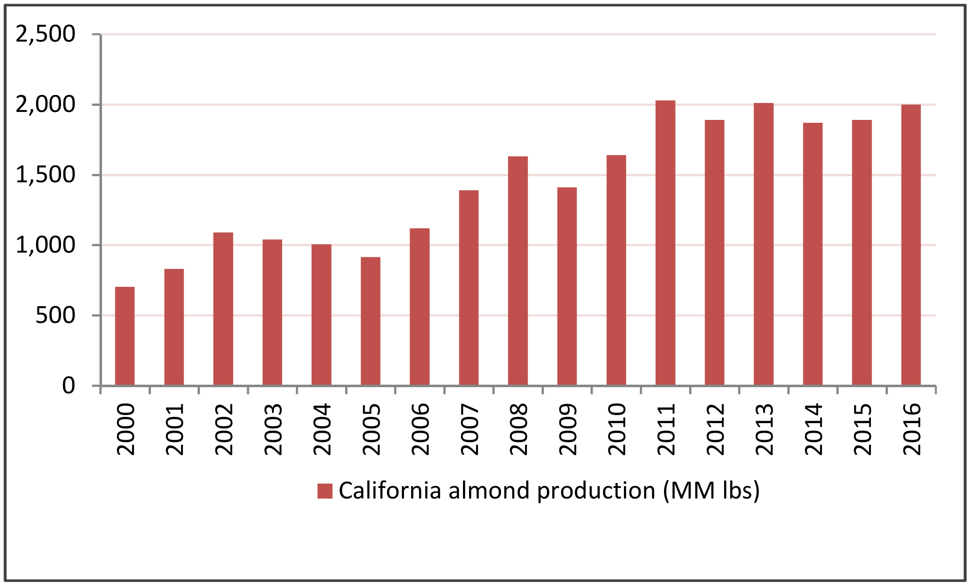
Chart 19.10 California Almonds Bearing Acreage



Source: USDA/NASS, Northern Shoreline Corp., Potash Ridge Corp.

Production of almonds in California showed even faster growth than growth in acreage (as per Chart 19.11). California's almonds production grew on average 8% yoy and increased 184% in absolute terms from 1.3 billion pounds to 2.0 billion pounds from 2000 to 2016.

Chart 19.11 California Almonds Bearing Acreage



We believe the Blawn Mountain project's realized SOP price will have a close approximation to the dynamics of the specialty crops (fruits, nuts and vegetables) markets in California, which continue to show a robust growth and are forecast to grow between 4% and 6% globally. Additionally, since most of the Blawn Mountain project's SOP production will be targeting the US markets, we view Compass Minerals' reported SOP prices as our closest and most accurate price reference point. Compass Minerals reported realized SOP price for the last four quarters was US$640/ton, and for the last eight quarters US$675/ton.

19.2.3 US Mountain West Sulfur Pricing

Based on PRC discussions with the acid marketer, the price for acid is based on Gulf Coast benchmarks adjusted for transportation costs to the local customer.

PRC is estimating that the price for acid, delivered to the end user in the Mountain West Region, to be in the $130 to $150 / ton range on 2020 once acid markets, and copper markets, return to long term pricing norms. This would translate to approximately $115t for the Blawn Mountain acid, FOB rail in Milford, Utah.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 REGULATORY ENVIRONMENT

20.1.1 State Regulations

Mining and processing operations in the US must comply with all applicable federal and state regulations. Utah has primacy over major environmental laws applicable to the project including mining, air and water permitting. Based on review of the current planned operations, the following state regulations apply to the Blawn Mountain Project:

- Title R647 Natural Resources; Oil, Gas and Mining; Non-Coal;
- Title R317 Environmental Quality, Water Quality;
- Title R307 Environmental Quality, Air Quality;
- Title R655 Natural Resources, Water Rights;
- Title R657 Natural Resources, Wildlife Resources;
- Title R850 School and Institutional Trust Lands Administration.

Mining operations must obtain proper permits and approvals and submit proper reclamation surety prior to mine start-up per the R647 State regulations. The Blawn Mountain Project will require permits and approvals from Utah Division of Oil, Gas and Mining (UDOGM).

Utah's Water Quality Act and associated regulations prohibit discharging water or depositing wastes or other substances without prior approval or authorization. Based on current mining plans, the Blawn Mountain Project requires a permit from the Utah Division of Water Quality (UDWQ) to manage their storm water and other process water.

The Blawn Mountain Project will have a septic system and an absorption leach field to handle sanitary waste. Disposal of sanitary waste will require a wastewater treatment facility permit issued by the UDWQ. Water will be treated onsite to provide potable drinking water to staff. Engineering plans and specifications for all public drinking water projects must be approved by the Division of Drinking Water prior to construction. Plans and specifications must be prepared by a Utah-licensed professional engineer.

Based on current mining plans, the Blawn Mountain Project requires a groundwater discharge permit from the UDWQ to address potential groundwater impacts. In addition, a construction permit may be required for certain structures that represent a potential "source" to impact groundwater.

All sources that emit a regulated pollutant are required to obtain an Approval Order from the Utah Division of Air Quality (UDAQ) prior to construction. Based on specific thresholds and its location, the Blawn Mountain Project will have to obtain a Prevention of Significant Deterioration permit from the UDAQ. This permit must be obtained prior to the start of construction of the source. One year after operations begin an operating permit, referred to as a Title V permit, will need to be obtained by the UDAQ.

The Division of Water Rights is the state agency that regulates the appropriation and distribution of water in Utah. A "water right" is a right to divert (remove from its natural source) and beneficially use water. PRC must obtain the necessary water rights to support the project.

State wildlife sensitive species are managed by the Utah Division of Wildlife Resources (UDWR). Field surveys have determined there are no state-sensitive wildlife species within the Blawn Mountain Project area.

20.1.2 Federal Regulations

When BLM lands (minerals or surface) are impacted, BLM approvals are required per the Federal Land Policy and Management Act. Federal actions requiring permits or approvals trigger compliance with the National Environmental Policy Act (NEPA). The level of scrutiny a project receives is based upon the BLM's discretion, the significance of impacts to the environment and/or the public's interest or involvement. The mine and processing plant are located on SITLA controlled mineral and surface land and is not expected to trigger a federal action.

Wetlands and waters of the US (WoUS), defined under the Clean Water Act, are regulated by the US Army Corps of Engineers (ACOE) regardless of land ownership. Based on current mine design and baseline surveys (delineation study), it appears that there are no wetlands, WoUS, or other ACOE jurisdictional waters which will be impacted by the Blawn Mountain Project so an ACOE permit should not be required.

The Endangered Species Act of 1973 was passed by congress in order to protect and recover endangered species and their habitat. Site-specific surveys completed for the Blawn Mountain Project area did not identify any threatened, endangered, or candidate species or potential habitat.

20.1.3 County Regulations

Beaver County's ordinances require mining operations to obtain a Conditional Use Permit (CUP) prior to construction. PRC has been in close coordination with Beaver County and county officials have responded with strong support for the project. In addition to the CUP, PRC will be required to obtain other ancillary permits and approvals from the county in accordance with the county's ordinances.

20.2 HISTORICAL ENVIRONMENTAL STUDIES

In the 1970s ESI proposed to develop the Blawn Mountain Project resource. At that time the land and minerals were managed by the BLM. In 1977, the BLM completed an environmental review, an EIS level study at the time, on the proposed project in compliance with NEPA. Subsequently, the BLM, through a land exchange process, granted the Blawn Mountain Project land and minerals and other surrounding lands to SITLA.

20.3 ENVIRONMENTAL SETTING

An environmental study area was delineated, covering all areas proposed for disturbance, plus adequate room to adjust areas as necessary. Surveys and studies have been completed for the following disciplines:

- Air quality;
- Archeological resources;
- Wildlife habitat including threatened, endangered, and sensitive species;
- Vegetation including threatened, endangered, and sensitive species;
- Soils;
- Surface and groundwater;
- Wetlands and waters of the US.

Each of these disciplines is discussed below.

20.3.1 Air Quality

In September 2012, a meteorological monitoring station was installed near the Blawn Mountain Project and a particulate monitoring station and meteorological station was installed closer to Milford, Utah. Both monitoring stations started recording data in October 2012 and completed the one-year monitoring requirement on September 30, 2013. The meteorological and particulate matter data has been reviewed internally and accepted by Utah Division of Air Quality. This data will be used to support an air quality permit application.

20.3.2 Archeological Resources

The entire environmental study area, the Revenue Basin/Willow Springs Road and the wellfield and water pipeline alignment have been surveyed for archeological resources. Archeological sites were identified. The significance of these sites has been evaluated by SITLA. Significant sites will be avoided, mitigated or managed as appropriate.

20.3.3 Wildlife Habitat Including Threatened, Endangered, and Sensitive Species

The entire environmental study area was surveyed in May 2013 to evaluate general wildlife habitats and determine the presence of any threatened and endangered species or their habitats protected under the Endangered Species Act, or other state designated sensitive species or their habitats. This information is necessary to support permit applications. No wildlife species listed under the Endangered Species Act or Utah State Sensitive Species were found within the project area.

20.3.4 Vegetation Including Threatened, Endangered, and Sensitive Species

The Blawn Mountain Project is located in the pinyon-juniper community as defined by the BLM (1977). Significant portions of the valleys of the project area have been extensively chained by local ranchers to remove Juniper and Pinyon to improve grass growth to support livestock grazing. Site specific surveys completed for the environmental study area did not find any vegetation species listed under the Endangered Species Act or any potential habitat.

20.3.5 Soils

An analysis of soils was conducted in order to determine soil suitability and salvage depths for use during revegetation. Results of this survey were used in the preparation of permit application and for reclamation planning. The soil survey for the environmental study area was completed in August 2013.

20.3.6 Surface and Groundwater

Surface and groundwater resources are detailed in Section 5 of this report.

20.3.7 Wetlands and Waters of the US

A WoUS inventory, which includes jurisdictional wetlands administered under Section 404 of the Clean Water Act, was completed for the entire lease area, the water development areas, pipeline route and access roads. Several springs were identified during the survey. None of these springs would be impacted or disturbed as a result of the project.

20.4 ACCESS AND UTILITIES

20.4.1 Access Road

Existing Beaver County maintained roads (Revenue Basin and Willow Springs roads) provide access to the Blawn Mountain Project area. In its current condition, the road is not adequate or wide enough to accommodate the type and amount of vehicles needed to support the project. The land adjacent to the road right-of-way is managed by the BLM and impacts to this land required for expansion require a (Right of Way) ROW grant from the BLM.

Beaver County submitted a SF-299 application to apply for a ROW across BLM lands in July 2012 on the basis that improvement of the road will enhance economic development for future uses in their county as well as adjacent counties. A draft Environmental Assessment was prepared documenting the baseline environmental conditions and the impacts to the environment from the proposed upgrading. The ROW grant was issued to the County in June 2013.

Preliminary construction activities started on the road later in the summer of 2013 with major construction at a later date. PRC will work with Beaver County to develop a road use agreement to allow use of the road to support the Blawn Mountain Project.

20.4.2 Natural Gas Line

Major natural gas suppliers in the state have been contacted regarding natural gas supply to the Blawn Mountain Project. Permits required to construct and maintain the gas pipeline will be the responsibility of the utility provider. Negotiations to allocate both permitting and construction costs associated with the supply of natural gas to the project have occurred with these providers.

20.4.3 Power Transmission Line

Rocky Mountain Power has assessed the engineering, cost and available capacity associated with a power transmission line to the project. Permits required to construct the line will be part of any construction agreement with Rocky Mountain Power and are expected to be obtained and managed by the utility.

20.4.4 Product Transportation

Initially, a rail line for transporting SOP and sulfuric acid was included in the project design. For that reason, PRC contacted Union Pacific (UP) to assess a rail line to the project. Since the original design, rail haulage has been replaced with truck haulage to the Milford area with a rail load-out in Milford. Permits required for the storage and loadout facility in Milford will be obtained by the trucking company. In the future, rail haulage from the site may become economically feasible. At that time, UP or other partners as determined, will assess engineering and cost associated with the rail line. Permits required to construct the rail line in the future will be part of any construction agreement with UP, or other partner, and managed by that entity.

20.4.5 Water Line

As discussed in Section 18, water necessary to operate the project will be provided mainly from a wellfield approximately seven miles north-east of the project. Water from this wellfield will be piped to the site. The pipeline and associated wells and infrastructure will be on SITLA lands and will require a ROW grant from SITLA prior to construction.

20.5 MAJOR OPERATING PERMIT AND AUTHORIZATIONS

The following discussion and Table 20.1 identify the major permits and approvals that need to be obtained prior to the construction and start-up of the mine and processing plant.

Table 20.1 Major Required Permits

Major Permits or Approvals	**Issuing Agency**
Exploration Permit	Utah Division of Oil, Gas and Mining
Large Mine Operation Approval	Utah Division of Oil, Gas and Mining
Water Appropriations	Utah Office of State Engineer
Groundwater Permits	Utah Division of Water Quality
Air Quality Permit	Utah Division of Air Quality
General Multi-Sector Industrial Storm Water Permit	Utah Division of Water Quality
Army Corps of Engineers Jurisdictional Waters Concurrence	US Army Corps of Engineers
County Conditional Use Permit and Other Permits	Beaver County
Water Treatment Plant	Utah Division of Drinking Water
Waste Water Treatment Plant	Utah Division of Water Quality

The permits listed in Table 20.1 are not meant to be all-inclusive and cover only the major permits required for the mine and processing plant. In addition, various ROWs across state lands will need to be obtained from SITLA in order to construct the water pipeline and to upgrade existing roads. PRC has been actively working with SITLA and obtaining these ROWs is expected to be very straightforward. Any federal ROWs and other permits required for utilities for the project, with the exception of the access road which has already been granted to Beaver County from the BLM, will be the responsibility of the service provider.

20.5.1 Exploration Permits

Exploration activities of minerals require an approval from UDOGM. Exploration activities within the lease area are being completed under exploration permits E/001/0171 and E/001/0182. The holder of these permits is UAC.

20.5.2 Approval for Large Mine Operation

The Notice of Intent to Commence Large Mining Operations must contain a complete description of the existing environmental resources and impacts. Environmental baseline studies necessary to support the application are complete. The Notice of Intent must include a description of mining methods, a comprehensive reclamation plan and will identify the financial security acceptable to UDOGM to cover the costs of reclamation to be completed by an independent third-party as required under R647 administrative rules. Execution of the acceptable financial security instrument will be required in advance of commencing mine activities.

Approval of a Notice of Intent to commence Large Mine Operations in Utah can occur within 6-9 months of an application submittal. The Notice of Intent was submitted to UDOGM in December, 2013 with an approval granted by UDOGM in August of 2014.

20.5.3 Water Appropriations

Water is available adjacent to the project area for which the state is willing to issue rights, or as appropriate, water for activities that will put the water to a beneficial use. Based on the criteria the state uses to issue water rights, a defensible appropriations application for water within the Wah Wah Valley was filed with the State Engineer’s Office. The State Engineer reviewed the application. A site visit with State Engineer representatives and other affected parties was completed in summer 2013 to discuss issues associated with the pending application. A hearing was conducted in November 2013, and a decision to grant water rights to the project issued in June, 2014.

20.5.4 Groundwater Discharge Permit

A groundwater discharge permit application requires the completion of sufficient groundwater investigations in order to evaluate potential impacts to nearby waters and if necessary, provide sufficient mitigation. Ten groundwater monitoring wells were drilled to help characterize the hydrogeologic conditions of the lease area, eight encountered water. These eight wells were completed and equipped for routine monitoring. The hydrogeologic interpretation of data from these wells will be included in the groundwater permit application for the project.

Subsequent to approval of the groundwater discharge application by the UDWQ, it may also be necessary to file a construction permit with UDWQ to validate the engineering and designs of storage facilities of materials with potential impacts to the groundwater system of the area. This would include engineering designs for the tailings facility or other operations that may potentially impact groundwater.

Groundwater discharge permit applications typically are processed in approximately 6-9 months. The groundwater discharge permit application was submitted to UDWQ in January, 2014. It was approved by UDWQ in July, 2014. Approvals for construction applications typically take a much shorter time and can be approved in as few as three months.

20.5.5 Air Quality Permit

In September 2012, a meteorological monitoring station was installed near the project area and a particulate monitoring station and meteorological station was installed closer to Milford, Utah. Both monitoring stations started recording data in October 2012. The one year data collection requirement was completed September 30, 2013.

Preliminary modeling will be completed to assess the impact of the project to ambient air quality. Once modeling is completed to demonstrate that the project can meet the applicable air quality standards, the application can be prepared and submitted for agency review. The review process can take between 9-12 months.

One year after the start of operations, PRC will apply for an Operating Permit, also referred to as a Title V Permit. This permit grants the source permission to continue to operate while self-reporting on performance.

20.5.6 General Multi-Sector Industrial Storm Water Permit

A storm water pollution prevention plan (SWPPP) must be prepared as outlined in the general industrial permit prior to receiving permit coverage. The drainage control plan developed as part of the mining and reclamation plan will be used to develop the SWPPP. The SWPPP must be fully developed and permit coverage granted prior to breaking ground at the site.

20.5.7 Army Corps of Engineer's Jurisdictional Waters

A delineation report and WOUS report for the entire lease area was completed. Site surveys for the entire lease area, the water pipeline route and access roads were included in this submittal. The delineation survey report and WOUS was submitted to the ACOE for their review and concurrence in the fall of 2013. A site visit of the area occurred in November 2013 with ACOE. In March, 2014 ACOE concurred with the findings of the delineation and WOUS reports and issued a "no permit required" letter at that time.

20.5.8 County Conditional Use Permit and Other Permits

PRC has been proactive in maintaining good communication with the local community. To date, county officials as well as local ranchers have expressed strong support for the project, and have expressed high interest in seeing the project succeed. With this level of support for the project, the CUP should be issued without significant challenges. Anticipated time for approval would be 2-4 months once all the supporting studies have been completed.

20.5.9 Water Treatment Plant Permit

Water will be treated onsite to provide potable drinking water to staff. Engineering plans and specifications for all public drinking water projects must be approved by the Division of Drinking Water prior to construction. Plans and specifications will be prepared by a Utah-licensed professional engineer. PRC will prepare and submit the application 6 months prior to when the approval is needed.

20.5.10 Wastewater Treatment Plant Permit

The Blawn Mountain Project will have a septic tank and an absorption leach field to handle sanitary waste. Disposal of sanitary waste will require a wastewater treatment facility permit by the UDWQ. PRC will prepare and submit the permit application to UDWQ in sufficient time required for the project.

20.6 SOCIAL OR COMMUNITY IMPACT

Representatives of Beaver County have expressed strong support for the project. Beaver County has pursued the ROW grant from the BLM in order to complete road improvements to Revenue Basin and Willow Springs roads to support economic development. The proposed project is consistent with the Beaver County General Plan. The Blawn Mountain Project will assist SITLA in meeting objectives outlined in the "School and Institutional Trust Lands Management Act" by optimizing trust land revenues.

SITLA has teamed with PRC in the development and submittal of a water appropriations application to support the project. SITLA's participation substantiates their interest in seeing the project succeed to further their mandate of promoting sound economic development.

20.7 SUMMARY

A strong permitting and environmental strategy has been developed and executed to support the permitting timeline. PRC has been very diligent in evaluating the project area's environmental conditions in order to satisfy permitting and regulatory requirements. All of the required environmental baseline field studies and surveys were completed to support the preparation of most major operating permit applications.

The likelihood of a federal action requiring a NEPA analysis for the mine and processing facility is minimal. Based on completed wetland and WOUS delineations and approval by ACOE, this potential and most probable trigger has been avoided.

The mining operation plans have been developed to meet all the regulatory requirements to operate a mine in the State of Utah. The Large Mine Operation and Discharge permits have been granted. Other significant permits including the Air Quality Approval order and Construction permit will be submitted when final design is completed.

21 CAPITAL AND OPERATING COSTS

As previously mentioned, this report summarizes an updated PFS performed on the Blawn Mountain Project. The previous PFS which was completed in late 2013, analyzed several different operating scenarios and examined their respective capital and operating costs. These analyses consisted of various trade-off studies to determine which scenarios produced the most favorable costs and examined such items as: flotation versus whole ore processing (calcining, roasting and leaching), various ROM production rates, optimizing ore cut-off grades, power supply options, etc. Ultimately, the results of the trade-off studies showed that whole ore processing in combination with higher ore cut-off grades produced the most favorable economics.

With the exception of the use of dry grinding and a different ROM production rate (currently 3.4Mtpy versus 10.6Mtpy in the previous PFS) in combination with further ore cut-off grade optimization, the primary basis of operation has largely stayed the same for this updated PFS. The basis of operation includes:

- Whole ore processing (calcining, roasting and leaching);
- Annual ROM ore production rate of 3.4Mtpy;
- Ore cut-off grades of 3.75% K_2O for Area 1 and 3.50% K_2O for Area 2;
- Line power;
- Dry grinding.

The production capacity of the calcining unit is the primary determining factor in the annual ROM production rate. This PFS assumes the processing facility will utilize a single calcining unit. A capacity scaling analysis was performed that showed that an ore feed rate of 425tph (3.4Mtpy) maximizes the capital effectiveness and scalability of the calcining area while maintaining positive project economics.

Millcreek examined several mine-plan and ore cut-off grade scenarios using the aforementioned production rate. Ultimately, the mine-plan was optimized to utilize ore cut-off grades of 3.75% K_2O and 3.50% K_2O for Areas 1 and 2 respectively. Lower grade ores (down to 2.75% K_2O) are stockpiled and processed later once active mining has ceased in Areas 1 and 2.

As with the earlier work associated with the project, power supply options were examined. A study was performed between a co-generation facility or utilizing a transmission line supplied by Rocky Mountain Power. The study concluded that the transmission line was the preferred option.

Crushing and grinding options were examined to find the optimal method. Previous study work identified wet grinding as the preferred method. This PFS examined five different dry crushing circuit options. Ultimately, it was found that a substantial economic benefit can be gained by using dry grinding.

21.1 CAPITAL COSTS

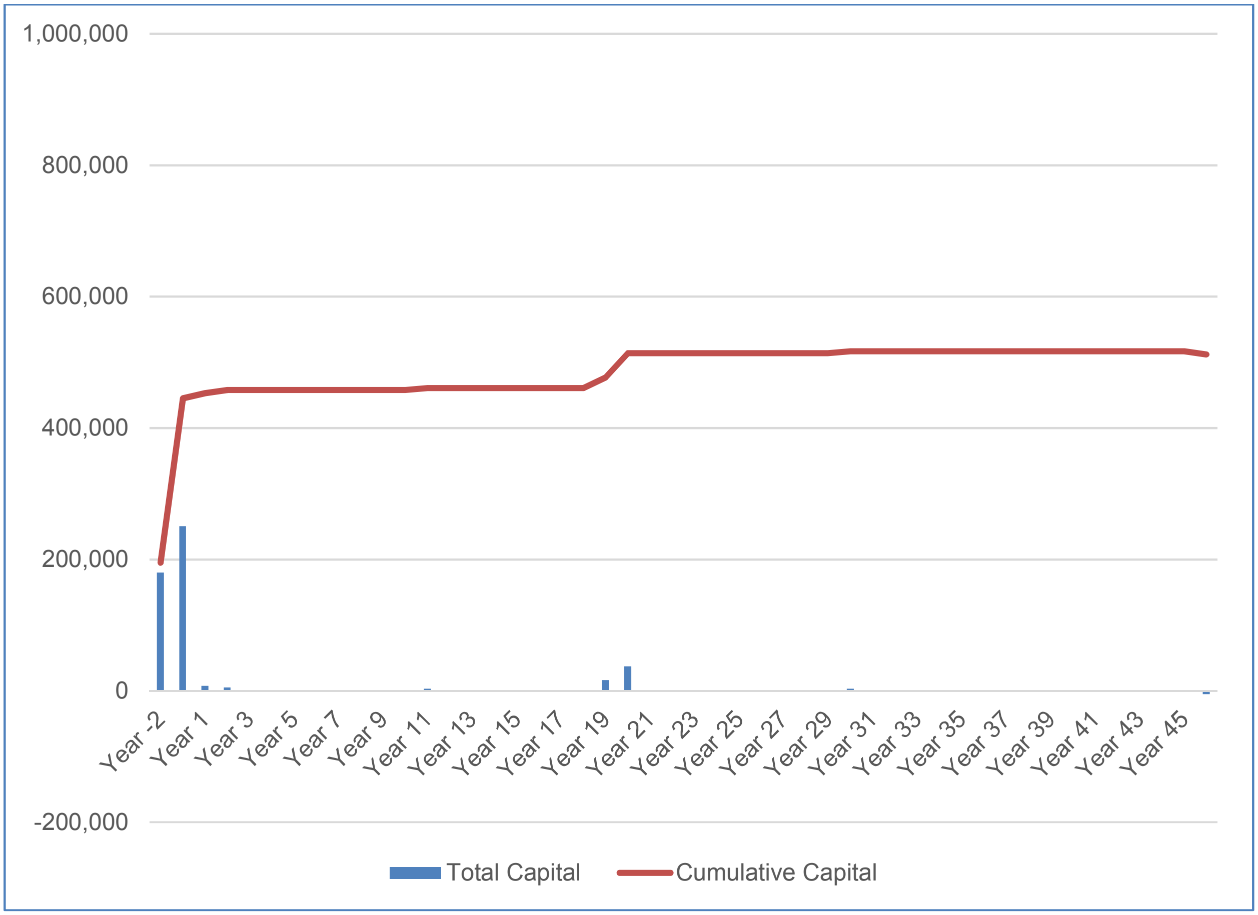
Capital costs for the Blawn Mountain Project were developed by Millcreek and RDI and are summarized below in Table 21.1. As previously mentioned, SNC developed capital costs for the processing facilities based on the trade-off and scaling studies. SNC's capital estimate for the processing plant total $411.4M (including 25% contingency). A breakout of the capital estimate is presented below in table 21.2. Table 21.1 shows the capital for project infrastructure, the processing facility, product storage, handling and indirect costs (engineering, construction management, field expenses, etc.). As mentioned earlier, mining operations will be performed by a contractor. Capital costs associated with the mining equipment are accounted for in the mine operating costs which are discussed later in the report. A contingency of 25% was added to the capital items shown in Table 21.1. The annual capital expenditures as well as the cumulative capital over the life of the project are illustrated on Chart 21.1 in 2016 constant dollars.

Table 21.1 Total Project Capital Estimate (US$M*)

	Year -3	Year -2	Year -1	Total Construction and Development	Sustaining Capital	Total Life of Project Capital
Project Infrastructure	$0.0	$1.3	$15.2	$16.4	$2.1	$18.5
Processing Plant & Product Handling	$0.0	$109.1	$133.4	$242.5	$47.2	$289.7
Indirect Costs	$12.1	$33.5	$64.6	$110.2	$0.0	$110.2
Contingency	$3.0	$36.0	$50.1	$89.1	$12.3	$101.4
Total	**$15.2**	**$179.8**	**$263.2**	**$458.2**	**$61.6**	**$519.8**

*Rounded

Chart 21.1 Annual and Cumulative Capital (US$M)



The information shown in the table and chart above does not include capital costs for access road upgrades, transmission line, natural gas pipeline, acid plant or the water supply facilities. These items are assumed to be either provided by a third-party or financed through state and federal programs and have been accounted for in the operating cost estimates which are discussed in more detail below. PRC has been in discussions with and received indicative estimates from various third parties with respect to the majority of these items. These estimates are discussed and presented in further detail below.

21.2 BASIS OF ESTIMATE

For this PFS update, several estimating techniques were employed to develop capital cost estimates including: industry costing guides, scaling and factoring information from the previous PFS where appropriate, historical data and information from other similar projects, factoring bulk materials by major equipment costs by area and budgetary quotations for the primary and secondary crushing equipment, calciner/roaster, crystallizer equipment and the acid plant. The accuracy of the estimate provided is approximately -30% / +30%. The default currency presented in this document, unless otherwise stated, is in constant 2016 United States dollars (US$).

21.3 PROJECT INFRASTRUCTURE

Infrastructure capital required to support the project that is not included in the processing plant capital is allocated in project infrastructure. This category includes the mining facilities (truck shop, warehouse, administration complex, fuel depot, etc.), tailings handling equipment, surface water management, (ponds, ditches, etc.), as well as water treatment and water management. The majority of these facilities will be installed during plant construction. A portion of the surface water management facilities will be put into place prior to any construction occurring at the site.

21.4 PROCESSING PLANT

Capital costs for the processing plant and associated infrastructure are categorized into several separate areas (Area 100 through Area 1000). Table 21.2 below summarizes the capital costs for each individual area. In general, the capital costs for each area include allocations for mechanical equipment, installation, piping, structural steel, concrete, foundations, buildings, electrical, instrumentation, excavation, site preparation, insulation and painting. Table 21.2 also presents capital cost information for indirect costs, which are discussed below in more detail.

21.5 INDIRECT COSTS

Indirect costs account for engineering, procurement and construction management (EPCM) services, field expenses, contractors overhead and profit, freight and sales tax. EPCM costs are estimated at 16% of direct costs. Field expenses include temporary construction, surveying, site security, supplies, consumables, etc., and are calculated at 10% of direct costs. The overhead and profit allocation for the construction contractor is assumed at 5% of direct costs. Freight costs have been included at 3% of direct costs and sales tax is 5.95% on 30% of the direct cost.

Table 21.2 Processing Plant Capital Detail (US$M)

Description	Total Cost (US$M)
Direct Cost Summary	
Area 100 - Primary Crushing	$ 4.6
Area 200 - Dry Grinding and Classification	$ 14.6
Area 300 – Calcination and Roasting	$ 113.4
Area 400 - Acid Plant (Third-Party Build, Own, Operate)	$ 0.0
Area 500 - Calcine Leaching and Solid/Liquid Separation	$ 17.5
Area 600 - Crystallization and Sop Product Solid/Liquid Separation	$ 20.2
Area 700 - Product Drying and Sizing	$ 15.9
Area 800 – Steam Plant, Substation & Backup Power Generation	$ 17.7
Area 900 – Plant Administration Complex and Warehouse	$ 4.0
Area 1000 – Product Loadout & Storage	$ 17.9
Auxiliary Services – Electrical, Steam Distribution, Site Prep., etc.	$ 16.8
Total Direct Cost	**$ 242.5**
Indirect Cost Summary	
EPCM Cost	$ 38.8
Field Expenses	$ 24.2
Contractors O & P	$ 12.1
Tax and Sales Tax	$ 11.5
Total Indirect Cost	**$86.7**
Total Installed Project Cost (Excluding Contingency)	**$329.2**
Contingency (25%)	**$82.3**
Total Installed Project Cost (Including Contingency)	**$411.4**

21.6 CONTINGENCY

As mentioned earlier, a contingency of 25% was added to the direct capital costs for the processing plant. Contingency is an integral part of the estimate and used as an allowance for the undetermined cost of items that will be incurred within the defined project scope. The contingency covers the cost of these unforeseen items due to the lack of detailed information.

21.7 THIRD-PARTY ASSUMPTIONS

PRC has pursued build-own-operate (BOO) arrangements for most of the Blawn Mountain Project's utility requirements as well as identifying funding opportunities through established government programs for certain infrastructure requirements. Under these arrangements, BOO service providers will initially finance the construction and ultimately own, operate and maintain certain assets.

This PFS assumes a BOO scenario for the natural gas transmission line, water supply system and the sulfuric acid plant. Additionally, current state and federal funding programs are available to assist with the upgrades to the County access roads and construction of the electrical transmission line. While mitigating PRC's capital obligations, these BOO arrangements assign experienced operators who are responsible for managing and operating these utility and infrastructure assets.

As noted earlier, it is assumed that a contractor will perform the mining. The terms include a return on capital and a markup on operating expenses. Annual costs for the assumption have been included in operating expenses.

Table 21.3 summarizes the BOO arrangements or, through indirect government funding. PRC has received indicative offers from various parties with respect to the majority of these support assets. These capital costs will not be incurred by PRC.

Table 21.3 Third-Party Project Capital (US$M)

	Third-Party Equipment and Infrastructure
Access Road Upgrades	$10.2
Power Transmission Line	$28.5
Natural Gas Transmission Line	$40.0
Acid Plant	$125.0
Water Supply System	$24.4
Initial Mine Capital	$14.7
Total	**$242.9**

21.7.1 Access Road Upgrades

The BLM has issued a right-of-way (ROA) to upgrade the existing county roads for improved access, including truck traffic. Beaver County, to whom the ROA was issued, holds bonding authority that can be used to fund the road upgrades and it is expected that this approach can be utilized. Should any considerations need to be addressed, PRC believes that arrangements with the county can be made so that the county can exercise its bonding authority in support of the road improvement. The county has bonded for other road improvements in the area in support of new projects and economic development. Included in operating costs for the project is an amount required to amortize the county bond payment over the life of the project at 3% interest.

21.7.2 Power Transmission Line

As with the access road upgrades, PRC believes that arrangements can be made with government entities to fund construction of the powerline. The operating costs include a bond payment over the life of the project at an infrastructure market interest rate.

21.7.3 Natural Gas Transmission Line

As discussed previously, suppliers have been contacted regarding natural gas supply to the Blawn Mountain Project. During the original PFS, PRC was given budgetary capital estimates from the gas suppliers to supply the project with 100,000M cubic feet per day (Mcf/d). Given that the project now requires roughly 26,000 Mcf/d, the original estimates have been scaled accordingly. The natural gas line capital cost is estimated at $40.0M, which includes both permitting and construction costs. The annual cost of this natural gas facility charge is included in operating expenses.

21.7.4 Sulfuric Acid Plant

Approximately 1,600tpd of sulfuric acid (H_2SO_4) is manufactured at the project site from sulfur dioxide (SO_2) produced by the decomposition of alunite during the thermal processing of ore over the life of the project. PRC has received indicative terms for third-party ownership and operation of the sulfuric acid plant as well as the marketing of a portion of the project's sulfuric acid. The terms include a return on capital and a markup on operating expenses. An annual cost of this arrangement has been included in operating expenses.

21.7.5 Water Supply

Water necessary for the project will be sourced primarily from a wellfield located roughly seven miles to the north-east of the project and piped to the processing plant. The pipeline, source-wells and associated infrastructure will reside on SITLA lands. PRC has received an indicative offer from a third-party to own, operate and maintain the wellfield, water pipeline, surge pond, storage tanks and storage and reclaim water. The terms include a return on capital and a markup on operating expenses. An annual cost of this arrangement has been included in operating expenses.

21.7.6 Mining Operation Capital

The equipment needs for the mining operation was developed from the annual volume requirements (ore and waste) of the mine-plan described earlier in Section 16. Capital cost estimates for the major mining equipment is based on the 2015 Western Mine and Mill Cost Estimating guide as well as equipment vendor information. These costs are summarized below in Table 21.4. A contingency of 10% was used given that mining equipment cost are generally subject to closer estimation.

Table 21.4 Initial Capital Cost Estimate for Major Mining Equipment (US$M)

Primary Equipment		Qty	Unit Cost	Total Initial Capital
Front-End-Loader	16yd^3	1	$2.2	$2.2
End Dump Trucks	100t	4	$1.2	$4.7
Support Equipment				
Water Truck	12,000gal	1	$0.8	$0.8
Grader	297Hp	1	$0.8	$0.8
Track Dozer	580Hp	1	$1.2	$1.2
Track Dozer	410Hp	1	$1.0	$1.0
Drill	45,000lb	1	$0.7	$0.7
Other Support Equipment				$2.0
Contingency (10%)				$1.3
Total Initial Mine Capital				**$14.7**

21.8 OPERATING COSTS

Table 21.5 below presents the average annual operating costs for the processing plant and mining operation in 2016 constant dollars.

Table 21.5 Average Annual Plant and Mine Direct Operating Costs (US$)

Direct Plant and Mine Cash Production Cost	Annual Average *Cost($) / Ton SOP	Life of Plant Annual* Average (000)*
SOP Tons Sold		230
Sulfuric Acid Tons Sold		524
Mining (Contract Mine Operator Cost)	**$45**	**$10,449**
Processing		
Labor	$57	$13,131
Crushing & Grinding	$22	$4,997
Drying & Calcination	$124	$28,513
Acid Plant (Third-Party)	$42	$9,690
Leaching & Crystallization	$12	$2,755
Drying and Compaction	$2	$415
Steam Plant	$23	$5,381
Water Supply (Third-Party Operator Cost)	$6	$1,487
Other		
Tailings, Pumping, Etc.	$8	$1,862
Access Road & Power Line (Third-Party)	$5	$1,162
Product Handling & Transportation (Third-Party)	$41	$9,499
Credit for Value of Acid	($251)	($57,842)
Total Direct Operating Cost (Mining and Processing)	**$137**	**$31,498**

*Rounded

Saleable products of SOP range from 132,400t to 277,000t, averaging 230,200t over the life of the project. Sulfuric acid ranges from 288,400t to 626,000t, averaging 523,900t over the life of the project. Labor costs were developed from updated manpower estimates and utilizing regional data and industry experience. SNC provided estimates of manpower requirements for the processing plant while Millcreek developed requirements for the mining operation.

Average personnel for the processing plant and mining operation varies throughout the project life. The processing plant averages 148 while the mining operation averages 56. Total headcount for the entire operation varies 176 to 245 over the life of the project.

Plant operating costs, except for the third-party portions, were developed by MMG, RDI and PRC. Costs for the third-party portions (acid plant, water supply, etc.) were provided by PRC and are based on third-party contract operation of those facilities.

Operating costs for the contract mining operation were developed by Millcreek and are based on Millcreek's experience with similar-sized operations, information from the 2015 Western Mine and Mill Cost Estimating Guide and information received from local mining contractors.

Quantities of consumables such as power, water and natural gas were developed by SNC or provided by PRC. Unit costs for consumables are based on current pricing information and were developed by Millcreek and PRC. Pricing for natural gas is $2.84 per Mcf delivered to the site. Electricity rates are assumed at $0.038 per kilowatt hour and are based on tariff rates provided by the local utility. Costs for the water supply and acid plants are based on third-party indicative offers to own and operate those facilities.

Operating costs were also developed for the BOO / third-party arrangements described earlier. Table 21.6 presents a summary of these annual costs, averaged over the life of the project using 2016 dollars.

Table 21.6 Third-Party Utility, Infrastructure and Mine Operating Costs

Utilities, Infrastructure and Mining	Third-Party Average Annual Costs (US$M)
Access Road and Power transmission line	$1.2
Water, Acid Plant and Natural Gas Line BOO Arrangements	$16.0
Average annual contract mining cost	$10.4
Total	**$27.6**

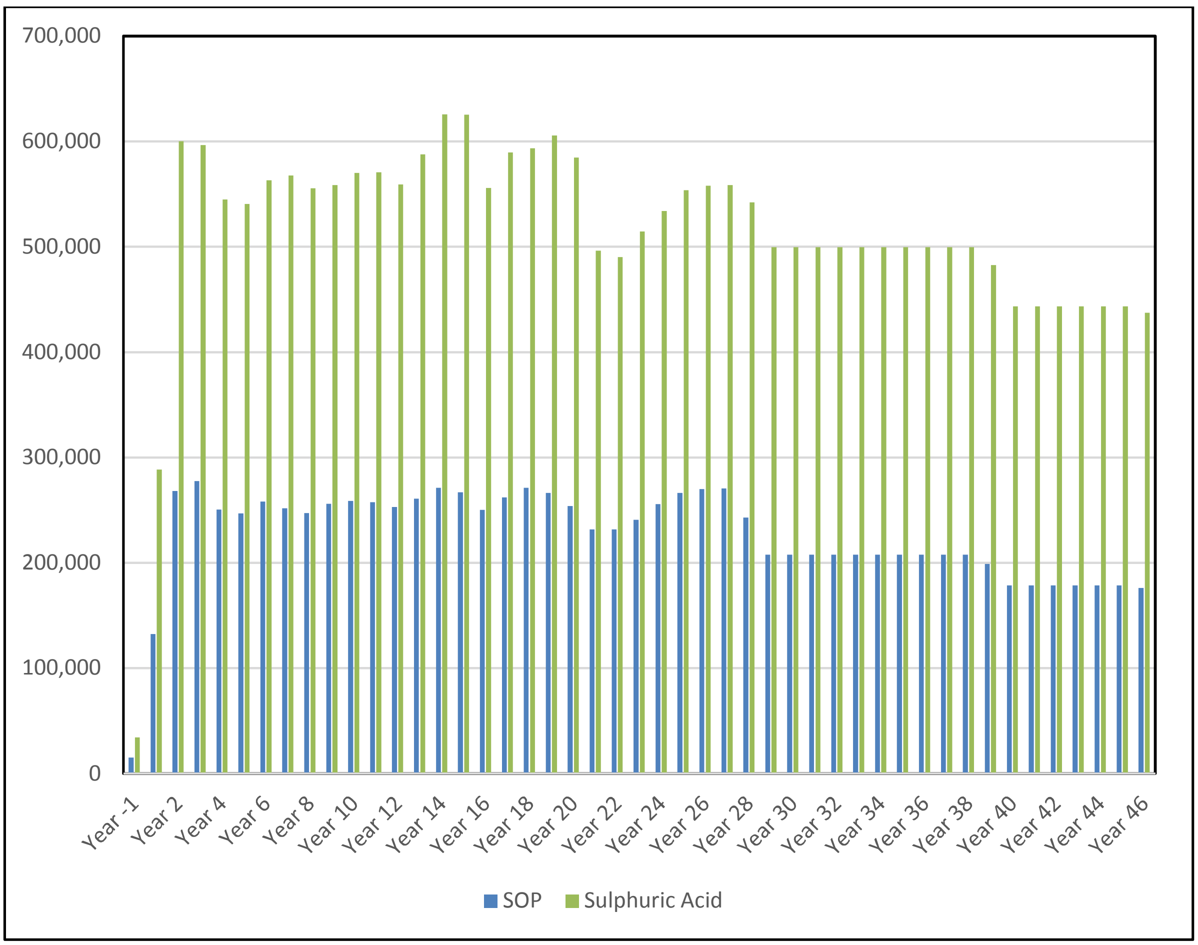
22 ECONOMIC ANALYSIS

22.1 PRINCIPAL ASSUMPTIONS

22.1.1 Saleable Product production and Schedule

Saleable product production averages roughly 230,000t of SOP over the 46 year life of the project, ranging from 132,400t to 277,000t. Sulfuric acid production ranges from 288,400t to 626,000t, averaging approximately 524,000t. The annual average ROM ore requirements are 3.4Mtpy after a short ramp-up period. Over the 46 year project life, there are 10.6Mt of SOP and 24.1Mt of sulfuric acid produced. Production volumes are shown below in Chart 22.1.

Chart 22.1 Production Volumes (SOP and Acid Tons)



22.1.2 Product Pricing and Transportation

Product sales prices are discussed earlier in Section 19, and were used in developing the cash flows for the project. Table 22.1 shows the average selling prices and annual revenues. As noted in Section 19, the selling prices below are at the plant gate. All prices are stated in constant 2016 dollars.

Table 22.1 Pricing Summary US$

Pricing and Transportation Costs	Unit	Life of Plant Annual Average $M
SOP Tons Sold	230,165	
Sulfuric Acid Tons Sold	523,933	
Average SOP Selling Price – FOB Plant	$675/t	$155
Average Sulfuric Acid Selling Price – FOB Plant	$115/t	$60
Average annual revenue		**$216**

22.1.3 Cash Production Costs

Direct cash production costs were discussed above in Section 21. Additional cash costs including site G&A expenses, property taxes, third-party costs, corporate overhead and royalties were developed by Millcreek. Royalties are based on lease agreements which provide for a royalty of 5% and 4% of selling price for SOP and sulfuric acid respectively. Property taxes are based on current regulations from Beaver County, Utah. The basis for third-party operating costs was discussed in Section 21. Total cash production costs are shown in Table 22.2 below. All costs are stated in constant 2016 dollars with no inflation.

Table 22.2 Total Cash Production Summary

Total Cash Production Costs	Annual Average Cost ($)/Ton SOP	Life of Plant Annual Average (000)
SOP Tons Sold		230
Sulfuric Acid Tons Sold		524
Direct Plant and Mine Cash Production Cost	**$388**	**89,212**
Credit for Value of Acid	($251)	($57,842)
Subtotal	**$137**	**$31,498**
Royalties	$44	$10,178
Site G&A	$20	$4,508
Property Taxes	$15	$3,347
Total Cash Production Cost	**$216**	**$49,603***

*Rounded

22.1.4 Income Taxes

US Federal income tax and Utah State income tax were applied at 35% and 5% respectively to estimated taxable income including the effect of the Alternative Minimum Tax provisions. Tax depreciation is calculated based on US Federal tax regulations. Percentage depletion is taken as a deduction in computing taxable income. PRC provided Millcreek with an opinion from a tax expert on the application of the percentage depletion to this project.

22.2 CASH FLOW

Cash flow from the project is summarized in Table 22.3 in 2016 constant dollars and inflated dollars.

Table 22.3 Total Cash Production Summary

Project Cash Flow Summary	Life of Plant Annual Average (2016$M)	Life of Plant Annual Average (Inflated $M)
SOP Tons Sold	230	230
Sulfuric Acid Tons Sold	524	524
SOP revenue FOB - Plant	**$155**	**$244**
Acid revenue FOB - Plant	**$60**	**$96**
Total revenue FOB - Plant	**$216**	**$340**
Direct Plant and Mine Cash Production Costs	$89	$132
Royalties	$10	$16
Site G&A	$5	$13
Property Taxes	$3	$6
Total Cash Production Costs	$107	$167
Operating Margin	$108	$172
Income Taxes	$25	$44
Cash Flow from Operations	**$83**	**$128**

Pre-production cash outflows total $458 million over the three year project execution and construction period. Cash flow turns positive during year 1. Payback occurs during year 5 which is approximately 8 years after the initial investment. Cash flow after payback averages $81M per year for a total net cash flow of $3.3 billion over the life of the project. Annual and cumulative cash flows are shown in Chart 22.2. Summaries of cash flow for the project are presented in Tables 22.4 and 22.5. Table 22.4 presents cash flows using an inflation rate of 2.0% beginning in year 2 (2021) and table 22.5 shows cash flows in 2016 constant dollars.

Chart 22.2 Annual and Cumulative Cash Flow

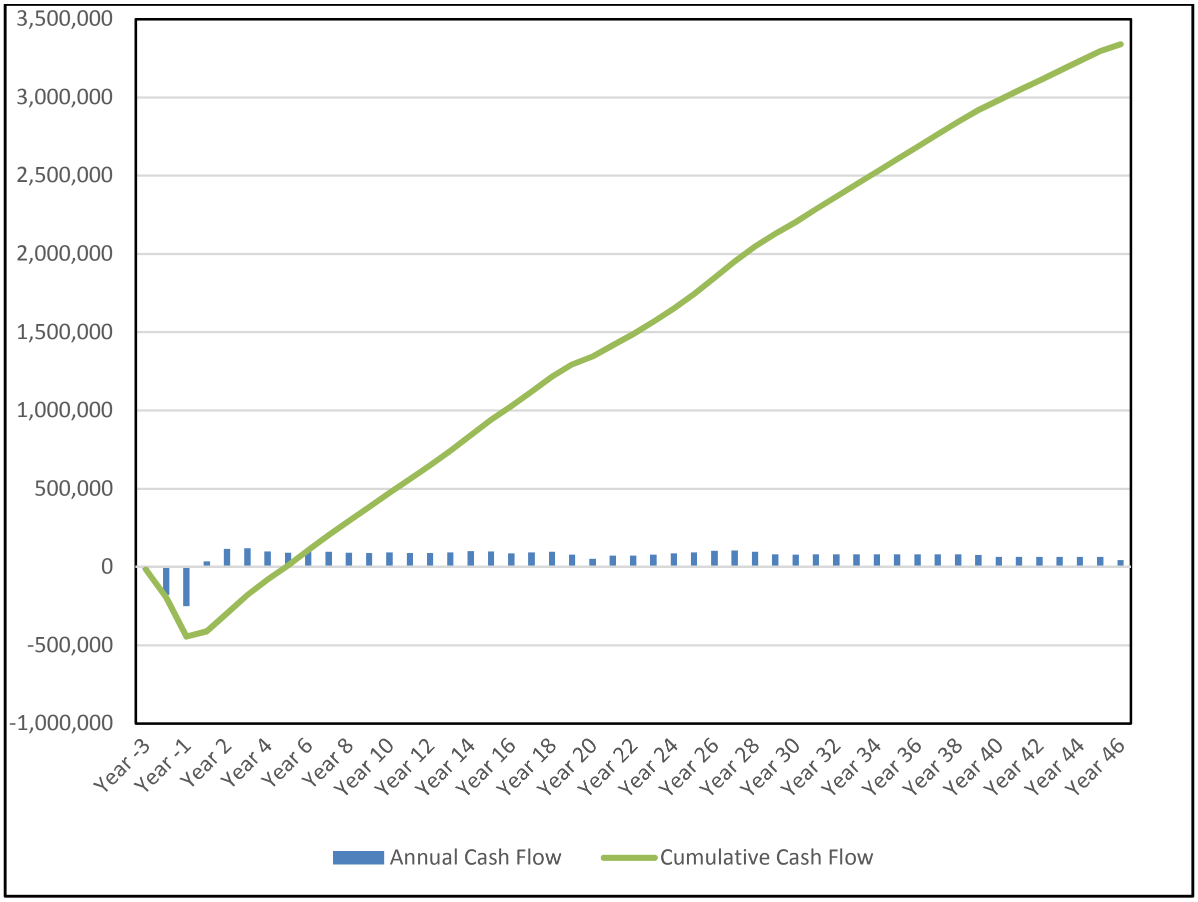


Table 22.4 Project Cash Flow (Infated Dollars)

	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6-10	Y11-15	Y16-20	Y21-30	Y31-40	Y41-46	Total
	Total	Total	Total	Total	Total	Total	Total	Total	Avg.	Avg.	Avg.	Avg.	Avg.	Avg.	
SOP Tons Sold	0	0	15	132	268	277	251	247	254	262	261	242	204	178	10,603
Sulfuric Acid Tons Sold	0	0	34	288	600	596	545	541	563	594	586	525	492	442	24,135
Net SOP revenue FOB Plant ($)	**0**	**0**	**0**	**89**	**185**	**195**	**179**	**180**	**197**	**224**	**246**	**266**	**272**	**279**	**11,223**
Net acid revenue FOB Plant ($)	**0**	**0**	**0**	**33**	**70**	**71**	**66**	**67**	**74**	**87**	**94**	**98**	**112**	**118**	**4,397**
Total Revenue ($)	**0**	**0**	**0**	**123**	**255**	**266**	**246**	**248**	**272**	**311**	**341**	**364**	**385**	**397**	**15,621**
Direct Plant and Mine Cash Production Cost	0	0	0	73	106	111	115	118	122	135	148	150	151	190	6,698
Royalties	0	0	0	6	12	13	12	12	13	15	16	17	18	19	737
Property Taxes	0	0	0	0	3	3	2	2	4	5	5	7	7	6	256
Total Cash Production Cost	**0**	**0**	**0**	**79**	**121**	**127**	**128**	**132**	**139**	**155**	**169**	**174**	**176**	**215**	**7,691**
Operating Margin	**0**	**0**	**0**	**43**	**134**	**140**	**118**	**116**	**133**	**156**	**171**	**190**	**208**	**182**	**7,930**
Income Taxes	0	0	0	0	12	18	15	18	28	39	45	52	58	48	2,001
Cash Flow from Operations	0	0	0	43	122	122	102	98	105	118	127	138	150	134	5,929
Capital Expenditures	**15**	**180**	**250**	**8**	**5**	**0**	**0**	**0**	**0**	**1**	**21**	**1**	**0**	**-1**	**570**
Net Cash Flow	**-15**	**-180**	**-250**	**35**	**117**	**122**	**102**	**98**	**105**	**117**	**106**	**138**	**150**	**135**	**5,359**

Table 22.5 Project Cash Flow (2016 Constant Dollars)

	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6-10	Y11-15	Y16-20	Y21-30	Y31-40	Y41-46	Total
	Total	Total	Total	Total	Total	Total	Total	Total	Avg.	Avg.	Avg.	Avg.	Avg.	Avg.	
SOP Tons Sold	0	0	15	132	268	277	251	247	254	262	261	242	204	178	10,603
Sulfuric Acid Tons Sold	0	0	34	288	600	596	545	541	563	594	586	525	492	442	24,135
Net SOP revenue FOB Plant ($)	**0**	**0**	**0**	**89**	**181**	**187**	**169**	**167**	**172**	**177**	**176**	**164**	**138**	**120**	**7,147**
Net acid revenue FOB Plant ($)	**0**	**0**	**0**	**33**	**69**	**69**	**3.**	**62**	**65**	**68**	**67**	**60**	**57**	**51**	**2,772**
Total Revenue ($)	**0**	**0**	**0**	**123**	**250**	**256**	**232**	**229**	**236**	**245**	**243**	**224**	**194**	**171**	**9,918**
Direct Plant and Mine Cash Production Cost	0	0	0	73	104	107	108	109	106	106	106	93	76	82	4,272
Royalties	0	0	0	6	12	12	11	11	11	12	11	11	9	8	468
Property Taxes	0	0	0	0	3	2	2	2	3	4	4	4	4	3	154
Total Cash Production Cost	**0**	**0**	**0**	**79**	**118**	**122**	**121**	**122**	**120**	**122**	**121**	**107**	**89**	**92**	**4,894**
Operating Margin	**0**	**0**	**0**	**43**	**132**	**134**	**111**	**107**	**116**	**123**	**122**	**116**	**105**	**79**	**5,024**
Income Taxes	0	0	0	0	12	17	14	16	23	29	31	30	28	20	1,173
Cash Flow from Operations	0	0	0	43	120	118	97	91	93	94	92	86	78	59	3,851
Capital Expenditures	**15**	**180**	**250**	**8**	**5**	**0**	**0**	**0**	**0**	**1**	**11**	**0**	**0**	**-1**	**512**
Net Cash Flow	**-15**	**-180**	**-250**	**35**	**115**	**118**	**97**	**91**	**93**	**93**	**81**	**86**	**78**	**59**	**3,339**

22.3 FINANCIAL ANALYSIS

Utilizing an inflation rate of 2.0% beginning in year 2 (2021), the after tax, inflated dollar, internal rate of return for the project is 20.1%. After tax net present values at 8%, 10%, and 12% are shown in Table 22.6.

Table 22.6 Net Present Value Results

Discount Rate	8%	10%	12%
After Tax Net Present Values	$728 million	$482 million	$315 million

The after tax, 2016 constant dollar, internal rate of return for the project is 18.4%. After tax net present values at 8%, 10%, and 12% are shown in Table 22.7.

Table 22.7 Net Present Value Results

Discount Rate	8%	10%	12%
After Tax Net Present Values	$519 million	$341 million	$216 million

The accuracy of resource and reserve estimates is, in part, a function of the quality and quantity of available data and of engineering and geological interpretation and judgement. Given the data available at the time this report was prepared, the estimates presented herein are considered reasonable. However, they should be accepted with the understanding that additional data and analysis available subsequent to the date of the estimates may necessitate revision. These revisions may be material. There is no guarantee that all or any part of the estimated resources or reserves will be recoverable.

22.4 SENSITIVITY ANALYSIS

Tables 22.8 and 22.9 present the sensitivity of the project economics to changes in selling price, direct operating costs, and capital costs and natural gas pricing. Table 22.8 presents results utilizing inflated dollars and Table 22.9 uses constant dollars.

Table 22.8 Sensitivities (inflated dollars)

Discount Rate	8%	10%	12%
Base Case	$728 Million	$482 Million	$315 Million
10% Increase in Revenue	$884 Million	$603 Million	$412 Million
10% Decrease in Revenue	$563 Million	$353 Million	$212 Million
10% Increase in SOP Selling Price	$860 Million	$584 Million	$397 Million
10% Decrease in SOP Selling Price	$595 Million	$378 Million	$232 Million
10% Increase in Acid Price	$781 Million	$523 Million	$348 Million
10% Decrease in Acid Price	$676 Million	$441 Million	$283 Million
10% Increase in Operating Costs	$650 Million	$421 Million	$266 Million
10% Decrease in Operating Costs	$805 Million	$542 Million	$364 Million
10% Increase in Natural Gas Price	$710 Million	$468 Million	$304 Million
10% Decrease in Natural Gas Price	$747 Million	$496 Million	$327 Million
10% Increase in Capital Costs	$685 Million	$441 Million	$275 Million
10% Decrease in Capital Costs	$771 Million	$523 Million	$355 Million

Table 22.9 Sensitivities (2016 Dollars)

Discount Rate	8%	10%	12%
Base Case	$519 Million	$341 Million	$216 Million
10% Increase in Revenue	$647 Million	$443 Million	$300 Million
10% Decrease in Revenue	$382 Million	$231 Million	$126 Million
10% Increase in SOP Selling Price	$627 Million	$427 Million	$287 Million
10% Decrease in SOP Selling Price	$409 Million	$253 Million	$144 Million
10% Increase in Acid Price	$562 Million	$375 Million	$244 Million
10% Decrease in Acid Price	$476 Million	$307 Million	$188 Million
10% Increase in Operating Costs	$454 Million	$288 Million	$173 Million
10% Decrease in Operating Costs	$583 Million	$392 Million	$259 Million
10% Increase in Natural Gas Price	$504 Million	$329 Million	$207 Million
10% Decrease in Natural Gas Price	$534 Million	$353 Million	$226 Million
10% Increase in Capital Costs	$477 Million	$300 Million	$177 Million
10% Decrease in Capital Costs	$561 Million	$382 Million	$256 Million

The project economics are more sensitive to the selling price of SOP than changes in capital or operating costs.

23 ADJACENT PROPERTIES

There is no data or information available for adjacent properties that are pertinent to this report.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT SCHEDULE

PRC intends to secure a fixed-price engineering, design and construction (EPC) contract in 2017 to design and build the mine and processing facilities for the Blawn Mountain Project. The Blawn Mountain Project schedule has been prepared in order to meet commissioning and production target dates in late 2019 / early 2020. A summary of this schedule is included below and is represented on Table 24.1.

- Exploration drilling of Areas 1 and 2 in support of the prefeasibility study was completed in August 2013. Additional delineation drilling will likely be completed in the summer of 2017 in two areas proximal to Area 2, discussed further in Section 26.3.
- Project financing includes timing to cover five major categories for financing through production. Identified long lead items will be procured starting in the first half of 2017 and continuing through the second quarter of 2018.
- With the exception of an Air Approval Order application which requires final design emission calculations, major environmental permits (mining and water discharge permits) have been obtained for the project. The remaining permits will be applied for in sufficient time to meet project requirements. Environmental permitting will span into fourth quarter 2017. Major operating permits and those required to start civil construction are anticipated to be in hand during the first quarter of 2018.
- EPC work begins in begins in late 2017 and will conclude by mid-2019.
- Third-party design/build utilities include power and gas. Engineering and environmental studies will start the first quarter of 2017. Construction of these utilities will be in the final stages by the end of 2018. These activities will be dependent on early financing structured to ensure that property alignments, environmental studies and plans of development can be executed to meet the plant commissioning portion of the schedule.
- Construction of the access road to the project area will be begin in the fourth quarter of 2017 and should be completed by mid-2018.
- Mine development and construction will begin in late 2018. Ore will be stockpiled until the processing plant is online.
- Plant commissioning quality control and construction support will end in the first quarter of 2020. Commissioning will begin in the third quarter 2019. Once complete, stockpiled ore will be processed until full production. Full production will start in the fourth quarter of 2020.

Table 24.1 Schedule Summary

Major Tasks	2017				2018				2019				2020			
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Confirmation Drilling		■	■													
Financing	■	■	■	■	■	■										
Permitting																
Environmental Permits	■	■	■	■												
Construction Permits	■	■	■	■	■											
Engineering, Procurement and Construction																
Third-Party Design Build Packages																
Power Line	■	■	■	■	■	■	■	■								
Gas Line	■	■	■	■	■	■	■	■								
Construction																
Access Road				■	■	■										
Civil					■	■	■	■	■							
Processing Plant (includes acid and calcine plant)					■	■	■	■	■	■	■	■				
Production																
Mining									■	■	■	■	■	■	■	■
Plant Commissioning											■	■	■			
Production													■	■	■	■

25 INTERPRETATION AND CONCLUSIONS

25.1 MINING AND RESOURCE

PRC controls the mineral tracts and Exploration/Option Agreement through SITLA. The property has undergone exploration which can be considered sufficient for the delineation of mineral resources in Area 1 and Area 2. Other areas (3 and 4) under PRC control are considered exploration targets. The drilling and surface mapping within Area 1 and Area 2 has led to a geologic interpretation of the deposit as rhyolite porphyries and ignimbrites that have experienced hydrothermal alteration and consequent enrichment in potassium and aluminum compounds, and termed alunite.

Mineral resources have been estimated and reported at a 1% K_2O cut-off grade for Area 1 and Area 2. Measured plus indicated in situ resources for Area 1 are 164.8Mt and 398.3Mt for Area 2.

Utilizing the resource estimate as defined in this report, Millcreek developed and examined several mine-plan options based on various parameters and ore cut-off grades. The mine plan was used to develop capital and operating costs for this PFS project. A cash flow was generated that formed the basis of a NPV and IRR calculation that confirmed the economic viability of the mine-plan, hence allowing Millcreek to define reserves.

Proven and probable reserves for this project are 153.3Mt with an average, combined cut-off grade of 3,90% K_2O. Over the life of the project, 10.6Mt of SOP and 24.1Mt of sulfuric acid are produced.

The accuracy of resource and reserve estimates is, in part, a function of the quality and quantity of available data and of engineering and geological interpretation and judgment. Given the data available at the time this report was prepared, the estimates presented herein are considered reasonable. However, they should be accepted with the understanding that additional data and analysis available subsequent to the date of the estimates may necessitate revision. These revisions may be material. There is no guarantee that all or any part of the estimated resources or reserves will be recoverable.

25.2 MINERAL PROCESSING AND METALLURGICAL TESTING

RDI reviewed a number of technical reports and results of previous metallurgical testing commissioned by PRC and performed at HRI during the 2011-2013 period as well as input from desktop simulations of unit operations by equipment manufacturers. The metallurgical tests examined comminution, beneficiation, flotation, calcination, leaching, crystallization, material handling and solid/liquid separation areas of the envisioned processing facilities.

The Blawn Mountain Project's exploration drilling program supplied the bulk samples and composites of drill core and rotary drill cuttings for the experiments. Additionally, historical test results from bench-scale and pilot plant experiments commissioned by ESI during the 1970s and performed at HRI were also critically reviewed. The conclusions are as follows:

The production to the process plant will be approximately 3.4Mtpy ROM ore (dry solids). The process plant is designed with a throughput of 425tph. The process plant will include:

- Whole ore feed, with ore blending, as required;
- Grinding circuit will generate a plant feed of 1000μm (1.2mm maximum), P_{80};
- Plant feed to gas suspension calciner;
- Processed ore to water leach for SOP recovery;
- Filtrate recycling to leach for SOP concentration build-up;
- MUR/crystallizer will process brine to extract SOP;
- Treated and packaged SOP product for markets;
- Roaster off-gases (SO_2) as feedstock to sulfuric acid plant;
- Washed tails will be stored as Al_2O_3 resource;
- Source water will come from groundwater wells at a rate between 800 to 1,000gpm;
- Water will be conserved through extensive re-use of effluents.

It is anticipated that the quality specifications for the standard SOP product are:

- Purity: 50wt% K_2O (92.5wt% K_2SO_4);
- Particle-size: 70 to 10 Tyler mesh;
- Chloride content: < 1.0%.

The by-product of the process includes sulfuric acid and alumina. The sulfuric acid will be produced at a production rate of approximately 1,600tpd. The product purity will be approximately 93% H_2SO_4. The alumina (Al_2O_3) will exist in the tails residue. This material will be stored in the tailings pond and may be able to be processed at a later date as part of a gamma phase alumina amenable to caustic leach.

25.3 PROJECT RISKS

Other than those noted elsewhere in this report regarding resource and reserve estimates, there are no further significant risks or uncertainties that affect the reliability or confidence in the exploration information, resource estimates or reserve estimates.

25.4 CONCLUSION

Based on the results of the PFS, Millcreek and RDI have reached the following conclusions:

- There are sufficient mineable tons of ore at an average grade of 3.90% K_2O to produce an approximate average of 230,000 tons of SOP over a 46 year project life;
- No fatal flaws have been identified at this stage of project development;
- Pre-production capital costs estimated at $458M along with several third-party build, own, operate arrangements will be required to bring this project into production;
- Cash costs of production per ton of SOP, after sulfuric acid credits and before royalties, is estimated at $137;
- Based on the assumptions presented in this report, the project will generate positive cash flows and achieve an after tax, 2016 constant dollar internal rate of return of 18.4%. Using an inflation rate of 2.0% beginning in year 2, the after tax, inflated dollar internal rate of return is 20.1%.

The overall conclusion is that the results of this study indicate positive economic results and the project should be continued to the next phase of development.

26 RECOMMENDATIONS

26.1 MINERAL PROCESSING AND METALLURGICAL TESTING

Recommendations on additional metallurgical test work and trade-off studies required for optimized flow sheet development and process plant design are as follows:

- Undertake metallurgical test work to investigate the potential for creating a high quality alumina product from the leach residue through physical separation, reverse flotation or some combination of these other processes.
- Perform mineralogical studies using such technology as QEMSCAN, an automated mineralogy and petrography system, to identify and delineate texture, grain sizes and mineralogical associations in the ore from different parts of the mine, calcine from the roasters and leach residue, which have a direct bearing on product grade and recovery.
- Conduct tests on dry ROM ore samples to verify the choice of crushing and grinding equipment for producing feed at P_{80} 1000µm for the calciner/roaster circuit and develop quantitative data on liberation of alunite as a function of grain size screen fractions from different areas of the mine.
- Evaluate the results of slurry rheology, sedimentation and filtration tests to establish type and dosage of flocculant, if required, and to select thickeners and filtration equipment to reduce the moisture content of the feed to the pyroprocessing steps.
- Perform calcining/roasting tests to determine the operating parameters and trade-off studies to assist in equipment selection, determination of energy requirements and composition of SO_2-bearing off-gases for recovery of sulfuric acid as by-product.
- Identify the phases (potassium sulfate, crystalline alumina, and residual alunite) in the respective calcines produced at **a range of temperatures** by XRD and microscopic examination.
- Perform agitated tank water-leach studies to determine operating parameters such as pulp density, residence time, temperature, intensity of agitation, as well as to identify the phases (potassium sulfate, crystalline alumina, and residual alunite) in the water-leach residues of calcines produced at **a range of temperatures** by XRD and microscopic examination.
- Conduct pilot plant tests on evaporation and crystallization of SOP product from the brine to determine the operating parameters and trade-off studies to assist in equipment selection, product quality, bleed requirements for impurity control, size of crystals formed and compaction of product and handling requirements.

26.2 RECOVERY METHODS

The following is a list of comments and recommendations for the PFS of Area 100 through 1000, excluding Area 900 and 1400. It should not be considered a complete list, but as a partial list of comments and recommendations to be considered should a feasibility study be undertaken at the beginning of the feasibility study:

26.2.1 Areas 100-200

- Validate assumptions of minus 15in. ROM ore to primary crusher and size gradation;
- Validate screening and secondary crushing sizing and design;
- Upon completion of testing, validate the need for a pebble crusher circuit.

26.2.2 Area 300

- Care should be taken to minimizing gas volume and limit O_2<2%;
- Engage vendors to provide pilot plant-scale tests of drying/calcining/roasting crushed and ground ROM ore with P_{80} = 1000μm (1.0mm) and not > 1.2 in size range should be completed to validate the thermal processing concept and to validate the estimated concentration of SO_2 in the off-gases as feedstock to the sulfuric acid plant.

26.2.3 Area 400

- Engage vendors to provide pilot plant testing to validate the feed stock composition including percent volumes of SO_2, O_2, CO_2, N_2, H_2O and SO_3;
- Consider consolidating plants into one large plant.

26.2.4 Area 500

- Complete testing to validate tailings thickener and belt filter capacity and performance;
- Optimize water and energy conservation measures.

26.2.5 Area 600

Complete testing of brine to validate crystallization equipment capacity and performance.

26.2.6 Area 700

At the completion of testing, verify SOP characteristics, percentage of fines and adjust equipment and flow streams accordingly.

26.2.7 Area 800

A final gas consumption and pressure analysis must be completed for the sizing of the gas line. A gas load of 100M standard cf/day was assumed at the beginning of the prefeasibility study to establish a base line for comparison between Kern River and Questar Gas.

26.2.8 Tailings

- Verify the quality of tailings for any potential contaminants. May consider oil & grease and other potential constituents of concern;
- Further evaluate tailings discharge handling methods;
- Geotechnical evaluation of foundation characteristics of the tailings area for tailings stability;
- Determine geology and permeability of materials underlying the tailings to assess potential for seepage from the site.

26.2.9 Auxiliary Services

- Provide detailed chemical analysis of water harvested from the wellfield, including a mass balance of cations and anions;
- Establish limits on the total dissolved solids (TDS) content and/or the concentration of residual reagents in water recycled to the processing plant;
- Provide recommendations on a method of recovering SOP values from the bleed stream established for controlling formation of glaserite;
- Determine infiltration rate and soil type of leach field area. Work with state regulators for approval as a sewage treatment option.

26.3 CONTINUED EXPLORATION AND RESOURCE DELINEATION

Exploration has identified significant measured and indicated resources beyond what has been allocated for reserves on Area 1 and Area 2. There are also other target areas within the Blawn Mountain Project that have the potential of hosting additional alunite mineralization. As the project advances towards development and into production, exploration should continue to evaluate resource potential of other targets within the project area including the following four targets:

26.3.1 Target 1

Extending resource limits southeastward from the central portion of Area 2 with additional drilling. The geologic model and field observations suggest mineralization may extend farther to the southeast than currently defined and may positively impact development of Area 2.

26.3.2 Target 2

Two rhyolitic eruptive centers occur west-southwest of Area 2 that are coincident with a lineament that projects into the main zone of mineralization in Area 1. Though historic mapping does not identify this lineament as a fault, the geometry of the two deposits suggests the lineament may be a source conduit for sulfide fluids and hydrothermal alteration. Outcrops on the two volcanic domes exhibit alunite veining. This area should be tested with drilling. The proximity of this target could have a positive impact toward development of Area 2.

26.3.3 Target 3

ESI exploration efforts identified alunite mineralization on two areas referred to as Area 3 and 4. Limited drilling was carried out by ESI on both areas and Potash Ridge completed two holes on Area 4. Millcreek considers Area 3 as lower priority targets that won't directly impact the development presented in this study. Once the project is in production, an ongoing exploration program should be developed to investigate alunite mineralization on Area 3.

26.3.4 Target 4

A prominent fault projects northeastward from the north side of Area 1 out across the alluvial valley. The fault separates a small ridge from Area 1 proper, and three small hills protrude through the alluvial fill along the fault. The fault marks the contact between Devonian carbonate rocks and Miocene volcanic rocks that host the alunite mineralization. Jasperoid development (silicification) has also been observed along the contact. Outcrop sampling and alteration mapping leading to drill testing along the fault should be planned to identify potential new alunite resources. This target is also considered a lower priority target that should be investigated as part of an ongoing exploration program once PRC is in production.

Table 26.1 presents the budget for carrying out exploration on Targets 1 and 2. Figure 26.1 identifies the proposed exploration targets for the Blawn Mountain Project.

Table 26.1 Exploration Budget for 2017

Description	Unit Cost ($)	January - May Units	January - May Cost ($)	June-July Units	June-July Cost ($)	Total ($)
Target 1						
Drill Planning	7,500	1	7,500			7,500
Site Preparation	1,500	7	10,500			10,500
RC Drilling	28	2,800	78,400			78,400
Geologic Personnel	1,850	8	14,800			14,800
Analyses	39	280	10,864			10,864
Abandonment/ Reclamation	3,700	7	25,900			25,900
Subtotal			**147,964**			**147,964**
Target 2						
Drill Planning	7,500	1	7,500			7,500
Site Preparation	1,500			6	9,000	9,000
RC Drilling	28			1,200	33,600	33,600
HQ Core Drilling	50			600	30,000	30,000
Geologic Personnel	1,850			12	22,200	22,200
Analyses	39			180	6,984	6,984
Abandonment/ Reclamation	3,700			6	22,200	22,200
Subtotal			**7,500**		**123,984**	**131,484**
Total			**155,464**		**123,984**	**279,448**

26.4 ENVIRONMENTAL

As the project advances towards development, additional information is required to assist in permitting activities. To evaluate these needs within the project area the following information should be collected:

- Evaluate tailings geochemistry to determine the need for a liner under the tailings area;
- Determine the seepage rate of the material under the tailings area;
- Evaluate the chemistry of the tailings materials by testing water released from the tailings to ensure they are non-toxic and non-hazardous.

26.5 WATER

Further aquifer characterization is required before establishing wells in the southern portion of the Wah Wah Valley to produce water at the rate needed for the project.

26.6 MINING

Geotechnical investigations of the roads, pads, pit walls, waste rock piles, tailings materials and collection and settlement pond embankments need to be conducted.

26.7 INFRASTRUCTURE

The following studies should be considered as part of developing this project in future efforts. These reports will assist in providing better definition of the project scope and designs.

- Geotechnical: for foundation for the processing plant and other facilities including any rail lines;
- Product storage requirements, capacity, liner requirements and size distribution of SOP;
- Reclaim system: options to meet product requirements

27 REFERENCES

Abbott, J.T., Best, M.G., and Morris, H.T. (1983) Geologic map of the Pine Grove-Blawn Mountain area, Beaver County, Utah: US Geological Survey Map I-1479, two sheets, scale 1: 24,000.

Ackerman, J. B. (1978). Alunite Flotation, M.S. Thesis, Department of Metallurgy and Metallurgical Engineering, University of Utah, Salt Lake City, UT.

AGRC (2011) National Agricultural Imagery Program (NAIP), 1 meter resolution color orthophotography: http://gis.utah.gov/aerial-photography/2009-naip-1-meter-orthophoto-graphy, accessed June 2011.

ALS Metallurgy Kamloops. (June 5, 2013). Metallurgical Assessment of The Blawn Mountain Project, Utah, United States. Work Performed on behalf of Potash Ridge Corp. Kamloops, BC, Canada.

Anonymous. (February, 1989). The NG Alunite Property of Earth Sciences, Inc. Beaver County, Utah.

Berge, Dale (1974) Preliminary report on archeological investigations in southern Wah Wah Mountains, Beaver County Utah: Provo, Brigham Young University, unpublished report of the Museum of Archeology and Ethnology.

Bowen, F. J. et al. (July 7, 1972). Metallurgical Studies of Utah Alunite Ore, for Earth Sciences, Inc. Hazen Project 1132, Hazen Research, Inc. Golden, CO.

Bowen, F. J. et al. (April 12, 1973). Metallurgical Studies of Utah Alunite Ore, for Earth Sciences, Inc. Hazen Project 1235, Hazen Research, Inc. Golden, CO.

Chapman, E.P., Jr. (1974) Interim report for Earth Sciences, Inc., on the NG Alunite Deposit Wah Wah Mountains, Beaver County, Utah, USA.: Unpublished reserve report by Chapman, Wood, and Griswold, Ltd., 18 p.

Canadian Institute of Mining, Metallurgy and Petroleum, (2014), The CIM Definition Standards on Mineral Resources and Reserves (CIM Definition Standards).

Couzens, T.R. (1975) Alumet project mine-planning for NG Alunite Area C: Unpublished report for Earth Sciences, Inc. by Pincock, Allen & Holt, Inc., 118 p.

David Chinloy, Hatch Ltd., Mississauga, ON. (March 9, 2012). Potash Ridge Alunite Processing, Project Memo to Potash Ridge: Ross Phillips/Paul Hampton. H340277-0000-90-220-0001, Rev.0.

Earth Sciences, Inc. (1974) Alunite mine/processing plant complex, western Beaver County, Utah: Environmental Impact Assessment prepared by Earth Sciences, Inc. for the US Bureau of land Management, two volumes, 744 p.

Gillespie, Daniel W. (March 1, 2012). Comminution Testing. Letter report from to Ross Phillips of Potash Ridge Corp. Hazen Project 11468-01, Hazen Research, Inc. Golden, CO.

Green, Don W., and Perry, Robert H. (2008). Perry's Chemical Engineers' Handbook, 8th ed. McGraw-Hill.

Grzmil, B. U. and B. Kic. (2005). "Single-Stage Process for Manufacturing of Potassium Sulfate from Sodium Sulfate," in Chemical Papers, Institute of Chemistry, Slovak Academy of Sciences, Vol. 59(6b), pp. 476-480.

Hall, R.B. (1978) World Nonbauxite Aluminum Resources – Alunite: US Geological Survey Professional Paper 1076-A, p. 35.

Hall, R.B., and Bauer, C.W. (1983) Alunite, in LaFond, S.J., Industrial minerals and rocks, 5th edition: Littleton, Society for Mining, Metallurgy, and Exploration, Inc., p. 417-434.

Hanson, Richard C. (March 18, 1977). ***Crushing and Grinding of Alunite Ore,*** for Alumet Co.

Hatch Ltd., Mississauga, ON. (March 19, 2012). Potash Ridge: Alunite Processing Process Design Basis. H340277-0000-109-0001, Rev. B.

Hazen Research, Inc. (July 7, 1972). Metallurgical Studies of Utah Alunite Ore. HRI Project No. 1132.

Hazen Research, Inc. (June 24, 2013). Alunite Process development and Pilot Plant Program—Interim report. HRI Project No. 11687.

Hodgson, Rodney and Thomas, P. N. (April 12, 1972). ROASTING AND LEACHING OF ALUNITE ORE, Progress Report No. 1, for Earth Sciences, Inc. Hazen Project 1132, Hazen Research, Inc. Golden, CO.

Hofstra, A.H. (1984) Geology, Alteration and Genesis of the NG Alunite Area, Southern Wah Wah Range, Southwestern Utah: Golden, Colorado School of Mines M.S. Thesis, 130 p.

Hu, Yue-hua, Li Har-pu, and Jiang, Yu-ren (October 2002). "Effect of Hydroxamic Acid Starch on Reverse Flotation Desilicate from Diasporic Bauxite," Trans. Nonferrous Met. Soc. China, Vol. 12, No. 5, pp. 974-978.

Imhof, R., Battersby, M., Parra, F., and Sanchez-Pino, S. (June 6 - 9, 2005). "The Successful Application of Pneumatic Flotation Technology for the removal of Silica by Reverse Flotation at the Iron Ore Pellet plant of Compania Minera Huasco, Chile," Centenary of Flotation Symposium, Brisbane, Queensland, Australia.

Jennings, L. D., and Bisque, R. E. (November 13, 1970). Preliminary Tests to Evaluate the Extraction of Alumina and Potassium Sulfate from Alunite Ores, for Earth Sciences, Inc.

Kesler, G. H. (February 10, 1977). Chemical Thermodynamics of Alunite Processing. Report No. SR-77-1, Alumet Company, Golden, CO.

Krahulec, Ken (2007) Mineral potential of the Blawn Wash Alunite area, Beaver County, UT: Unpublished report prepared by the Utah Geological Survey for the Utah School and Institutional Trust Lands Administration under a Memorandum of Understanding, 29 p.

Lange, Norbert A., and Forker, Gordon M. (1967). Handbook of Chemistry, 10th ed. McGraw-Hill.

Liu, Wen-gang, Wei, De-zhou, Wang, Ben-ying, Fang, Ping, Wang, Xiao-hui, and Cui, Bao-yu (February 18, 2009) "A New Collector Used for Flotation of Oxide Minerals," Science Digest, Elsevier Science Press.

Loost, Kent W. (June 21, 1977). Recovery of Aluminum from Alunite Ore Using Acid-Leach to Purify the Residue for Bayer Leach, US Patent 4,031,182, filed March 24, 1976, and issued June 21, 1977.

MECS, Inc. (June 2009). MECS Acid Technology, Corporate brochure.

Mellon, Robert J. (May 21, 2012). Alunite Development Program, prepared for Utah Alunite LLC and Potash Ridge Corporation. Hazen Project 11468, Hazen Research, Inc. Golden, CO.

Mellon, Robert, Hazen Research, Inc. (March 28, 2013), Comminution Testing: Hazen Project 11687, Report and Appendices.

Mellon, Robert, Hazen Research, Inc. (2013). Periodic Updates on Bench-scale and Pilot Plant Test Results and Personal Communication.

Miller, G.M. (1966) Structure and stratigraphy of southern part of Wah Wah Mountains, southwest Utah: Bulletin of the American Association of Petroleum Geologists, v. 50, p. 858-900.

Myertons, Ralph (May 27, 1976). Flotation of Alunite and Alumina, prepared for Alumet Company, Golden, CO.

NorFalco. (April 2007). Sulfuric Acid Handbook.

Norwest Corporation (2012) Technical Report, Blawn Mountain Project, Beaver County, Utah: CIM Technical Report for Potash Ridge Corporation, effective date April 16, 2012, Published on SEDAR.com.

Norwest Corporation. (November 5, 2012). Technical report: Preliminary Economic Assessment Blawn Mountain Project, Beaver County, Utah. Submitted to Potash Ridge Corporation effective date October 30, 2012, Published on SEDAR.com.

Norwest Corporation (November 8, 2013). Prefeasibility Study Report: Blawn Mountain Project, Beaver County, Utah, submitted to Potash Ridge Corporation.

Oxtoby, David. W., Gillis, H. P., and Campion, Alan. (2008). Principles of Modern Chemistry, Thomson Books/Cole.

Parkinson, Gerald (1974) Golden pilot plant points way to 500,000-tpy alumina-from-Alunite mine and plant in Utah: Engineering and Mining Journal, August, p. 75-78.

Perry, Harry (1977_ Mining plan for NG Alunite area C – first five years: Unpublished Earth Sciences, Inc. report, 122 p.

Potash Ridge Corporation, Toronto, and Hatch Ltd., Mississauga, ON. (February 16, 2012). Blawn Mountain Alunite Project: Project Update. Power Point Presentation.

Rinckhoff, J. B. (1959). “Modern American Contact Plants, Section I. Chemico-Designed Contact Plants,” in The Manufacture of Sulfuric Acid, American Chemical Society, pp. 230-245.

Rowley, P.D., Lipman, P.W., Mehnert, H.H., Lindsay, D.A., and Anderson, J.J. (1978) Blue Ribbon lineament, an east-trending structural zone within the Pioche mineral belt of southwestern Utah and eastern Nevada: US Geological Survey Journal of Research, v. 6, p. 175-192.

Salisbury, Tyler J. (June 24, 2013). “Alunite Process Development and Pilot Plant Program Interim Report,” Hazen Project 11687, Hazen Research, Inc. Golden, CO.

Shawe, D.R., and Stewart, J.H. (1976) Ore deposits as related to tectonics and magmatism, Nevada and Utah: American Institute of Mining Engineers Transactions, v. 260, p. 225-232.

Shouldice, Tom, Sloan, Robert, and Pojhan, Aralia, ALS Metallurgy Kamloops. (June 5, 2013). Metallurgical Assessment of The Blawn Mountain project, Utah, United States. KM3816.

Sinnott, R. K. (2005). Coulson & Richardson,Chemical Engineering, Vol. 6: Chemical Engineering Design, 4th ed. Elsevier.

Sisselman, R. (October 1975). Engineering and Mining Journal (E&MJ), Vol. 176, N0. 10, pp. 79-84.

SNC-Lavelin Inc., 2017, Blawn Mountain Scaling Study Report, 639527-0000-40RA-0008, 47 p.

Stephens, J.C. (1974) Hydrologic reconnaissance of the Wah Wah Valley drainage basin, Beaver and Millard Counties, Utah: US Geological Survey Open-File Report 137, 79 p. [also Utah Department of Natural Resources Technical Publication 47].

Susarla, V. R. K., Chudasama, K. M., Mohandas, V. P., and Ghosh, P. K. (June 2007). “Glaserite Preparation by Sodium Sulfate and Potassium Chloride,” in Journal of Scientific and Industrial Research, Vol. 66, pp. 444-449.

Tippin, R. B., and Huiatt, J. L. (March 7, 1977). A New potash Flotation Process from Laboratory Testing Through Pilot Plant Studies and on to Full-Scale Operation, Paper presented at the 106th AIME Annual meeting, Atlanta, GA.

Tippin, R. B. (July 18, 1977). “Potash Flotation Process Handles Variable Feed,” in Chemical Engineering, pp. 73-75.

Thompson, David L. (December 9, 2011). Recovery of K_2SO_4 and H_2SO_4 (and Possibly Alumina) from Alunite Deposits in Southwestern Utah, for Potash Ridge Corporation. HAZEN Project No. 11416, Hazen Research, Inc. Golden, CO.

Thompson, A.J.B. (1991) Characteristics of acid-sulfate alteration in the Marysvale-Pioche mineral belt: a guide to gold mineralization: Utah Geological and Mineral Survey Miscellaneous Publication 91-2, 29 p.

US Bureau of Land Management (1977) Alunite Project – Final Environmental Statement: US Bureau of Land Management, US Department of the Interior, variously paginated.

US Bureau of Land Management (2011)[a] Cedar City Field Office Resource Management Plan, Major Rights-of-Way Map: Online at, https://www.blm.gov/epl-front-office/projects/lup/7100/17354/17554/CCFO_ROW_42x50L.pdf, accessed 07/05/2011.

US Bureau of Land Management (2011)[b] Cedar City Field Office Resource Management Plan, Allotment Map: Online https://www.blm.gov/epl-front-office/projects/lup/7100/17353/17553/CCFO_HMA-Allotment_42x50L.pdf, accessed 07/05/2011.

US Bureau of Land Management (2011)[c] Cedar City Field Office Resource Management Plan, Wilderness Study Areas and Recreation Map: Online < https://www.blm.gov/epl-front-office/projects/lup/7100/17355/17555/CCFO_Wilderness-3PeakSRMA42x50L.pdf>, accessed 7.511.

US Bureau of Land Management (2011)[d] Cedar City Field Office Resource Management Plan, Wildlife map: Online https://www.blm.gov/epl-front-office/projects/lup/7100/17352/17552/CCFO_Wildlife_42x50L.pdf, accessed 7/5/2011.

US Bureau of Land Management (2011)[e] Cedar City Field Office Resource Management Plan, Renewable Energy Resources Map: Online at, https://www.blm.gov/epl-front-office/projects/lup/7100/17350/17550/CCFO_RenewableResources_42x50L.pdf, accessed 07/05/2011.

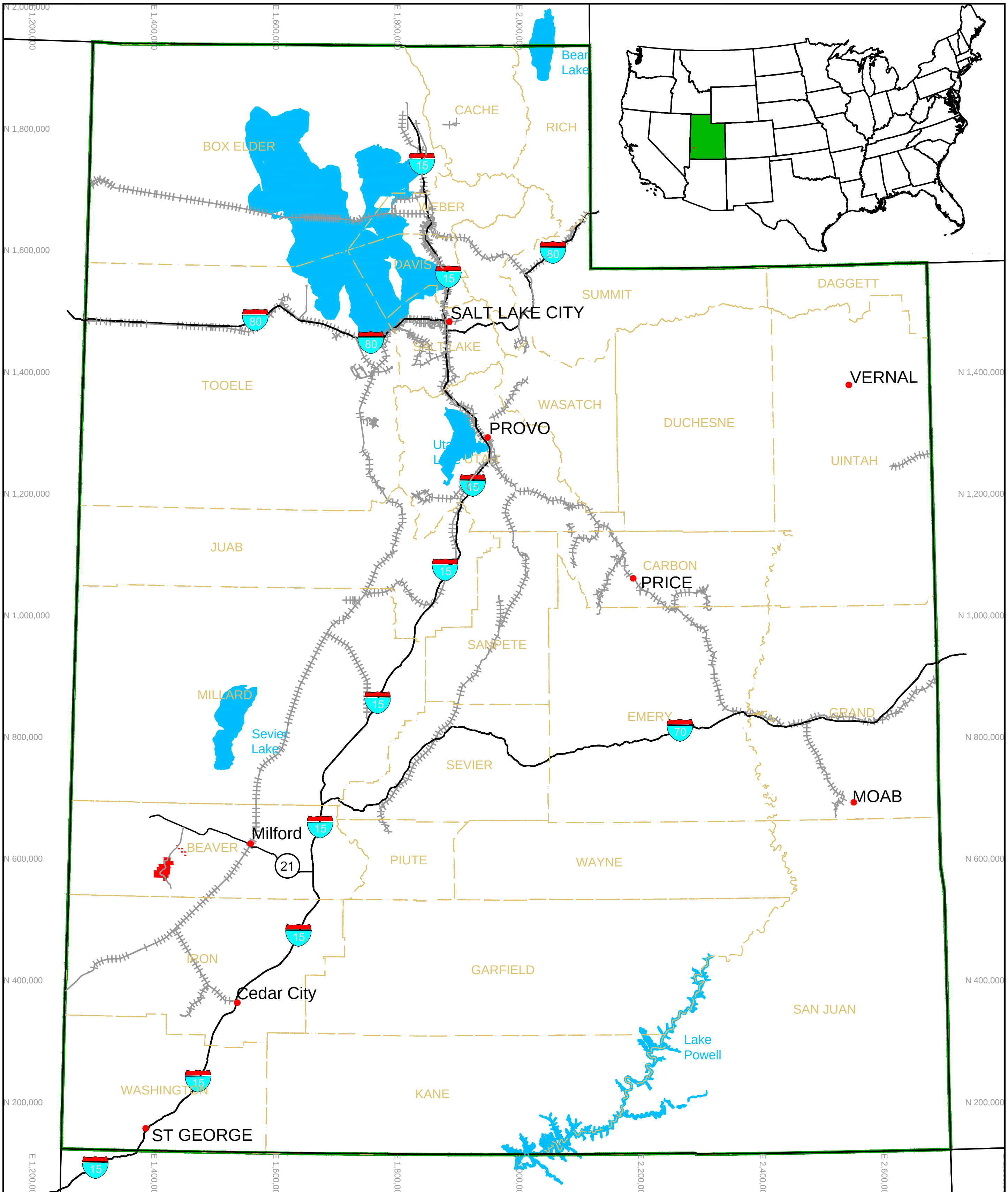
US Department of Energy (2011) West-wide Energy Corridor Programmatic Environmental Impact Statement: Online at: http://corridoreis.anl.gov/, accessed July 6, 2011.

Walker, W.W. (1972) Report of results phase III exploration program on the NG Alunite property Beaver County, Utah: Unpublished report for National -Southwire Aluminum Co., and Earth Sciences, Inc., 123 p., appendices, and plates.

Walker, M.W. (1973) Evaluation of the tonnage and grade of the White Mountain Alunite deposit, Beaver County, UT: Unpublished report for Earth Sciences, Inc., 21 p., appendices and plates.

Whelan, J.A. (1965) Hydrothermal Alteration and Mineralization, Staats Mine and Blawn Mountain Areas, central Wah Wah Range, Beaver County, Utah: Utah Geological and Mineralogical Survey Special Studies 12, 31 p

28 ILLUSTRATIONS



POTASH RIDGE

LEGEND

Freeway

Railroad

City or Town

Counties

Blawn Mountain Project

Major Lakes

0 10 20 30

MILE

UTAH STATE PLANE-SOUTH ZONE NAD27

FIGURE 4.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

GENERAL

LOCATION MAP

DATE: 10/24/2016

BY:

Millcreek Mining GROUP

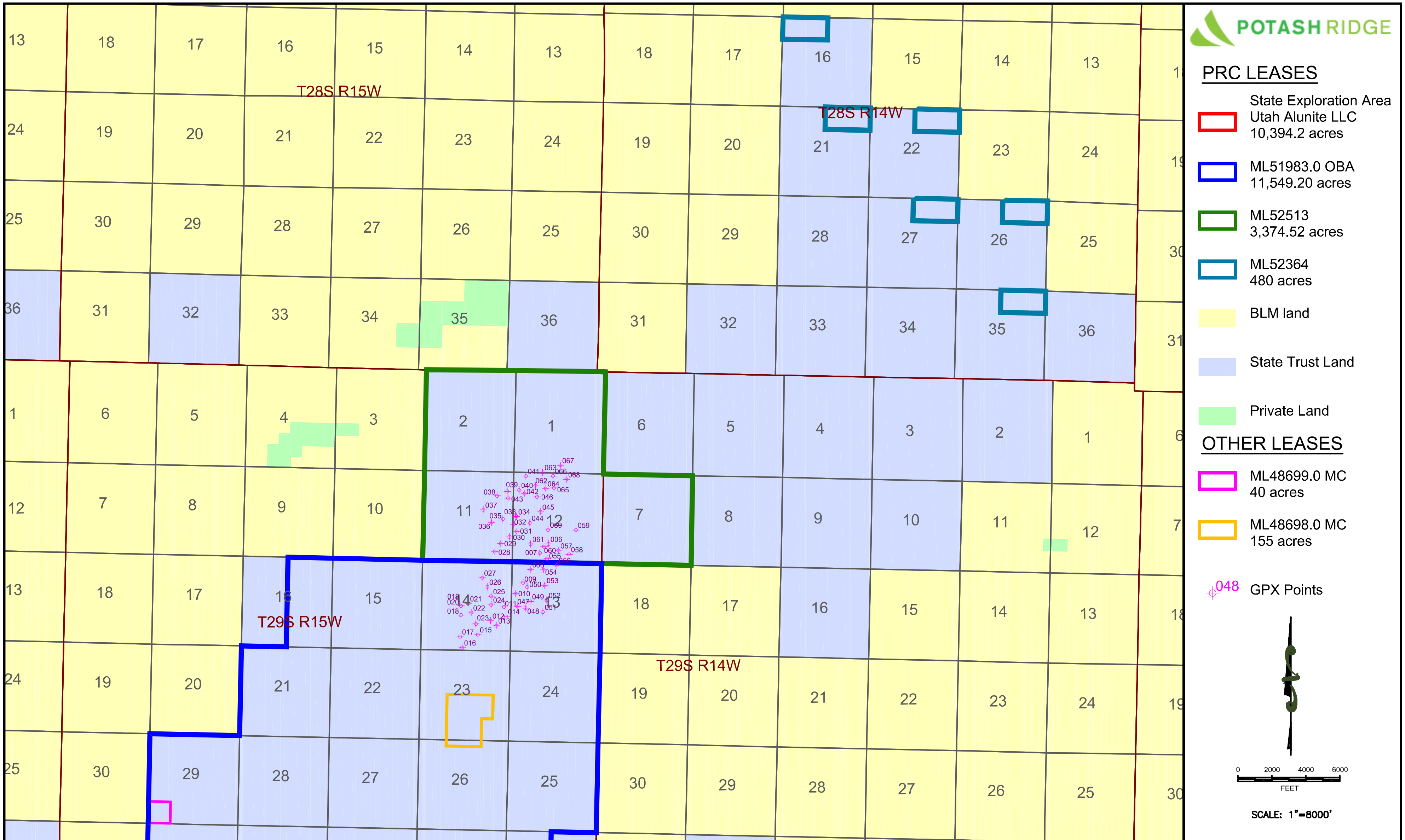


FIGURE 4.2

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

EXPLORATION/OPTION AREA LOCATION

DATE: 10/24/2016
BY: [signature]

Millcreek Mining GROUP

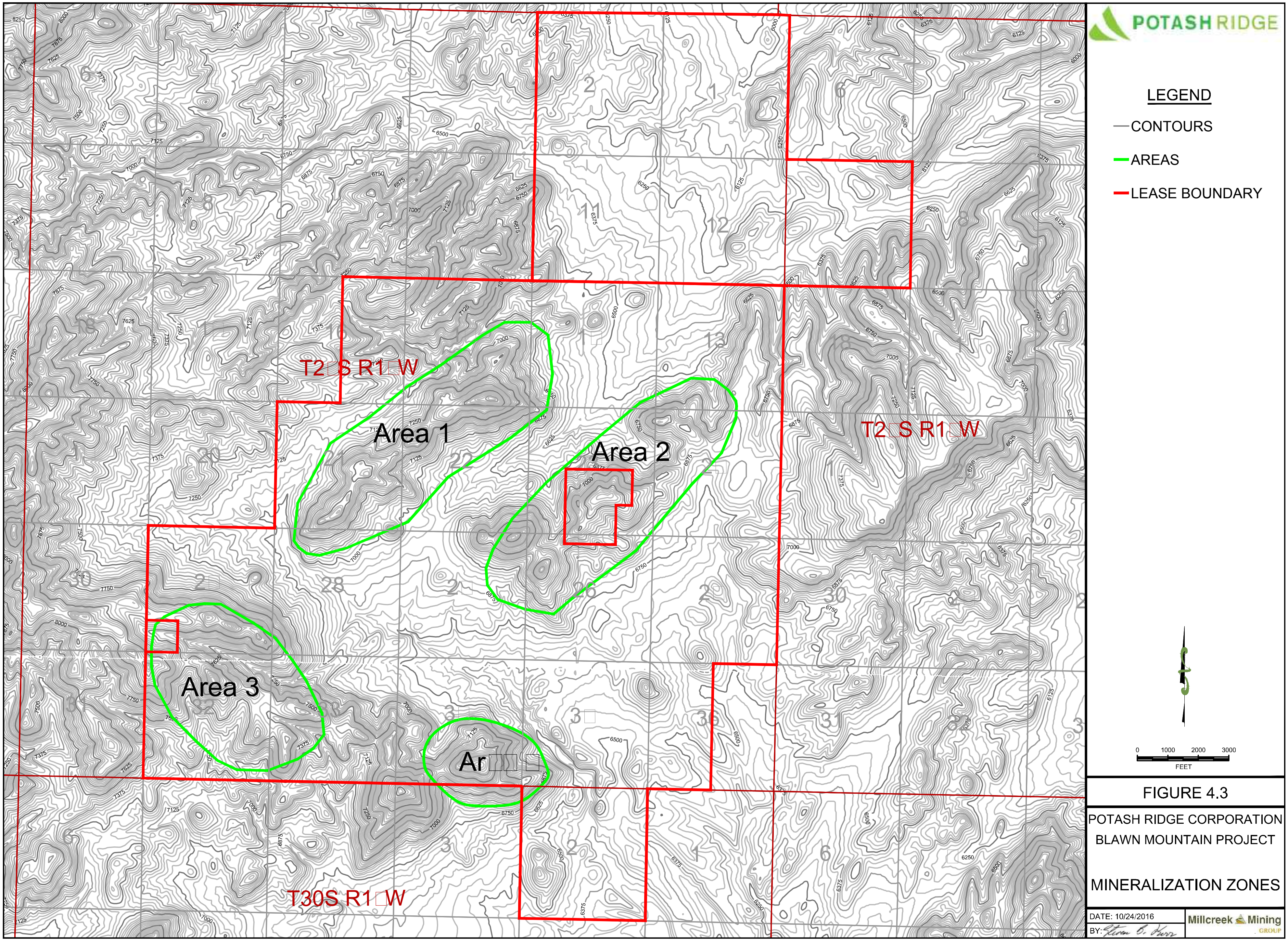
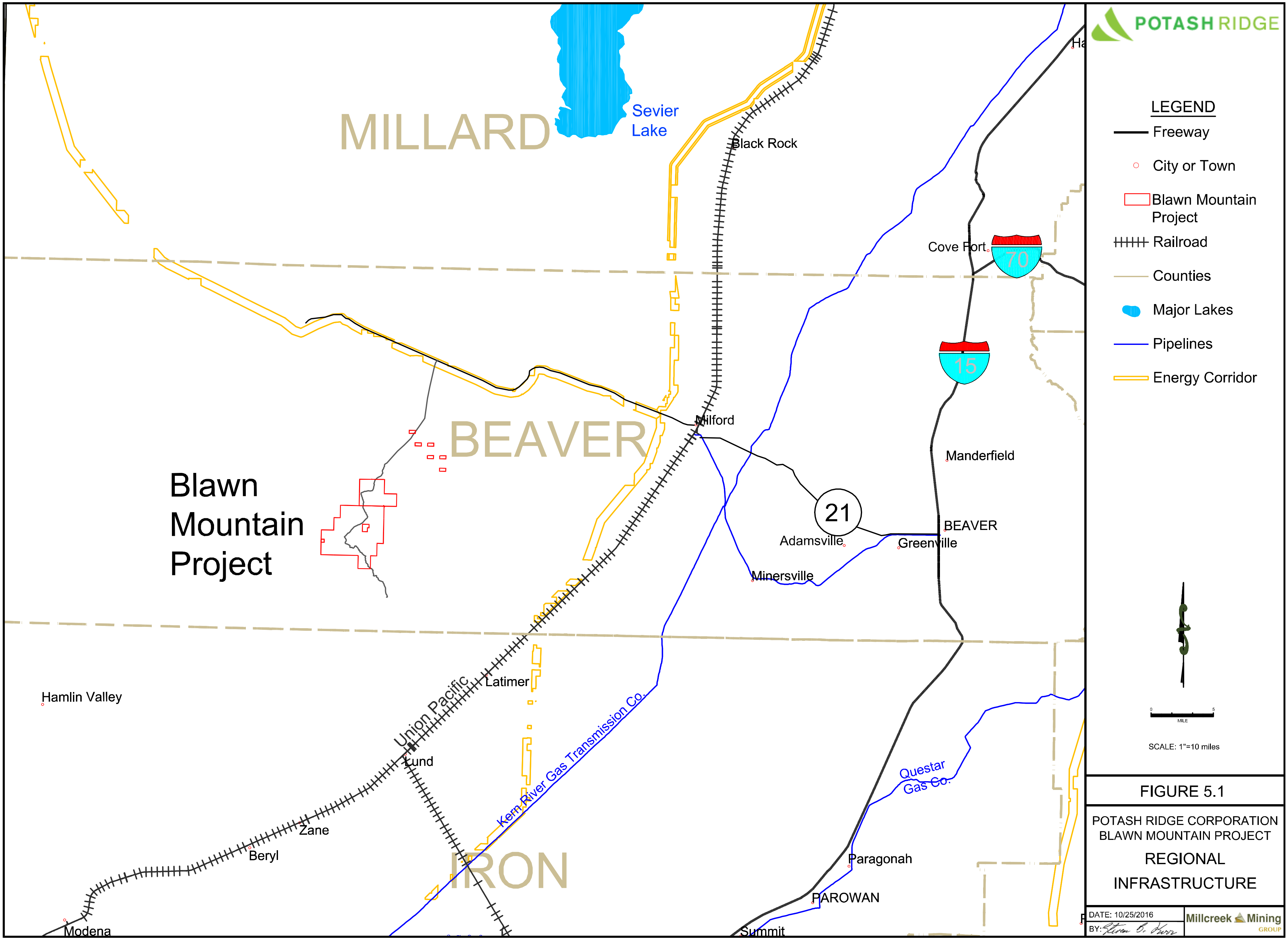


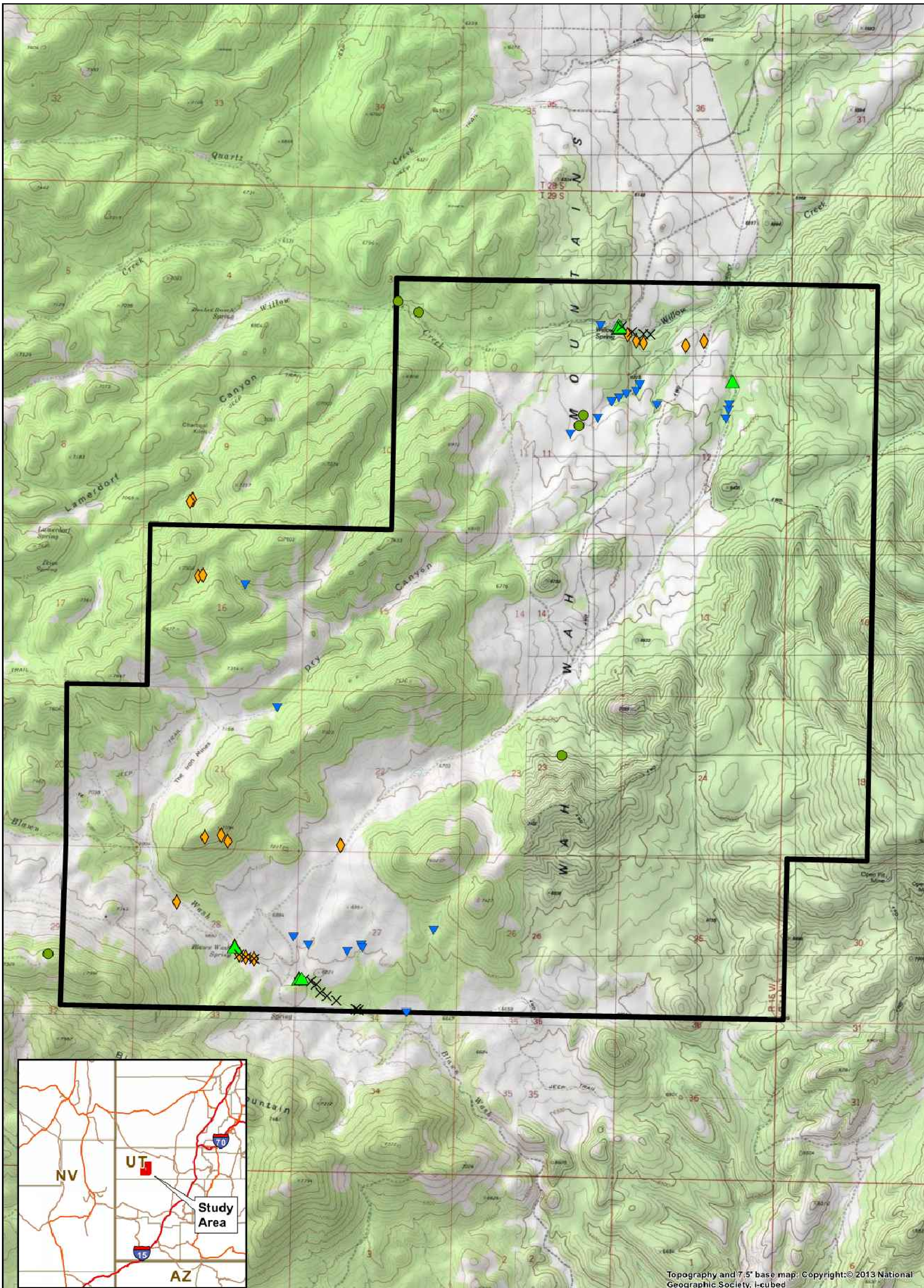
FIGURE 4.3

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

MINERALIZATION ZONES



MILLARD
Sevier Lake
Black Rock
Cove Fort
70
15
Milford
BEAVER
Manderfield
21
BEAVER
Adamsville
Greenville
Minersville
Blawn Mountain Project
Hamlin Valley
Latimer
Union Pacific
Lund
Kern River Gas Transmission Co.
Questar Gas Co.
Zane
Beryl
Modena
IRON
Paragonah
PAROWAN
Summit
POTASH RIDGE
LEGEND
Freeway
City or Town
Blawn Mountain Project
Railroad
Counties
Major Lakes
Pipelines
Energy Corridor
SCALE: 1"=10 miles
FIGURE 5.1
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT
REGIONAL INFRASTRUCTURE
DATE: 10/25/2016
BY:
Millcreek Mining GROUP



POTASH RIDGE

LEGEND

Study Area Boundary

Historic Water Sources

2013 Survey

Spring

Seep

Observation point

Notable Vegetation

0 3,000 6,000

Feet

FIGURE 5.2

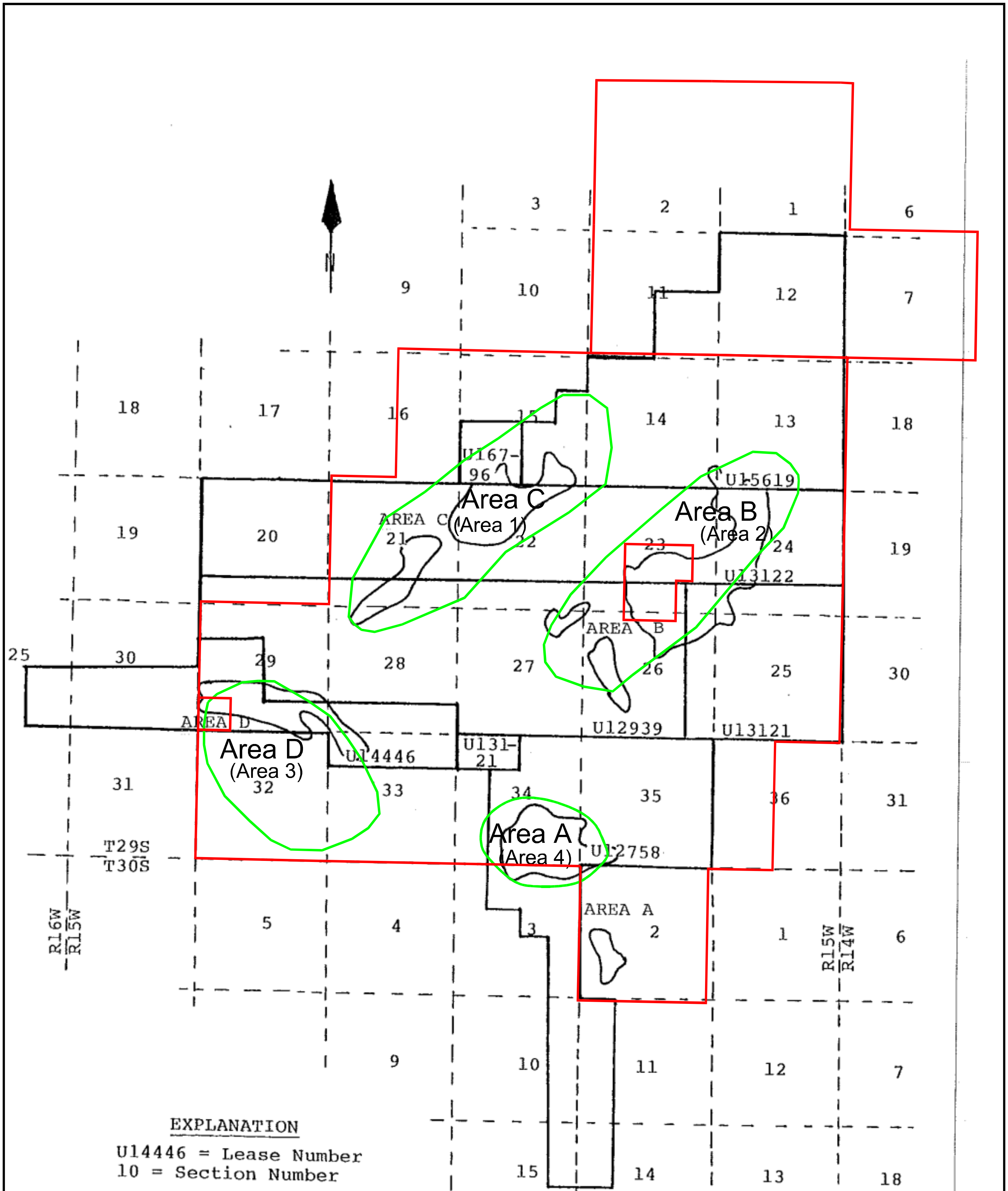
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

SPRING & SEEP SURVEY
APRIL - MAY 2013

DATE: 10/26/2016

BY: Steven B. Kerr

Millcreek Mining GROUP



LEGEND

CURRENT BLAWN MTN. PROPERTY

FORMER NG ALUNITE PROPERTY

0 1000 2000 3000 4000 5000
FEET

POTASH RIDGE

FIGURE 6.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

FORMER NG ALUNITE PROJECT AREAS

DATE: 10/26/2016

BY: Steven B. Kerr

Millcreek Mining GROUP

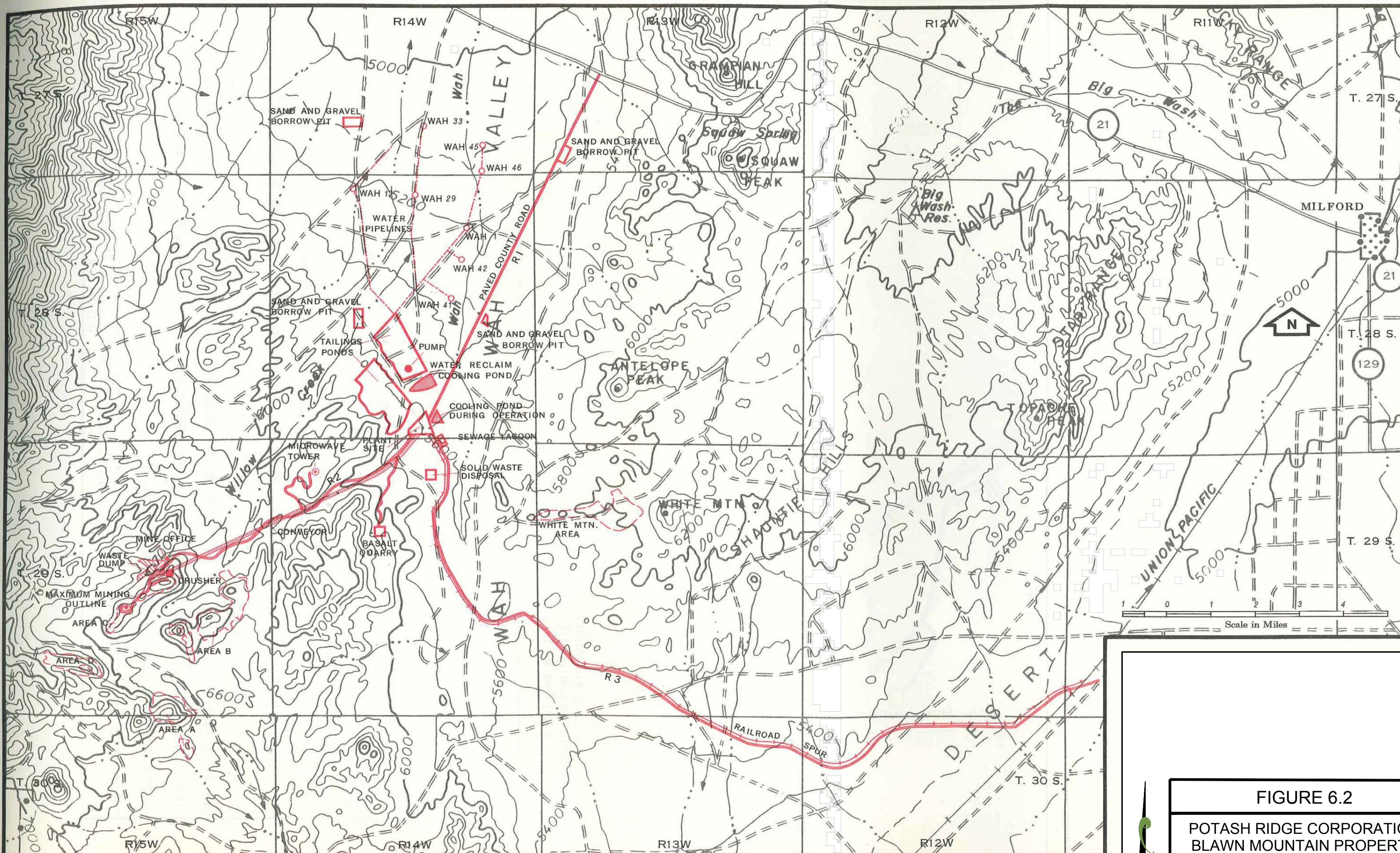


FIGURE 6.2

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROPERTY

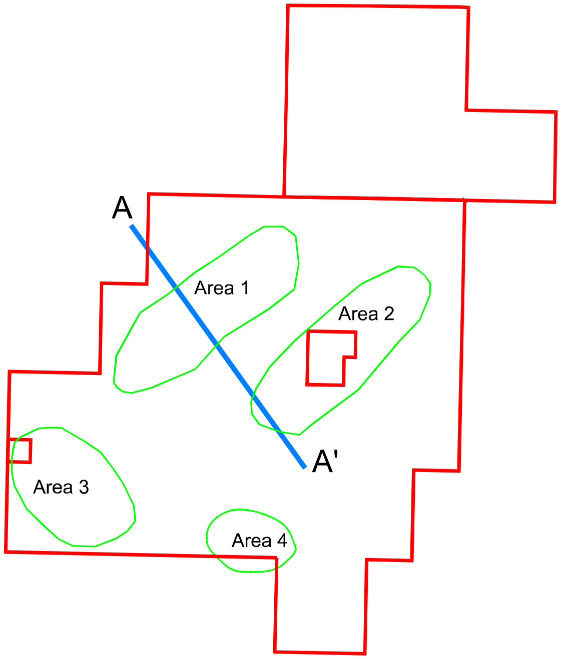
FORMER NG PROPERTY
DEVELOPMENT PLAN

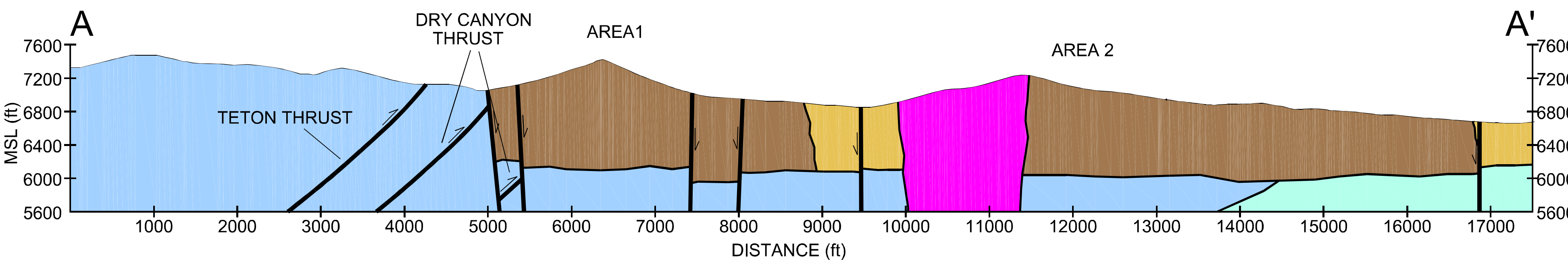
DATE: 10/26/2016

BY: Steven B. Kerr

Millcreek Mining GROUP

POTASH RIDGE





Modified from Abbott and others (1983)

VOLCANIC ROCKS

	Trp	RHYOLITE PORPHYRY
	Tva	ALTERED IGNIMBRITES
	Tvu	UNDIFFERENTIATED VOLCANICS

SEDIMENTARY ROCKS

	Mzu	MESOZOIC SANDSTONE AND SHALE
	Pzu	PALEOZOIC CARBONATES AND QUARTZITES

FAULT - ARROW INDICATES RELATIVE DIRECTION OF MOVEMENT

CONTACT

POTASH RIDGE

FIGURE 7.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

THEMATIC CROSS SECTION

DATE: 11/1/2016

BY: [signature]

Millcreek Mining GROUP

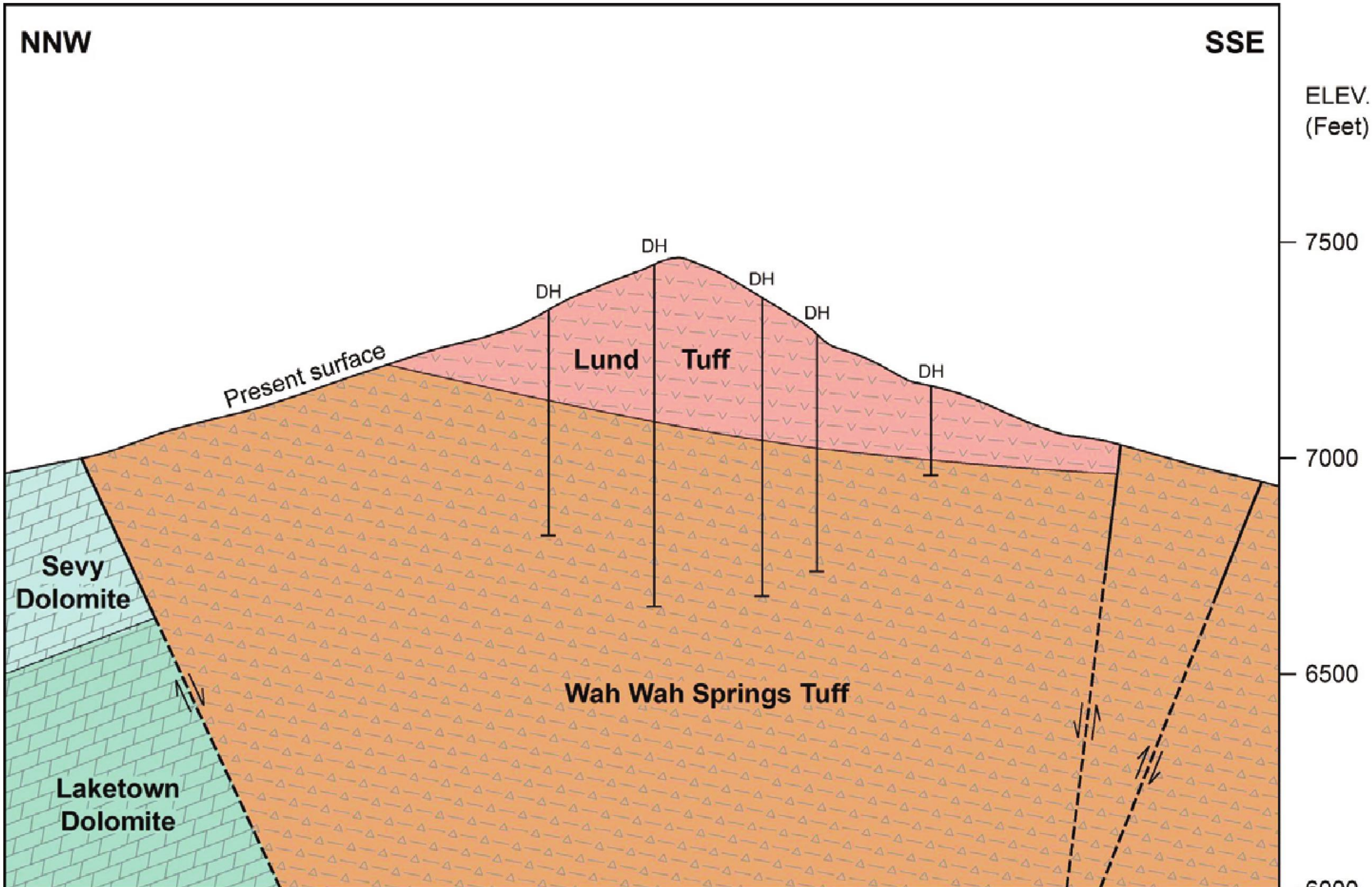


FIGURE 7.2

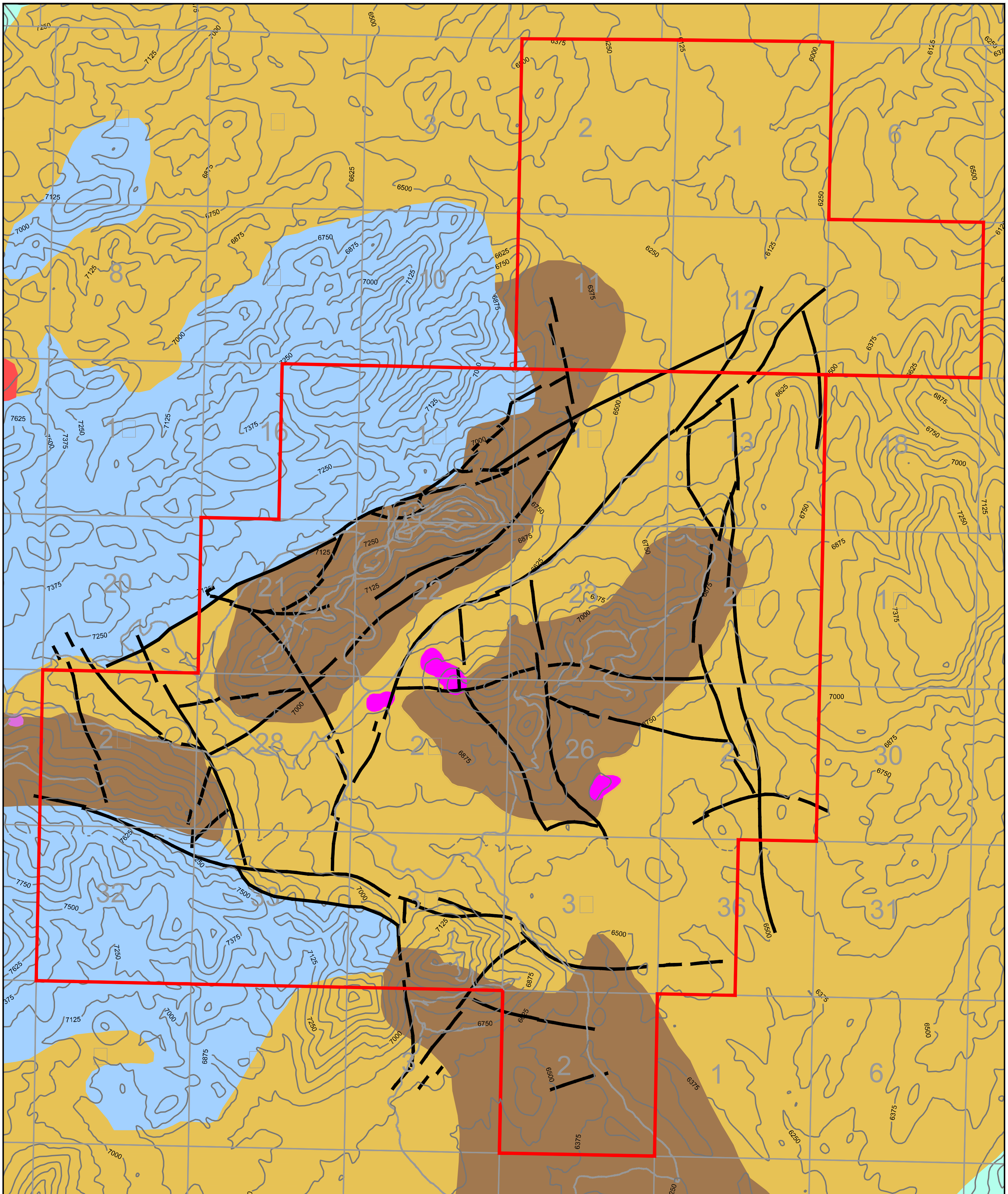
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

LOCAL GEOLOGICAL CROSS SECTION

DATE: 10/28/2016
FILE: Fig 7.2 and 7.5.dwg

Millcreek Mining GROUP

POTASH RIDGE



LEGEND

VOLCANIC ROCKS

- Tr□ — RHYOLITE PORPHYRY
- T□□ — ALTERED IGNIMBRITES
- T□□ — UNDIFFERENTIATED VOLCANICS

SEDIMENTARY ROCKS

- T□□ — CONGLOMERATE
- M□□ — MESO□OIC SANDSTONE AND SHALE
- P□□ — PALEO□OIC CARBONATES AND □UART□ITES

- ROADS
- LEASE BOUNDARY
- NORMAL FAULTS
- THRUST FAULTS

0 2000 4000 6000

FEET

POTASH RIDGE

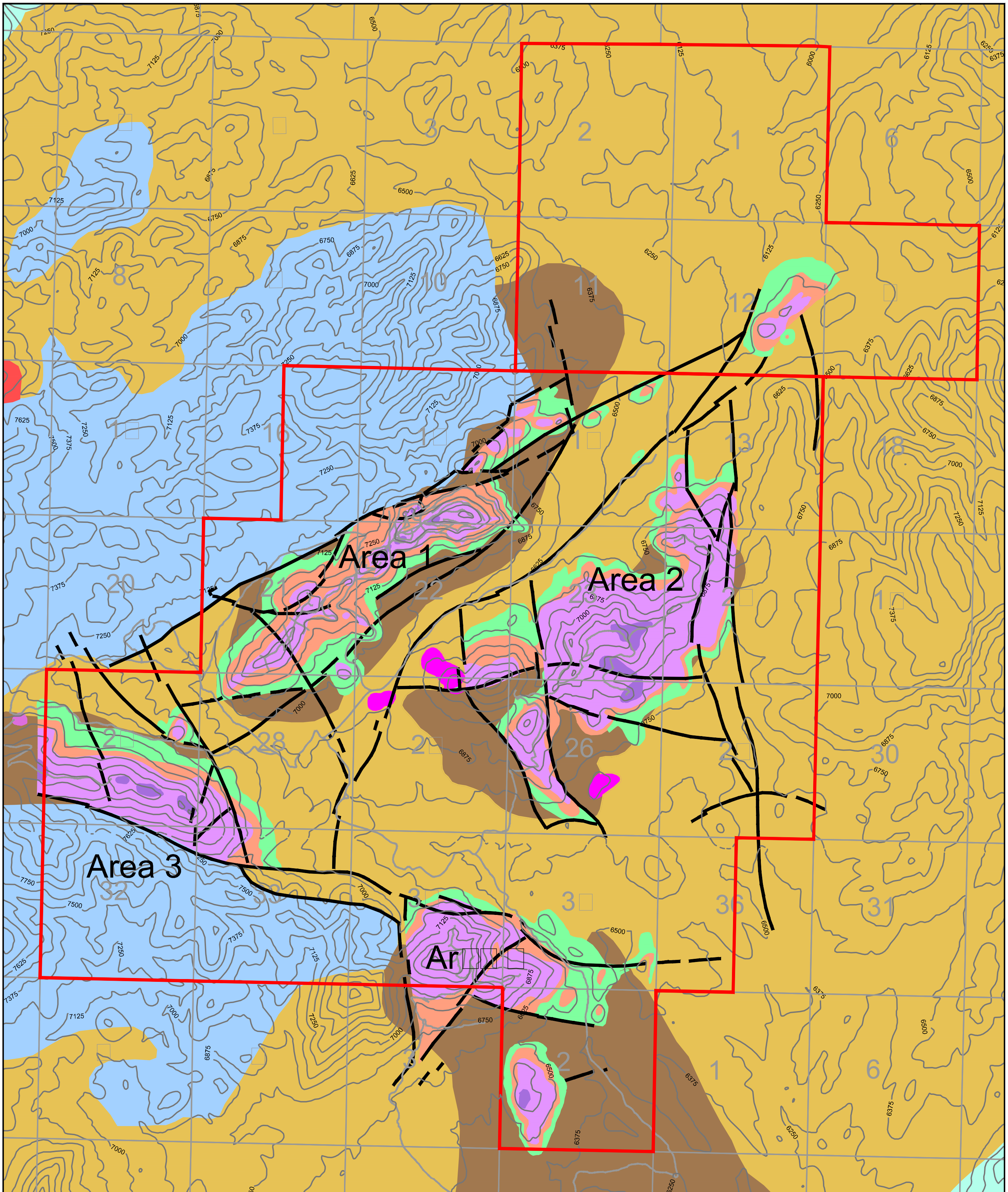
FIGURE 7.3

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

SURFACE
GEOLOGY MAP

DATE: 11/1/2016
BY:
SCALE: 1:55,000

Millcreek Mining GROUP



LEGEND

ALTERATION

- SILICA □ONE
- □UART□ALUNITE
- HEMATITE□CLAY
- PROPYLITIC □ONE

VOLCANIC ROCKS

- Tr□ RHYOLITE PORPHYRY
- T□□ ALTERED IGNIMBRITES
- T□□ UNDIFFERENTIATED VOLCANICS

SEDIMENTARY ROCKS

- T□□ CONGLOMERATE
- M□□ MESO□OIC SANDSTONE AND SHALE
- P□□ PALEO□OIC CARBONATES AND □UART□ITES

- ROADS
- LEASE BOUNDARY
- NORMAL FAULTS
- THRUST FAULTS

0 2000 4000 6000
FEET

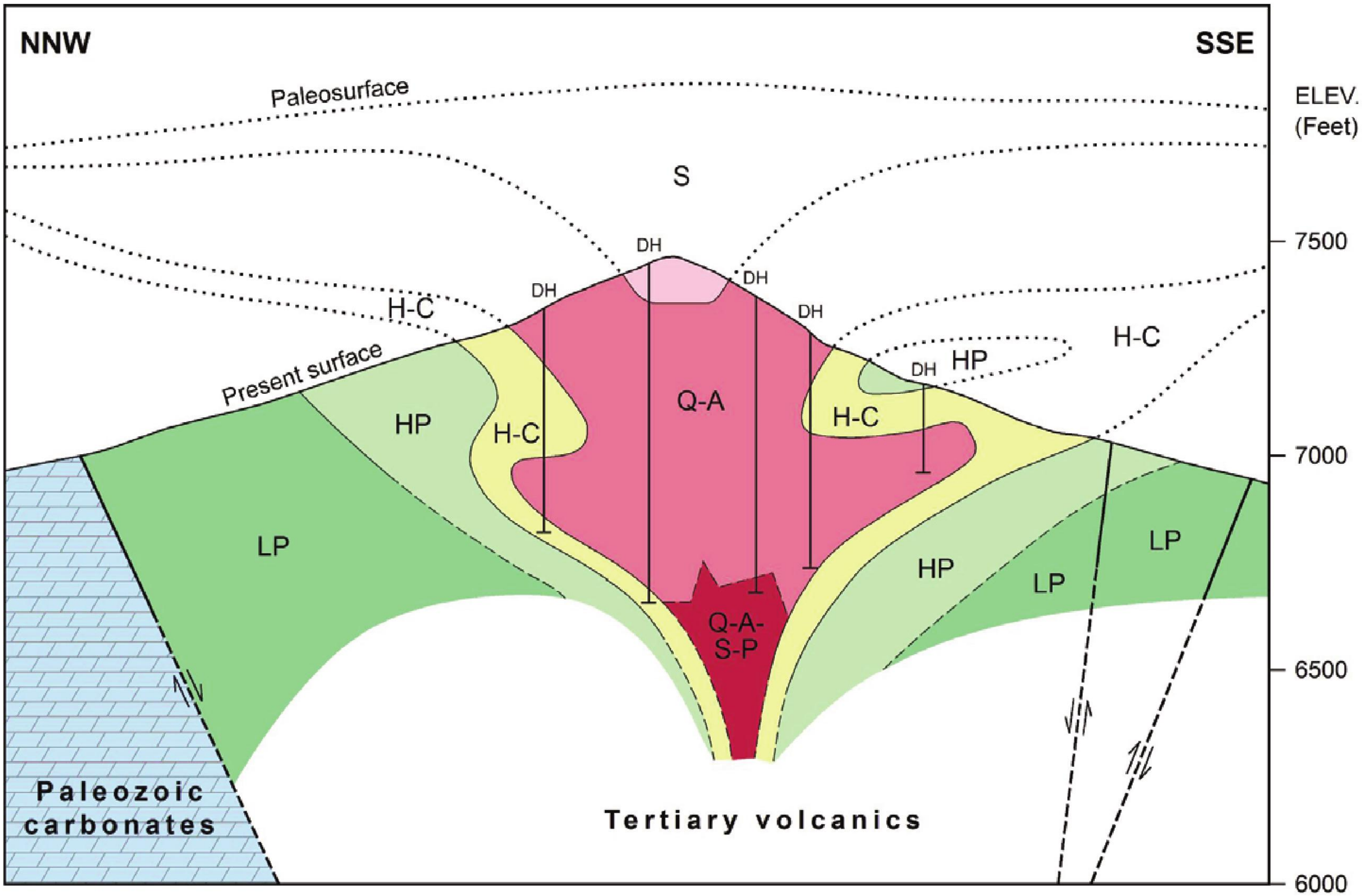
POTASH RIDGE

FIGURE 7.4

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

ALUNITE
ALTERATION AREAS

DATE: 11/1/2016
BY:
SCALE: 1:55,000
Millcreek Mining GROUP



Q-A-S-P	Quartz-alunite-sericite-pyrite zone
Q-A	Quartz-alunite zone
S	Silica zone
H-C	Hematite-clay zone
HP	High propylitic zone
LP	Low propylitic zone

POTASH RIDGE

FIGURE 7.5

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

ALUNITE ALTERATION
CROSS SECTION

DATE: 06/25/2013
FILE: Fig 7.2 and7.5.dwg

Millcreek Mining GROUP

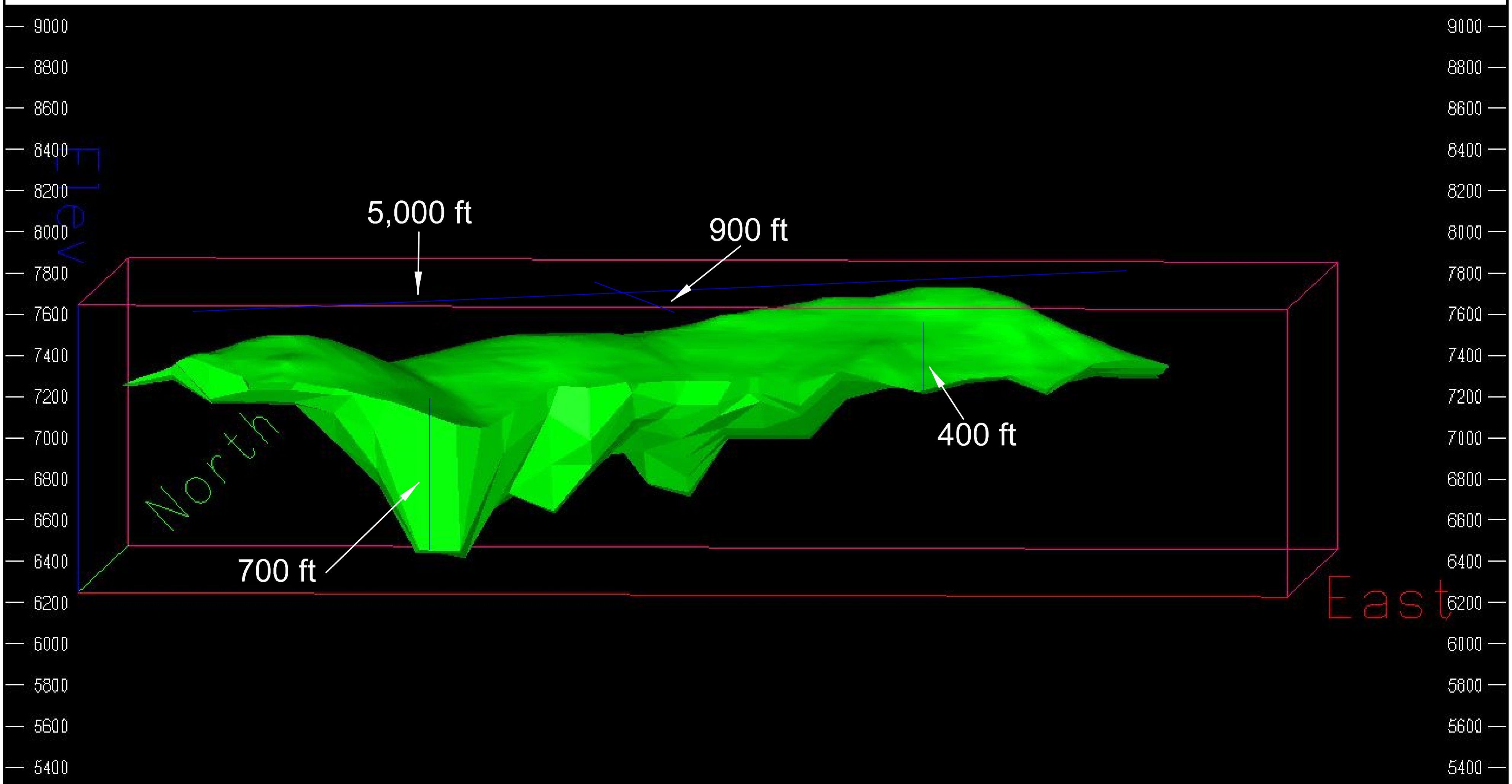


FIGURE 7.6

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

3DBM REPRESENTATION
OF AREA 1 ALUNITE ZONE

DATE: 11/1/2016
BY:
SCALE: as noted

POTASH RIDGE

Millcreek Mining GROUP

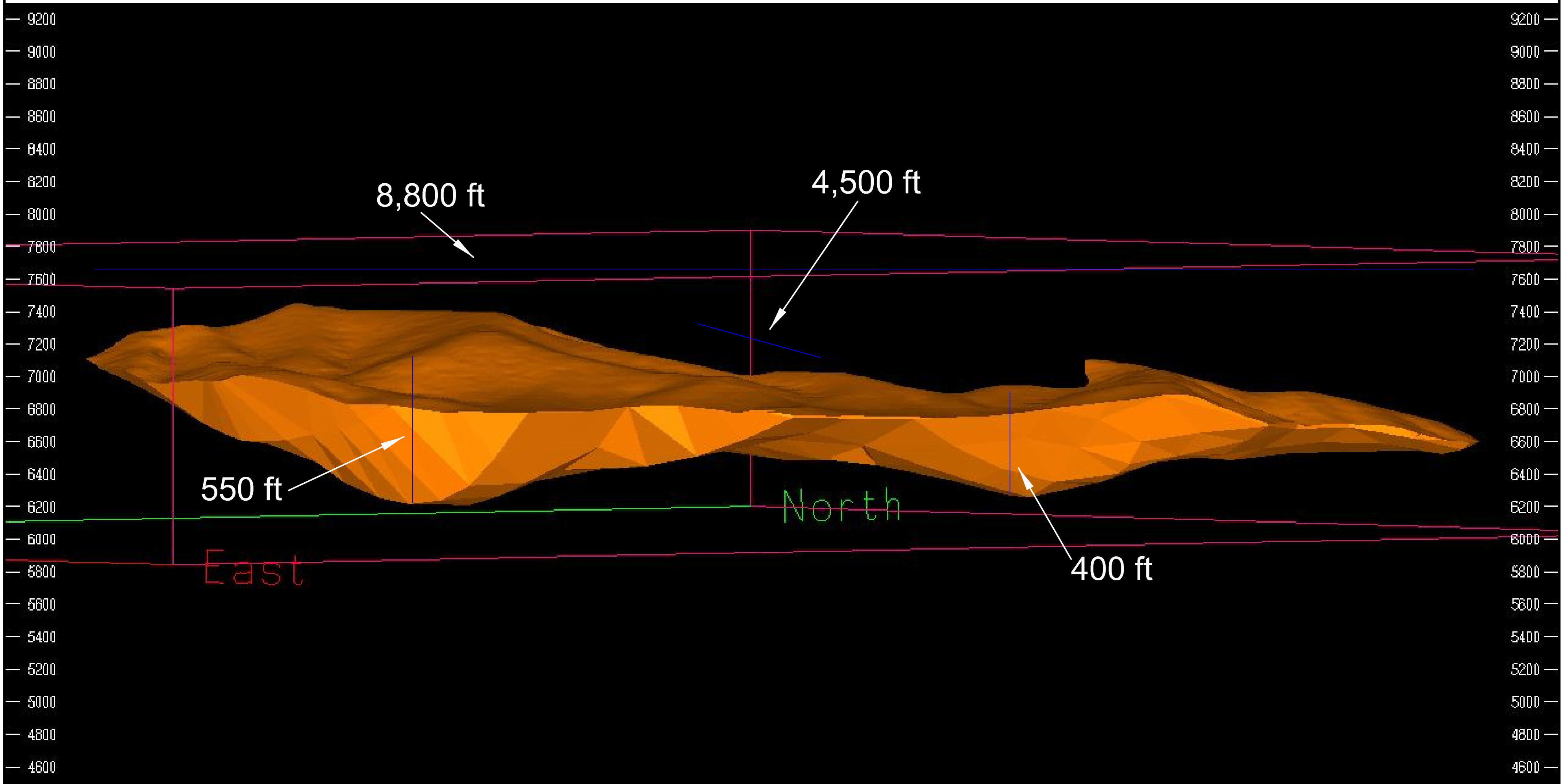


FIGURE 7.7

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

3DBM REPRESENTATION
OF AREA 2 ALUNITE ZONE

DATE: 11/1/2016	SCALE:	Millcreek Mining GROUP
BY: [signature]	as noted	

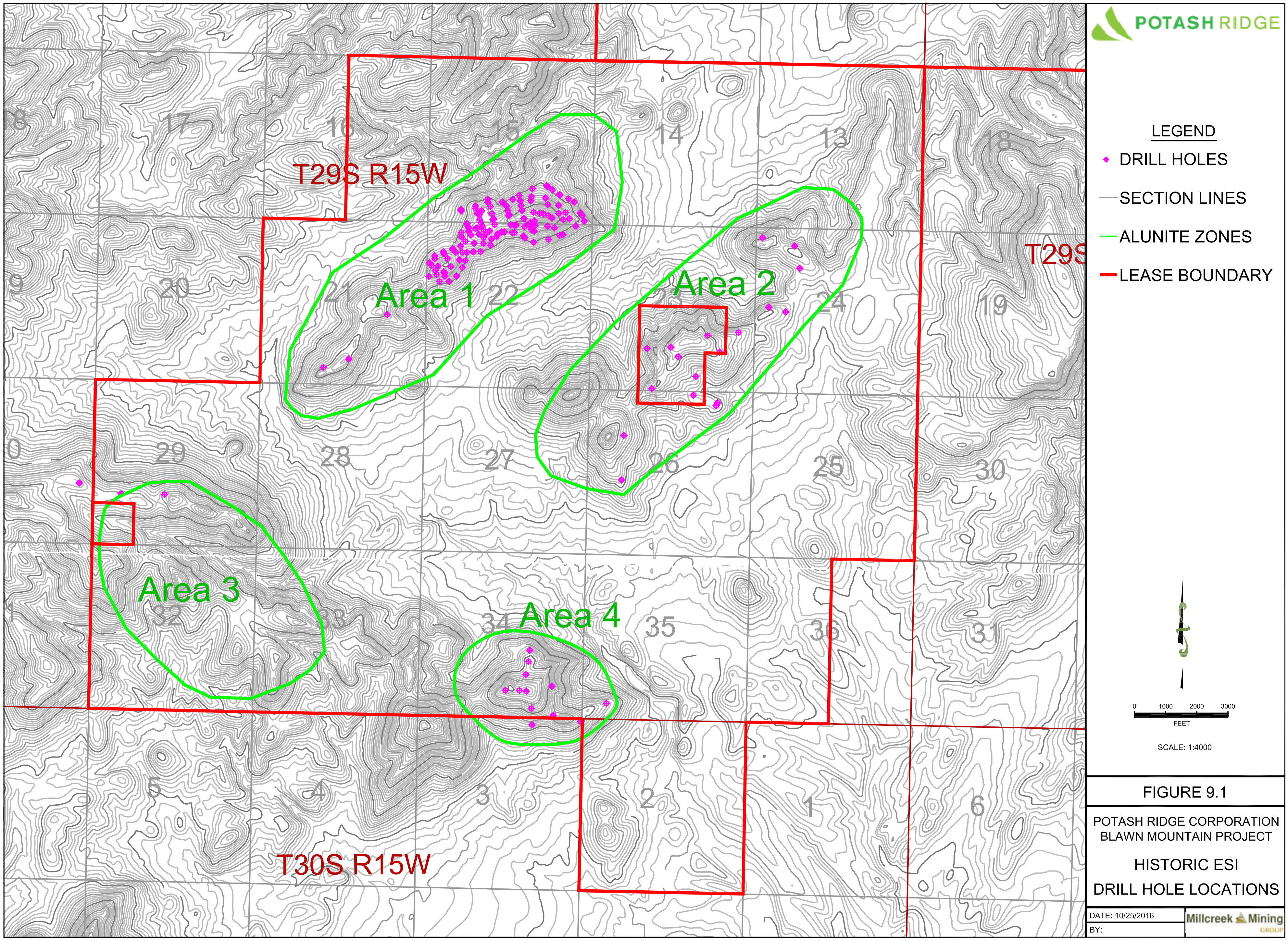
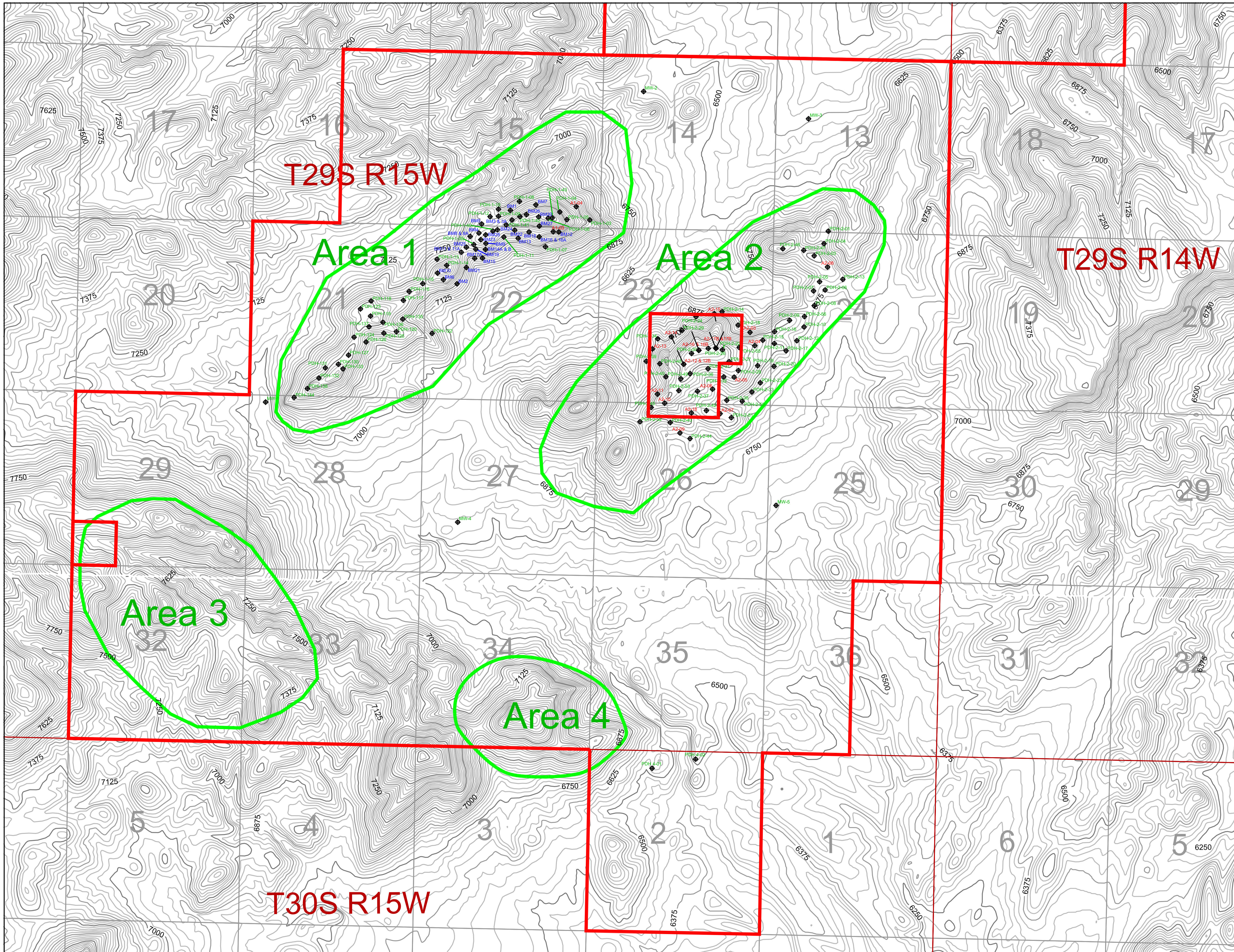


FIGURE 9.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

HISTORIC ESI
DRILL HOLE LOCATIONS



POTASH RIDGE

BM7 2012 PHASE 1 DRILL HOLES

PDH-2-49 2012 PHASE 2 DRILL HOLES

A2-06 2013 DRILL HOLES

SECTION LINES

ALUNITE ZONES

LEASE BOUNDARY

CONTOURS

0 1000 2000 3000 4000

FEET

FIGURE 10.1

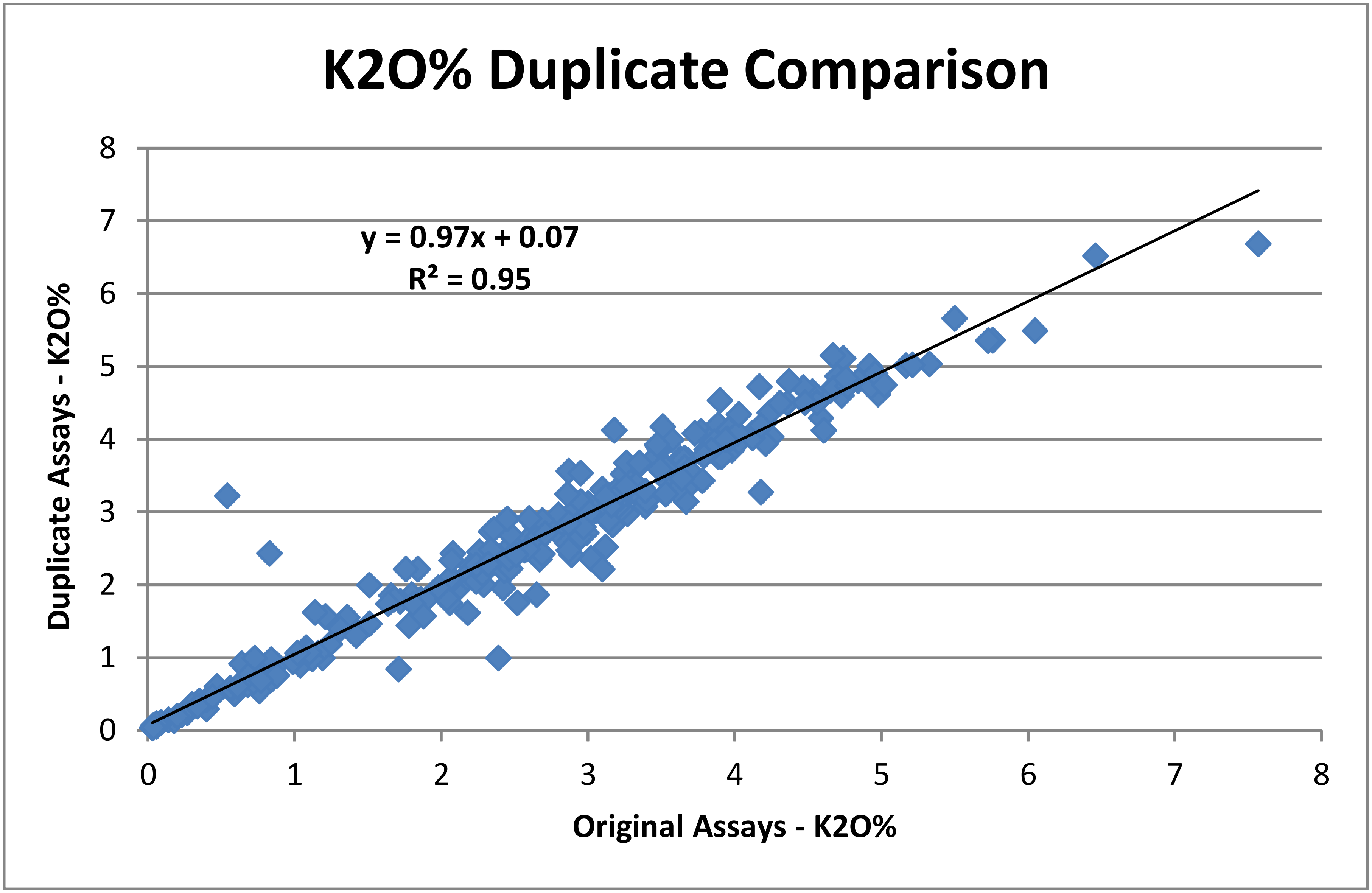
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

DRILLING LOCATIONS

DATE: 11/01/2016

BY:

Millcreek Mining GROUP



POTASH RIDGE

FIGURE 12.1

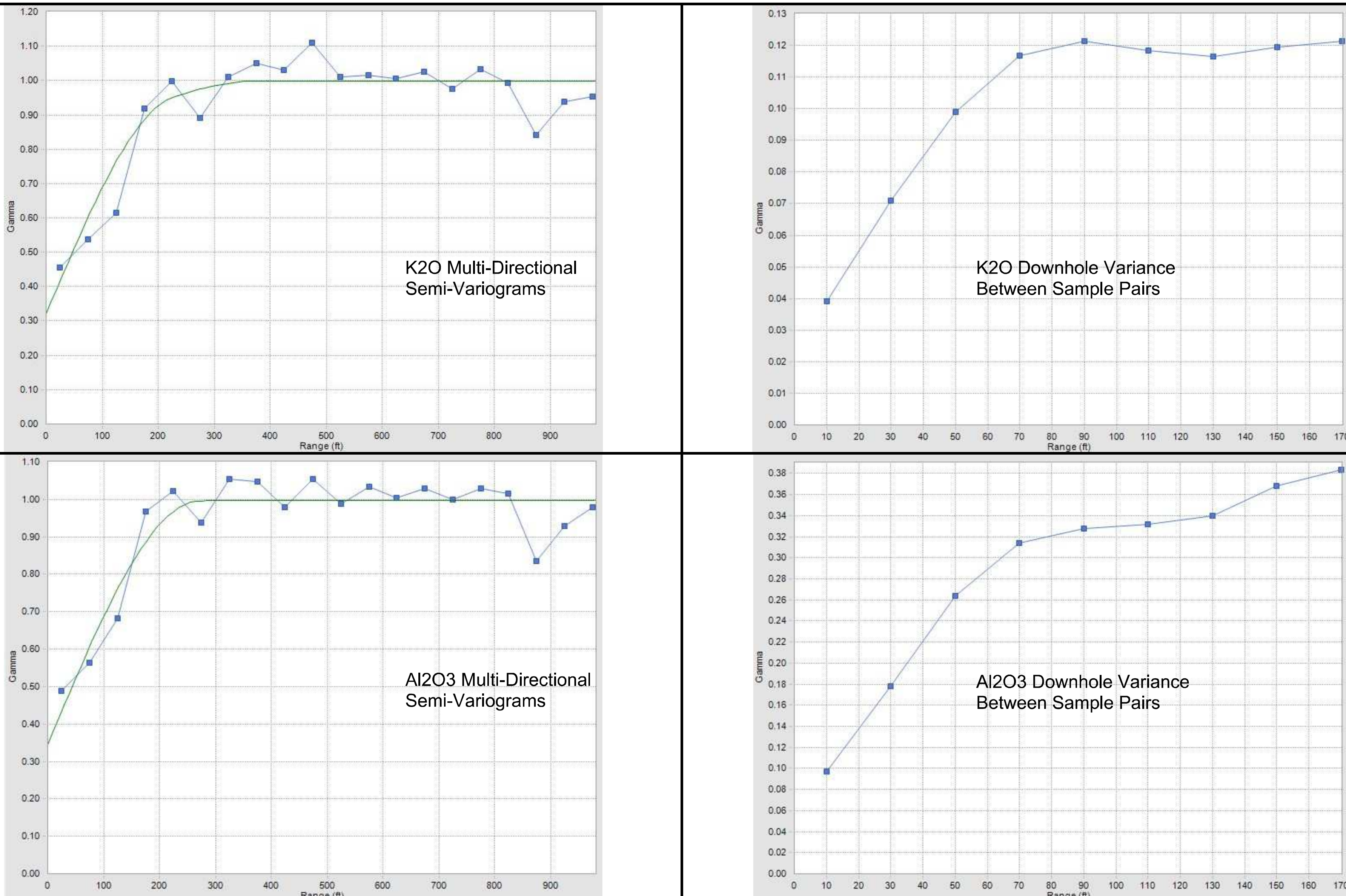
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

K2O COMPARISONS FOR ORIGINAL and DUPLICATE ASSAYS

DATE: 11/2/2016

BY:

Millcreek Mining GROUP



POTASH RIDGE

Best Fit Experimental Semi-Variograms (Spherical Model)

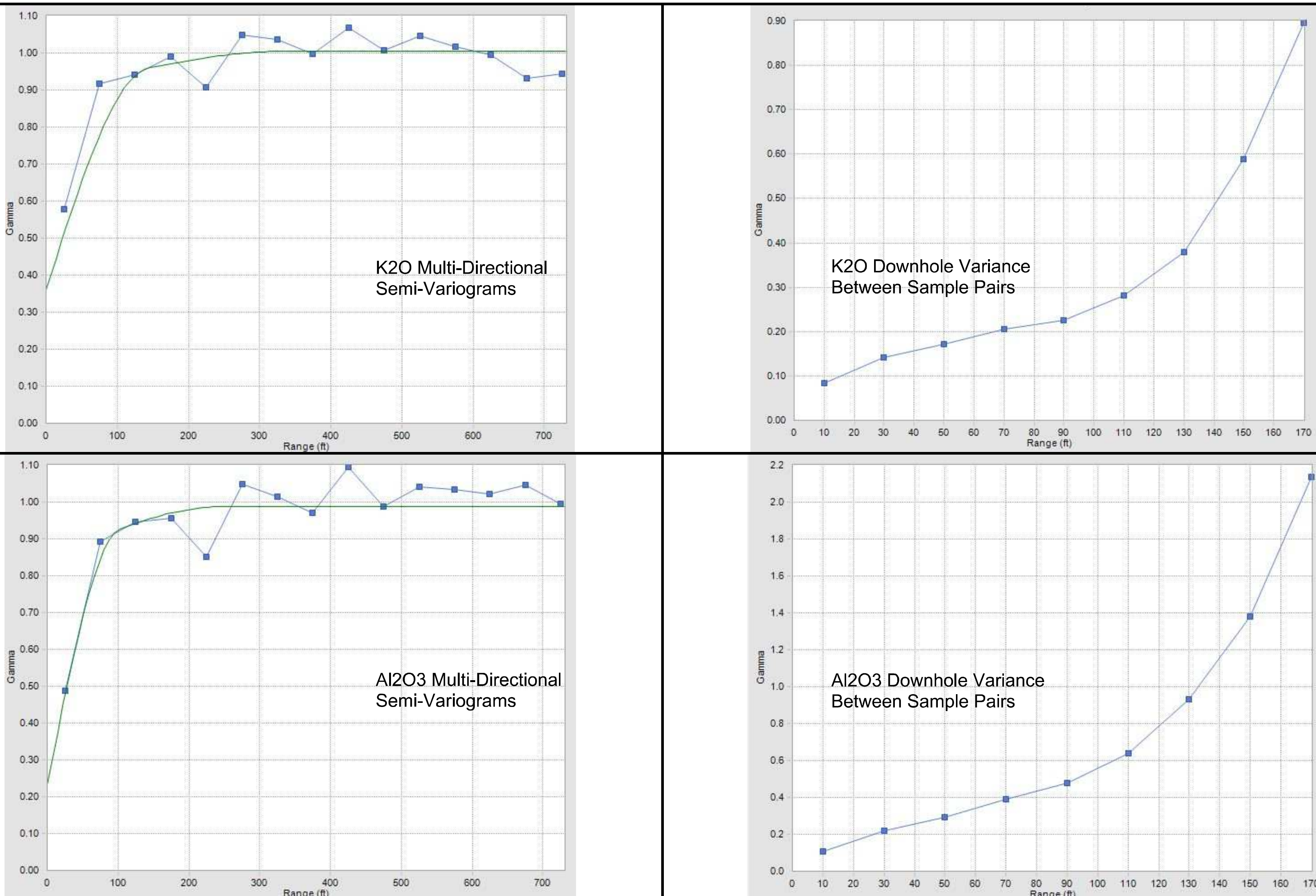
FIGURE14.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 1 VARIOGRAMS OF ANALYTICAL RESULTS

DATE: 11/2/2016
BY:

Millcreek Mining GROUP



POTASH RIDGE

Best Fit Experimental
Semi-Variograms (Spherical Model)

FIGURE14.2

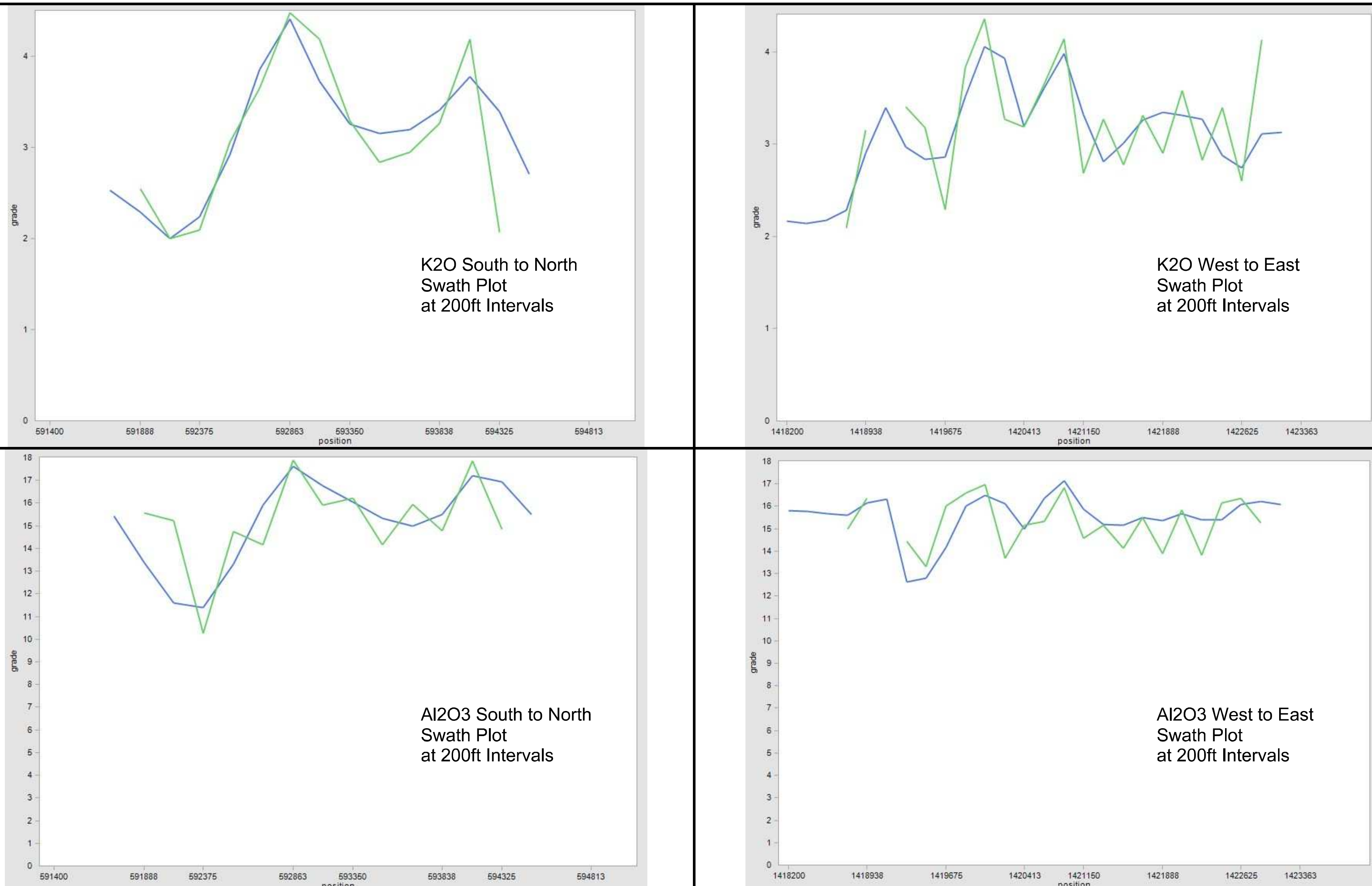
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 VARIOGRAMS OF
ANALYTICAL RESULTS

DATE: 11/2/2016

BY:

Millcreek Mining GROUP



POTASH RIDGE

Drillhole Data

Block Model Data

FIGURE 14.3

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

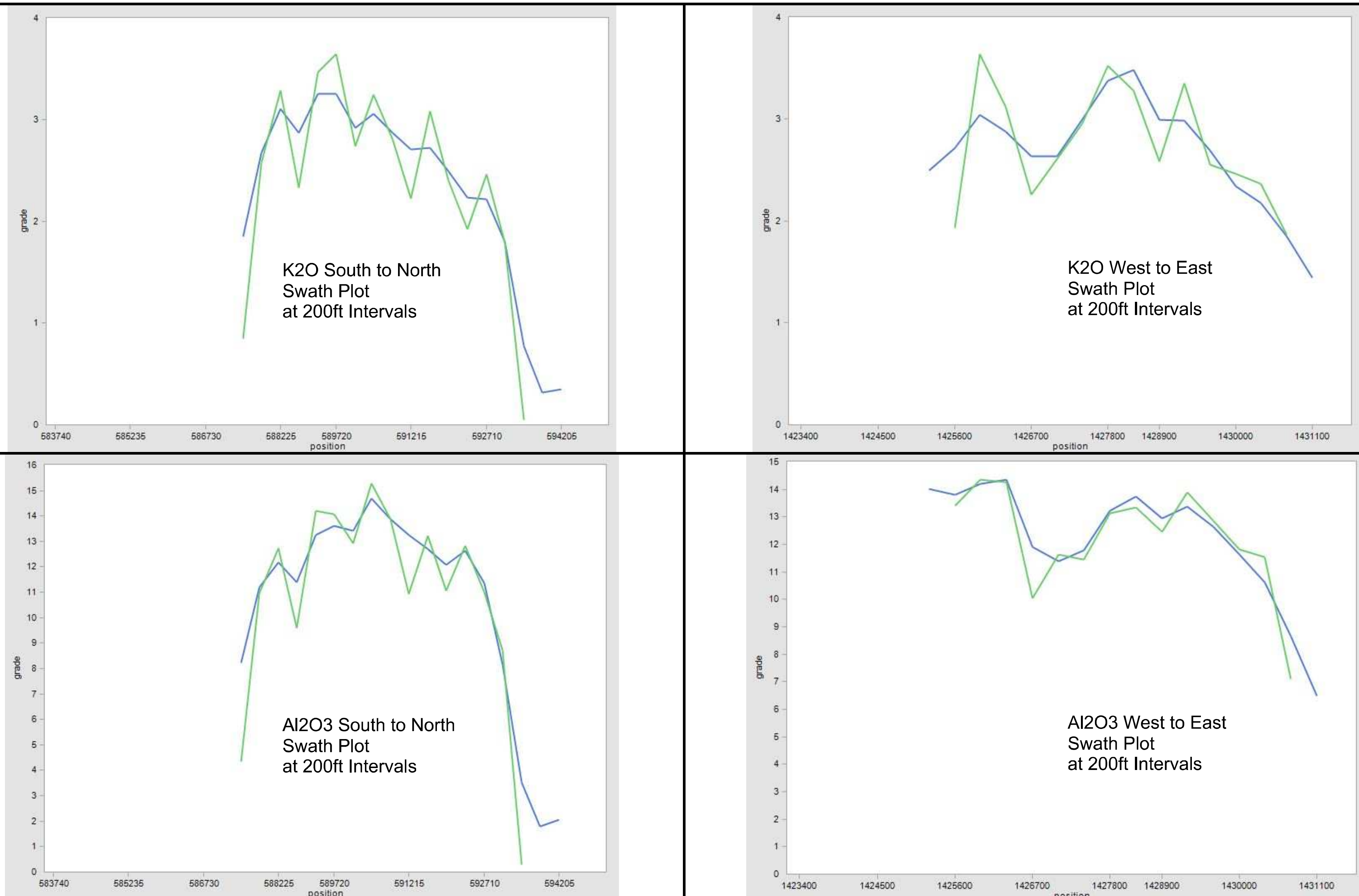
AREA 1 VALIDATION

SWATH PLOTS

DATE: 11/2/2016

BY:

Millcreek Mining GROUP



POTASH RIDGE

Drillhole Data

Block Model Data

FIGURE 14.4

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

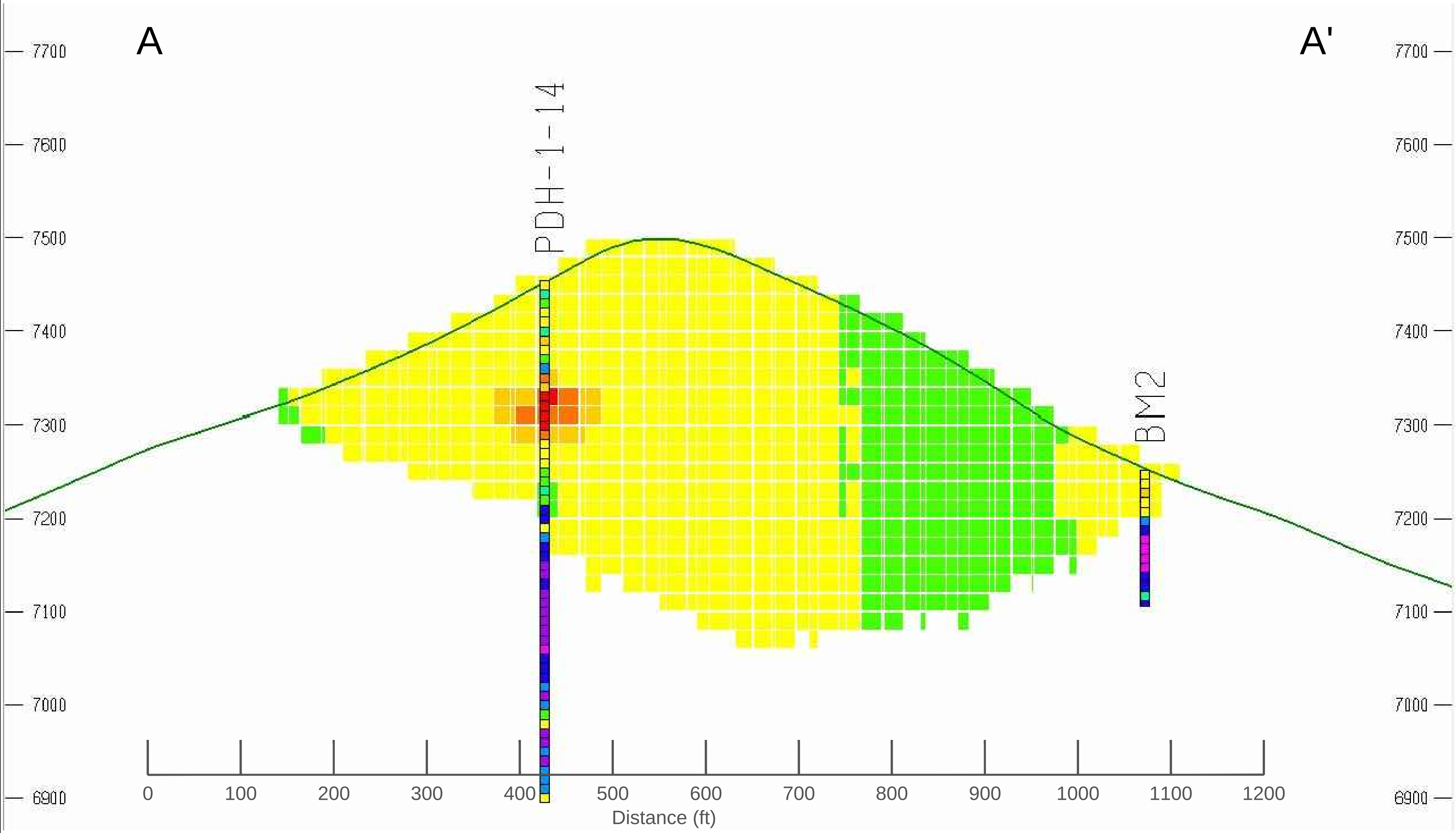
AREA 2 VALIDATION

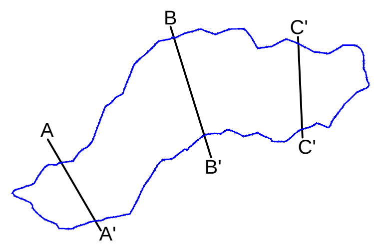
SWATH PLOTS

DATE: 11/2/2016

BY:

Millcreek Mining GROUP





POTASH RIDGE

Horizontal offset for drill holes can be up to 100 feet from section line.

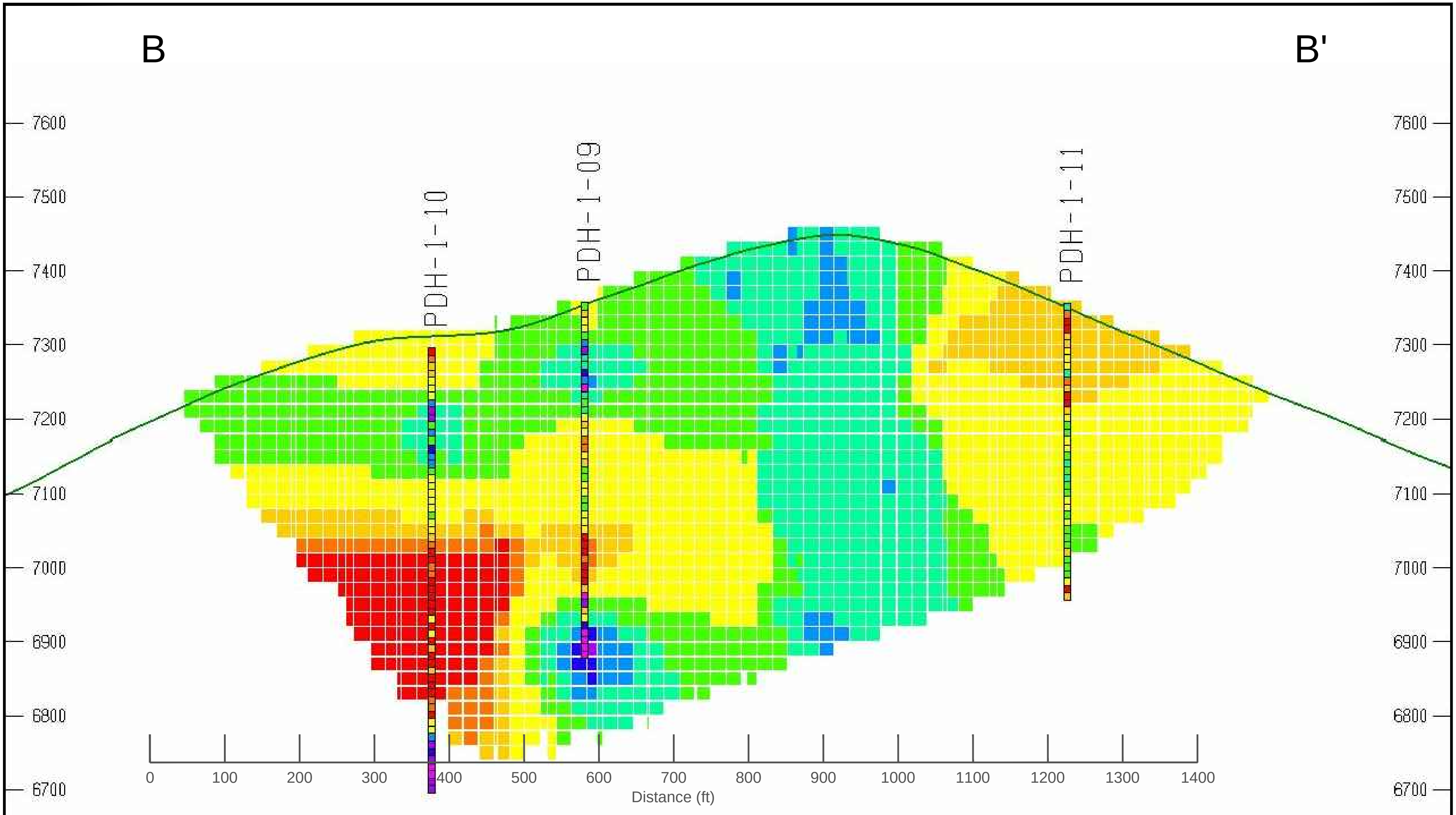
FIGURE 14.5

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 1 BLOCK MODEL
CROSS-SECTION A-A'

DATE: 11/2/2016
BY:

Millcreek Mining GROUP



POTASH RIDGE

Horizontal offset for drill holes can be up to 100 feet from section line.

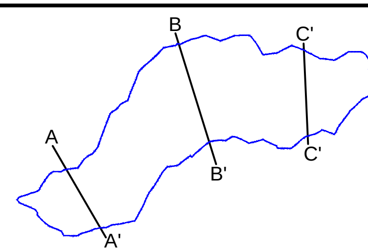


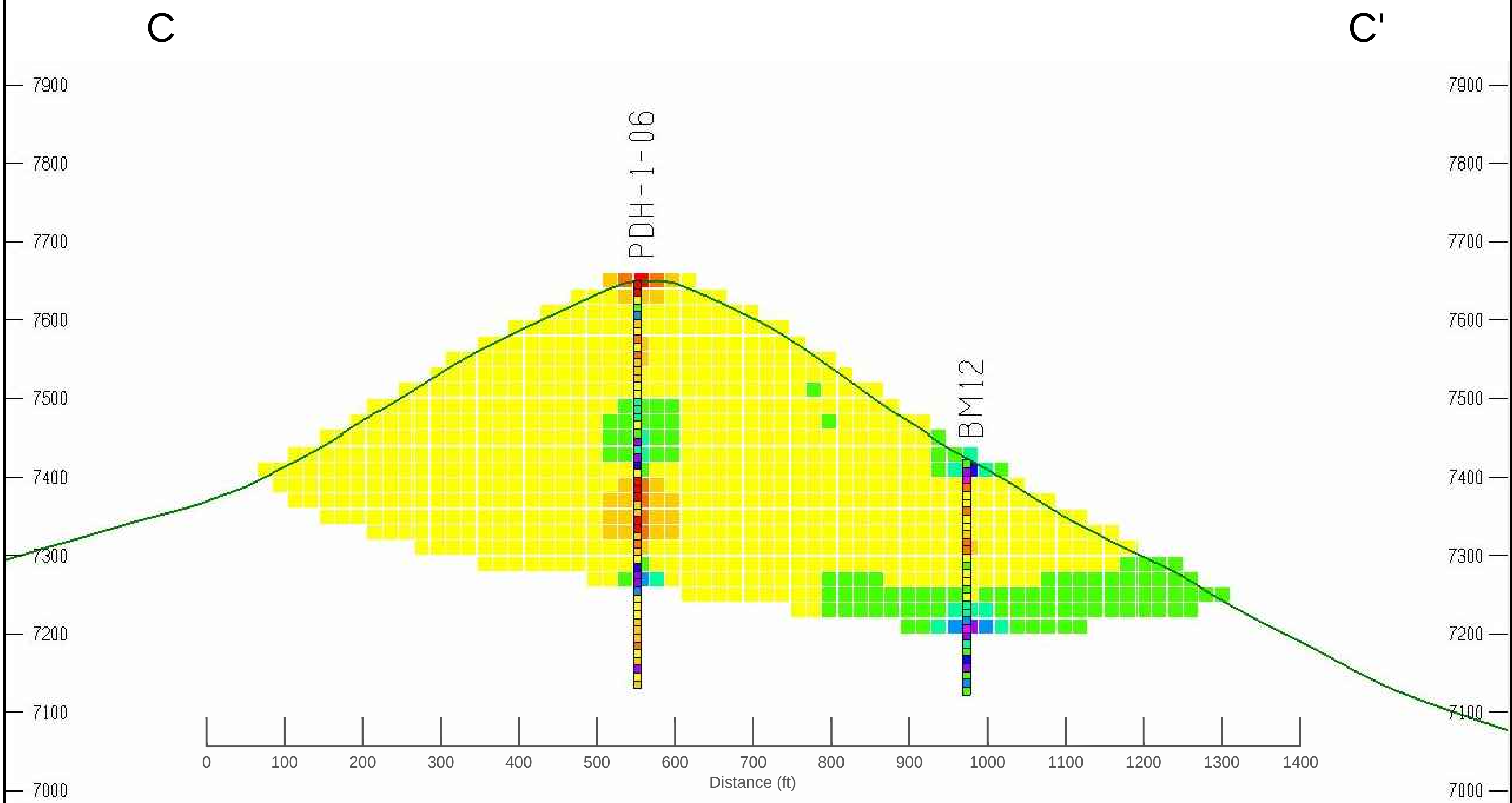
FIGURE 14.6

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 1 BLOCK MODEL
CROSS-SECTION B-B'

DATE: 11/2/2016
BY:

Millcreek Mining GROUP



POTASH RIDGE

K2O%

K2O%
0
0.5
1
1.5
2
2.5
3
3.5
4
4.5
5

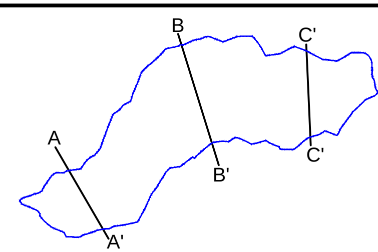


FIGURE 14.7

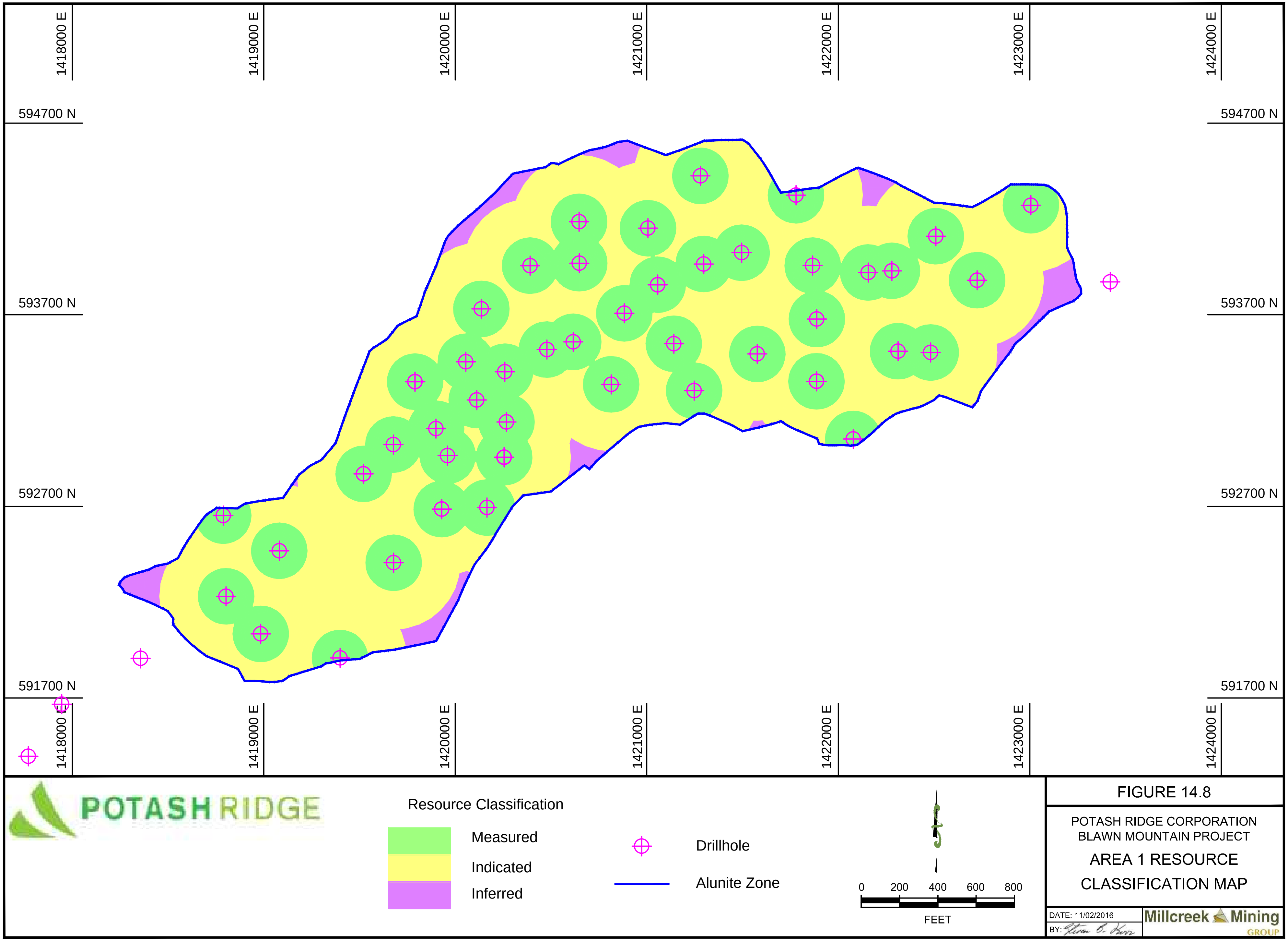
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

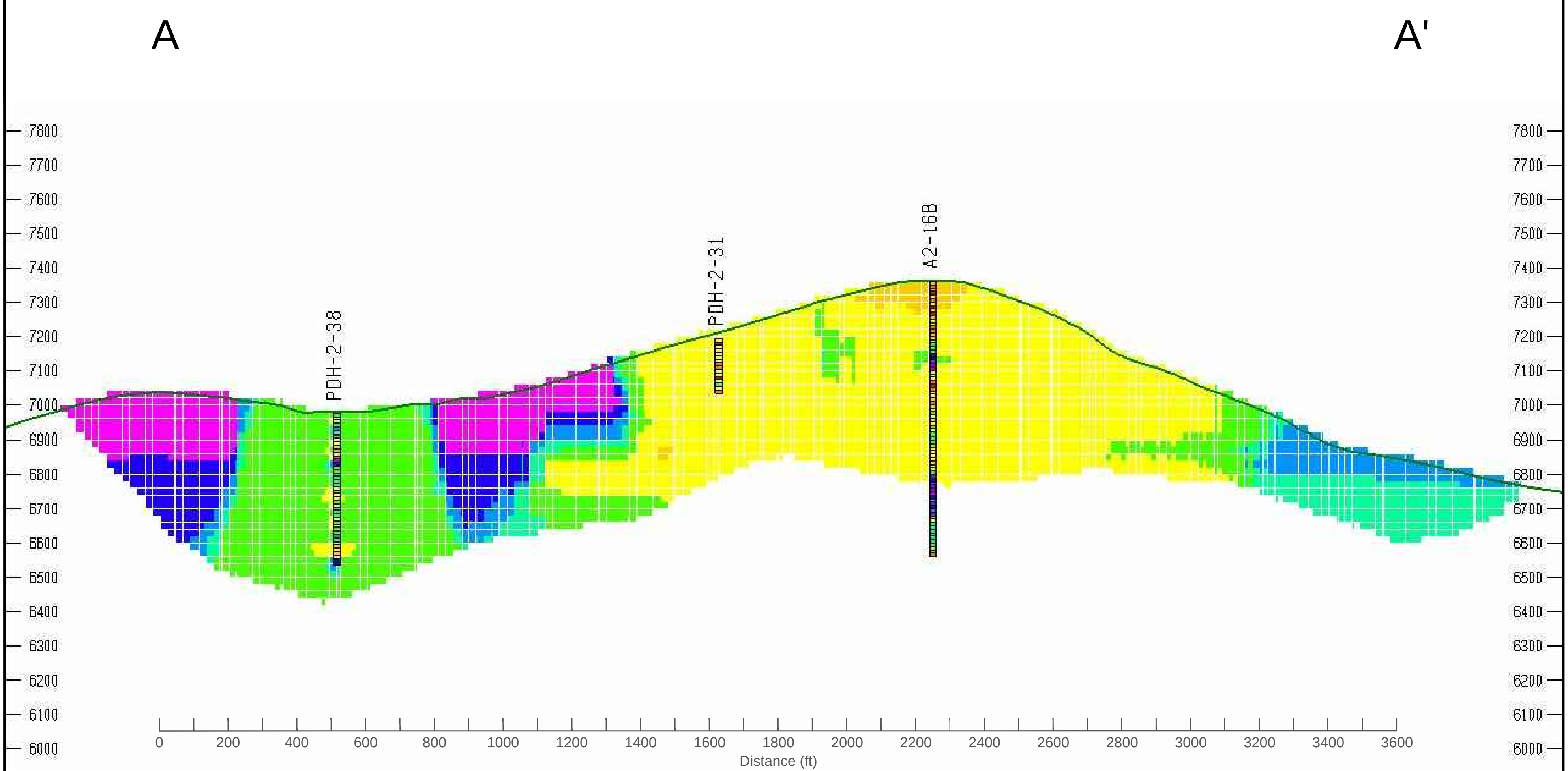
AREA 1 BLOCK MODEL
CROSS-SECTION C-C'

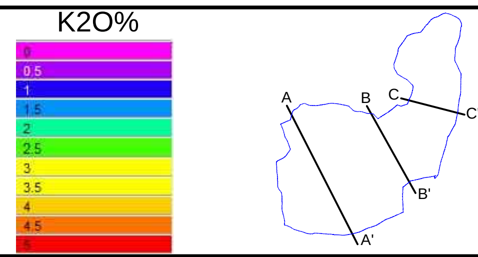
DATE: 11/2/2016
BY:

Millcreek Mining GROUP

Horizontal offset for drill holes can be up to 100 feet from section line.


1418000 E
1419000 E
1420000 E
1421000 E
1422000 E
1423000 E
1424000 E
594700 N
593700 N
592700 N
591700 N
POTASH RIDGE
Resource Classification
Measured
Indicated
Inferred
Drillhole
Alunite Zone
0
200
400
600
800
FEET
FIGURE 14.8
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT
AREA 1 RESOURCE
CLASSIFICATION MAP
DATE: 11/02/2016
BY:
Millcreek Mining
GROUP





Horizontal offset for drill holes can be up to 100 feet from section line.

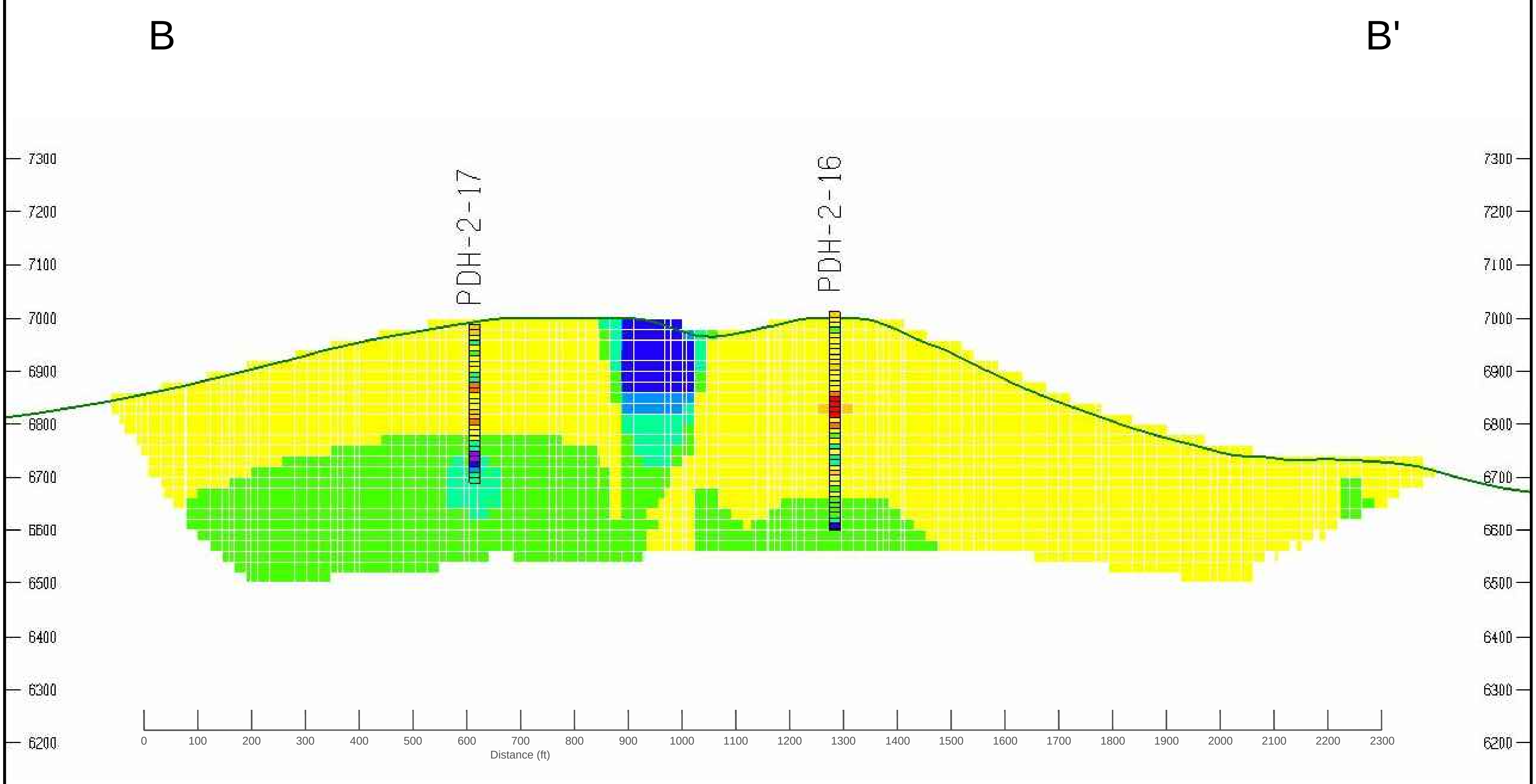
FIGURE 14.9

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 BLOCK MODEL
CROSS-SECTION A-A'

DATE: 11/2/2016
BY:

Millcreek Mining GROUP



POTASH RIDGE

Horizontal offset for drill holes can be up to 100 feet from section line.

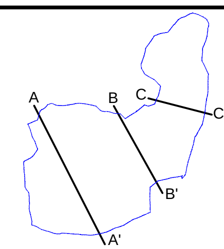


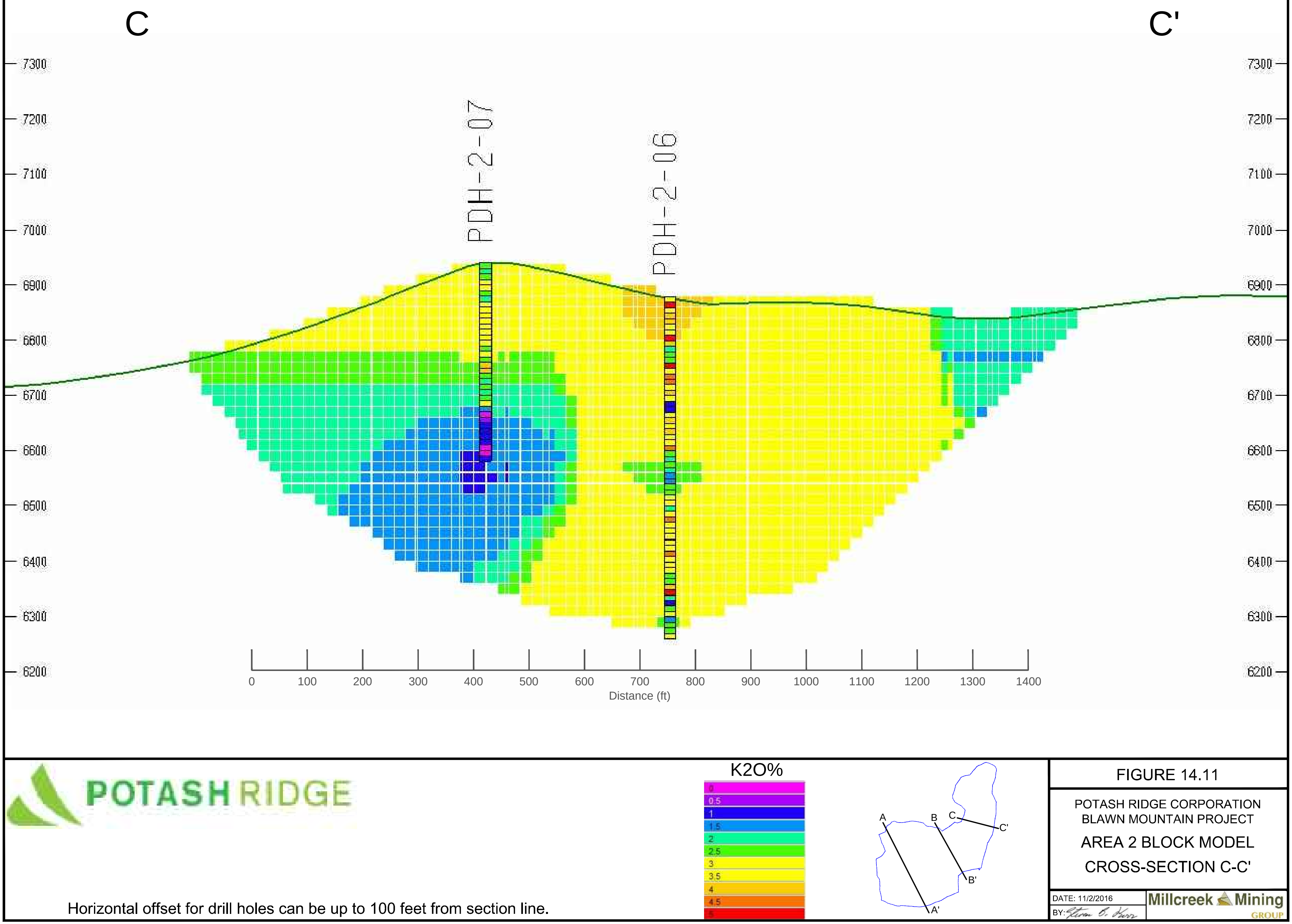
FIGURE 14.10

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 BLOCK MODEL
CROSS-SECTION B-B'

DATE: 11/2/2016
BY:

Millcreek Mining GROUP



POTASH RIDGE

Horizontal offset for drill holes can be up to 100 feet from section line.

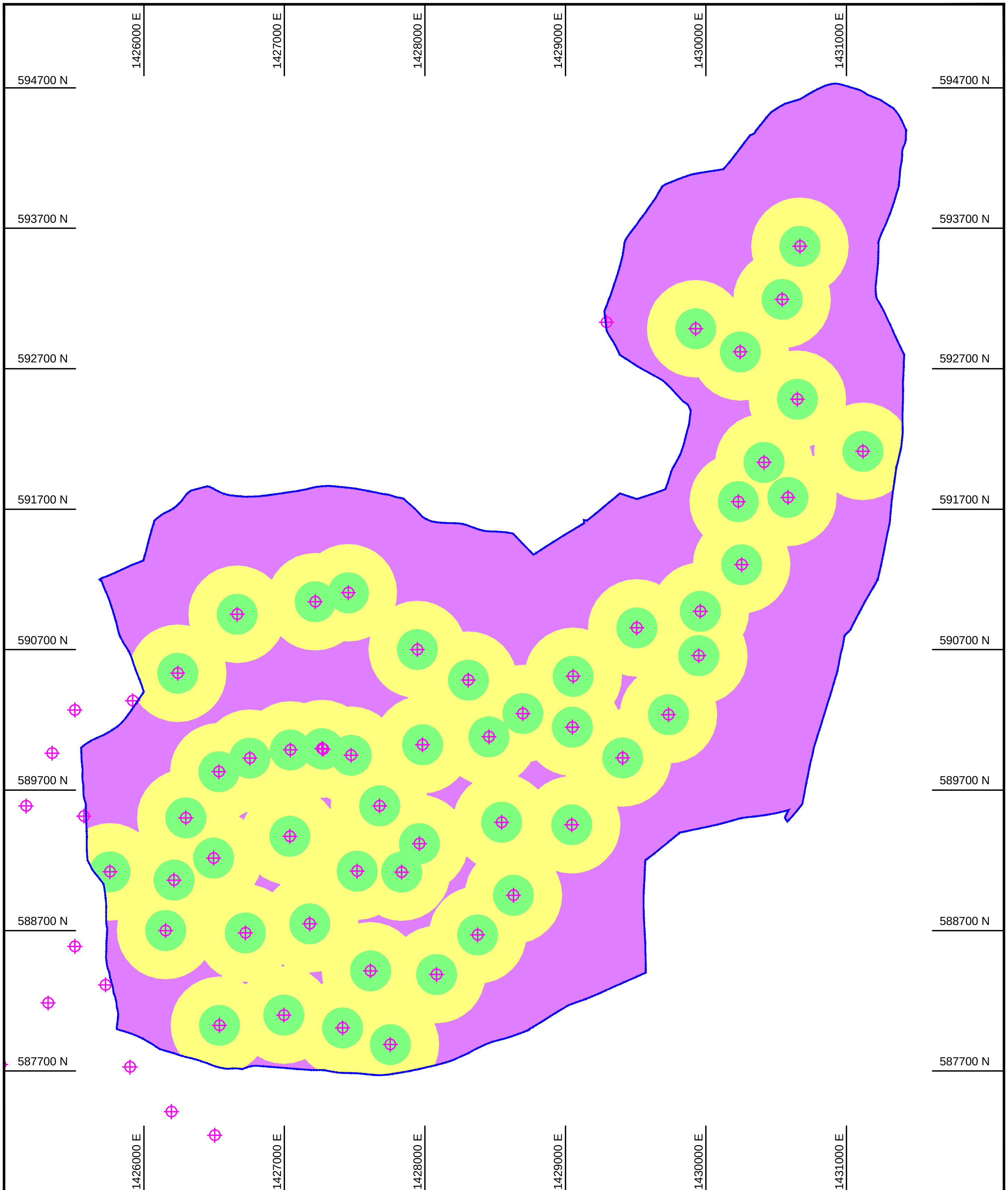
FIGURE 14.11

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 BLOCK MODEL
CROSS-SECTION C-C'

DATE: 11/2/2016
BY:

Millcreek Mining GROUP



POTASH RIDGE

Resource Classification

Measured

Indicated

Inferred

Drillhole

Alunite Zone

0 200 400 600 800

FEET

FIGURE 14.12

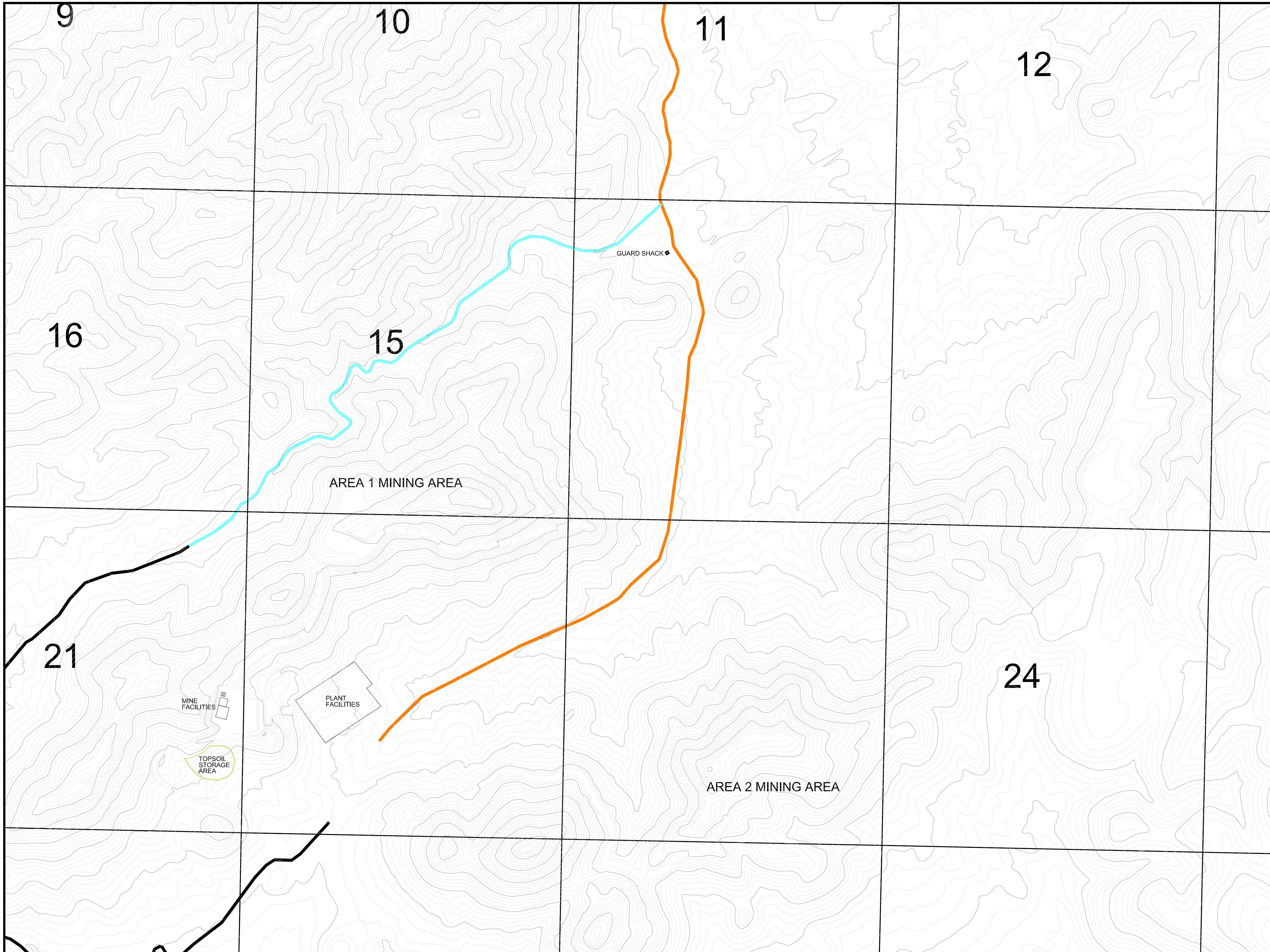
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 RESOURCE
CLASSIFICATION MAP

DATE: 11/02/2016

BY: Steven B. Kerr

Millcreek Mining GROUP



POTASH RIDGE

LEGEND

SITE ACCESS ROAD
BY-PASS ROAD
BEAVER COUNTY ROAD
20 FT TOPOGRAPHY CONTOUR

0 1000 2000 3000
FEET
SCALE: 1:1500

FIGURE 16.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

MINE PROGRESSION

YEAR -1

DATE: 11/18/2016
BY:

Millcreek Mining GROUP

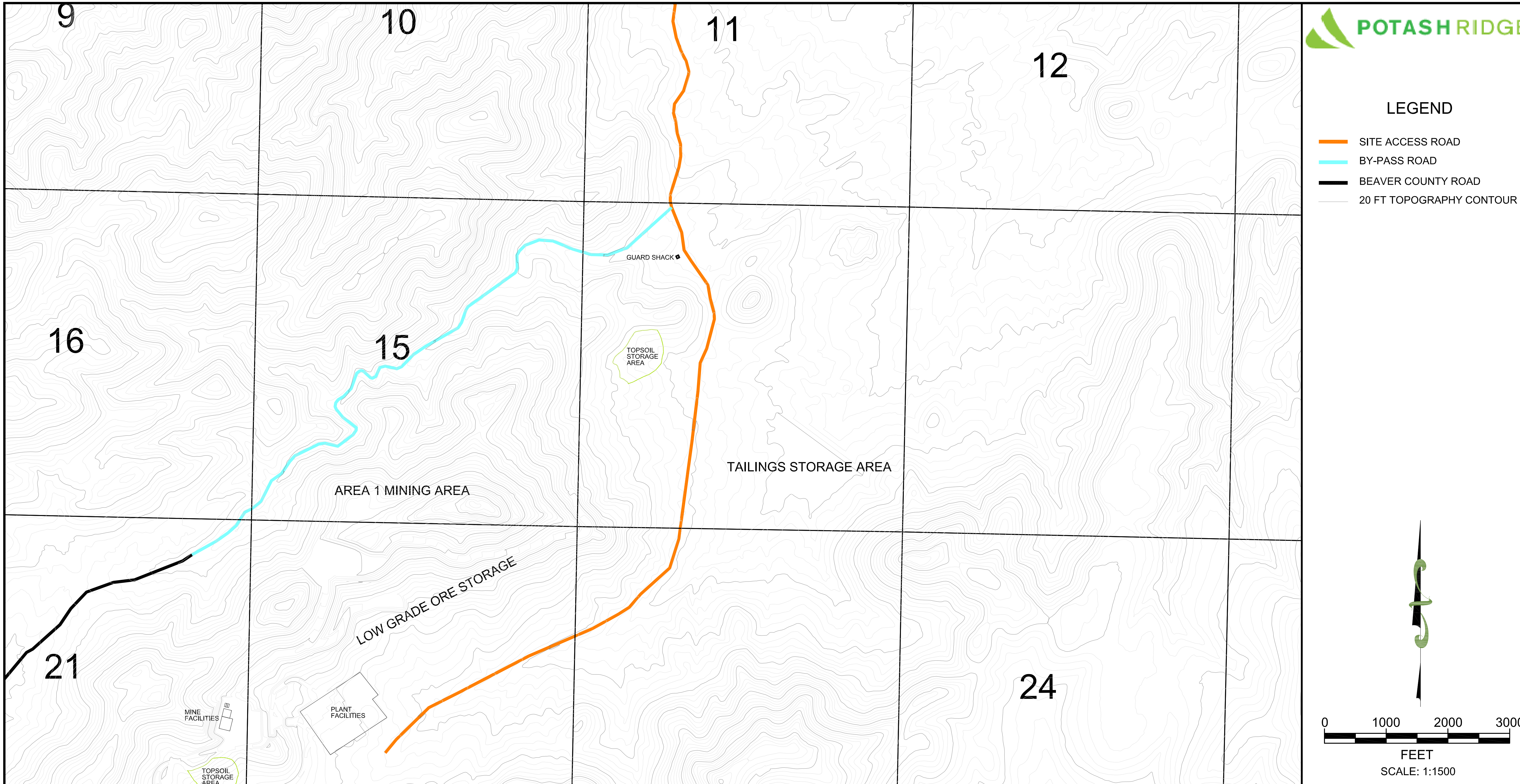


FIGURE 16.2

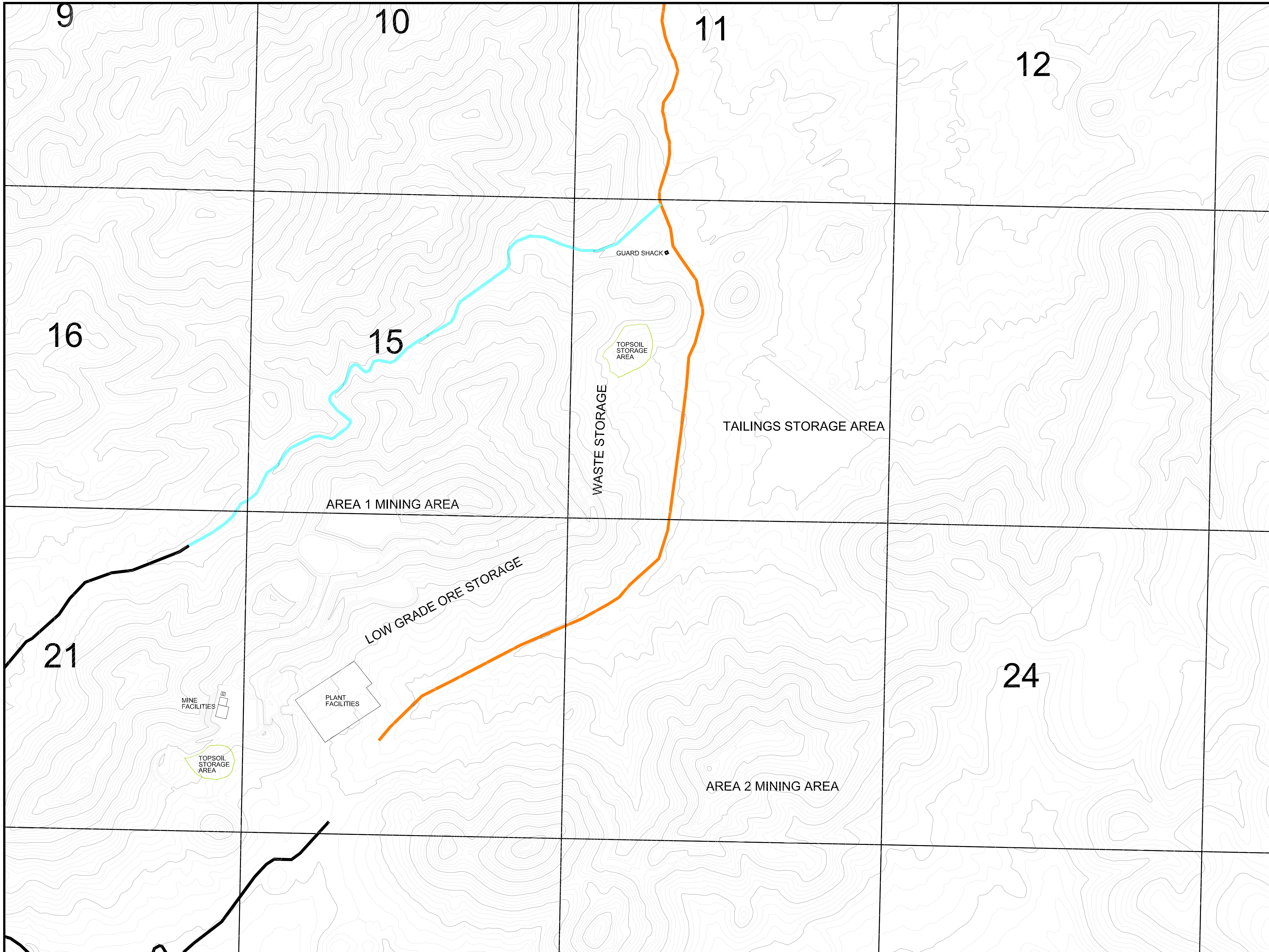
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

MINE PROGRESSION

YEAR 1

DATE: 11/18/2016
BY:

Millcreek Mining GROUP



POTASH RIDGE

LEGEND

SITE ACCESS ROAD
BY-PASS ROAD
BEAVER COUNTY ROAD
20 FT TOPOGRAPHY CONTOUR

0 1000 2000 3000
FEET
SCALE: 1:1500

FIGURE 16.3

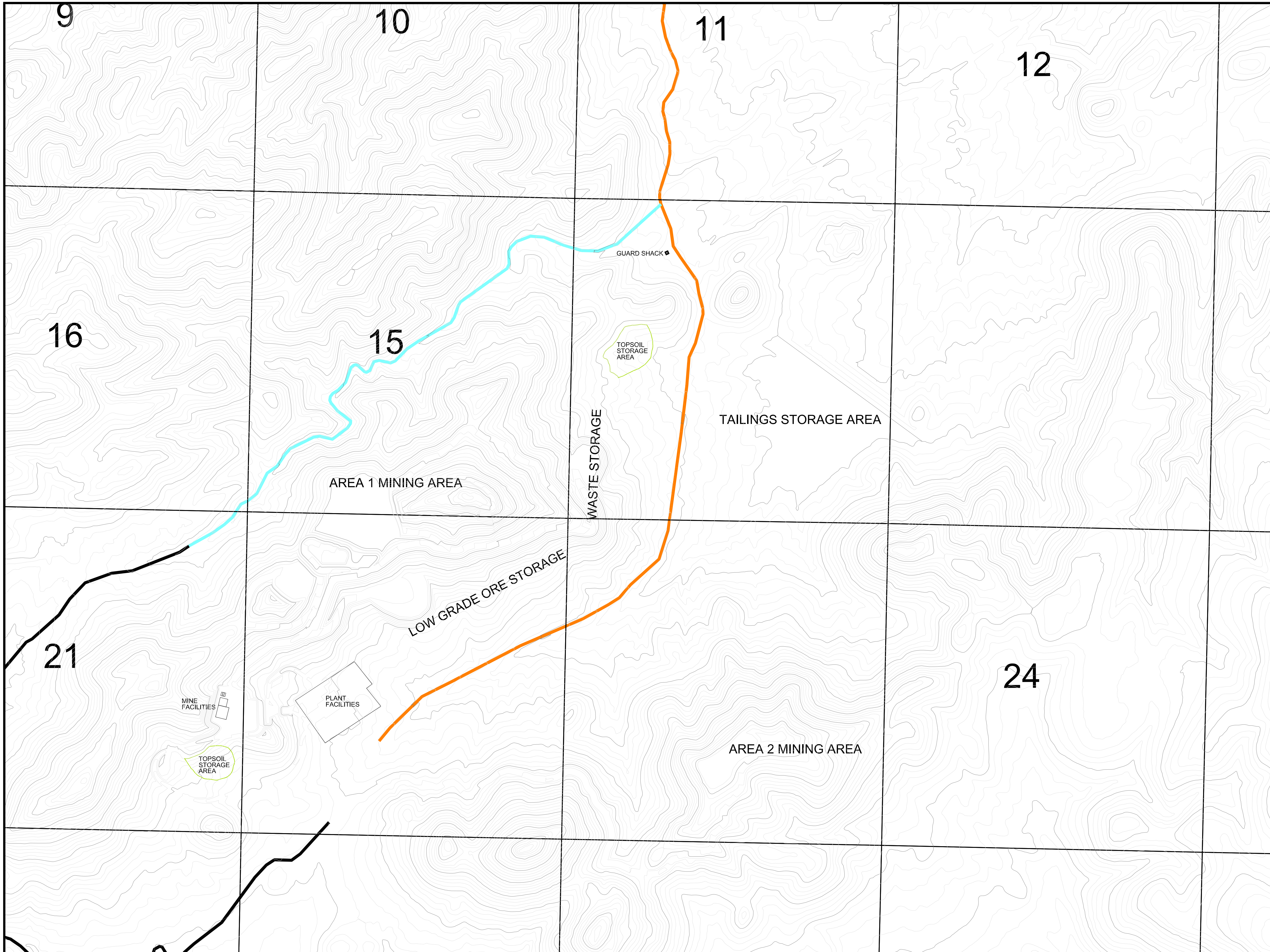
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

MINE PROGRESSION

YEAR 2

DATE: 11/18/2016
BY:

Millcreek Mining GROUP



POTASH RIDGE

LEGEND

- SITE ACCESS ROAD
- BY-PASS ROAD
- BEAVER COUNTY ROAD
- 20 FT TOPOGRAPHY CONTOUR

FIGURE 16.4

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

MINE PROGRESSION

YEAR 3

DATE: 11/18/2016
BY:

Millcreek Mining GROUP

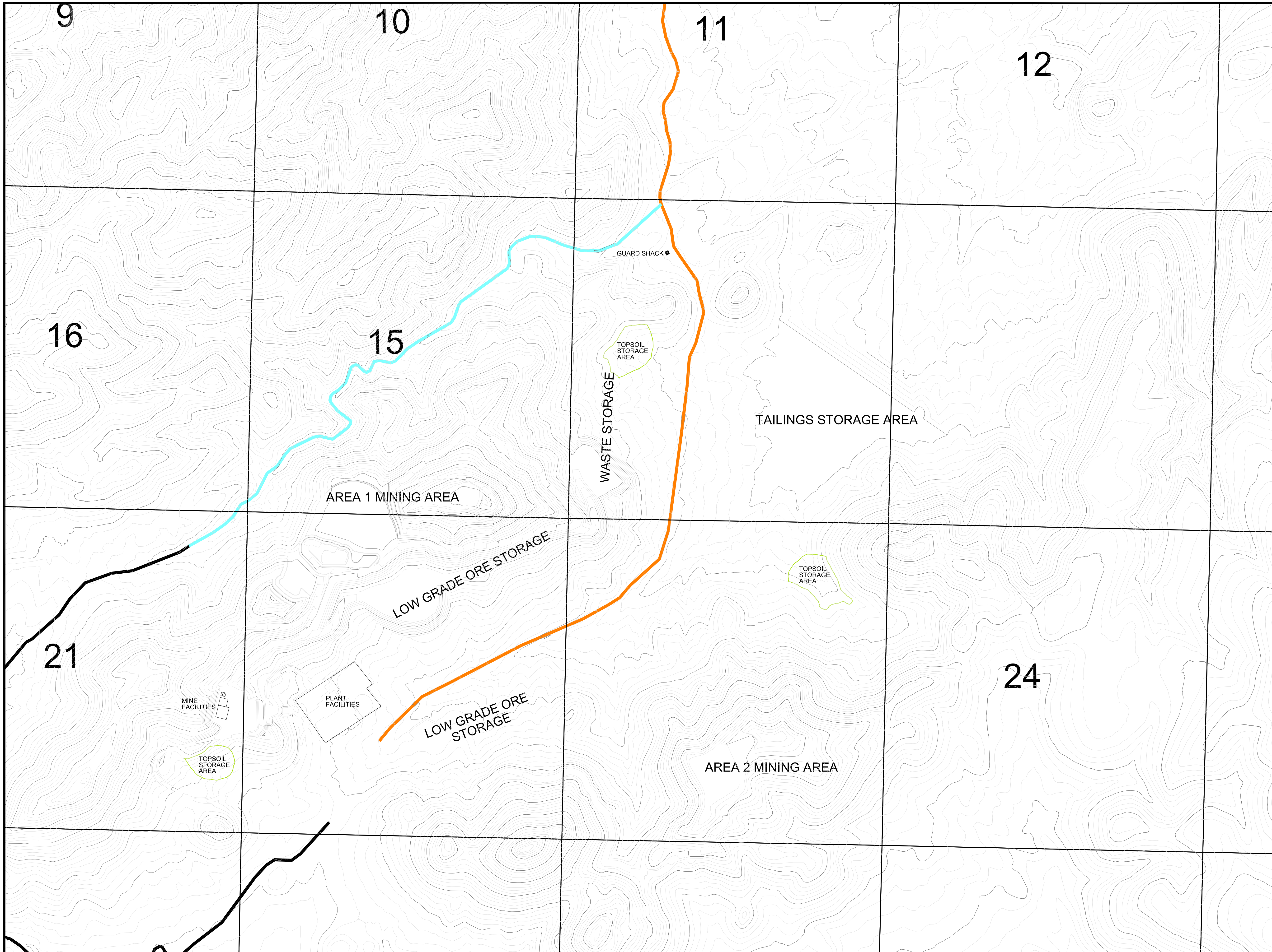


FIGURE 16.5

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

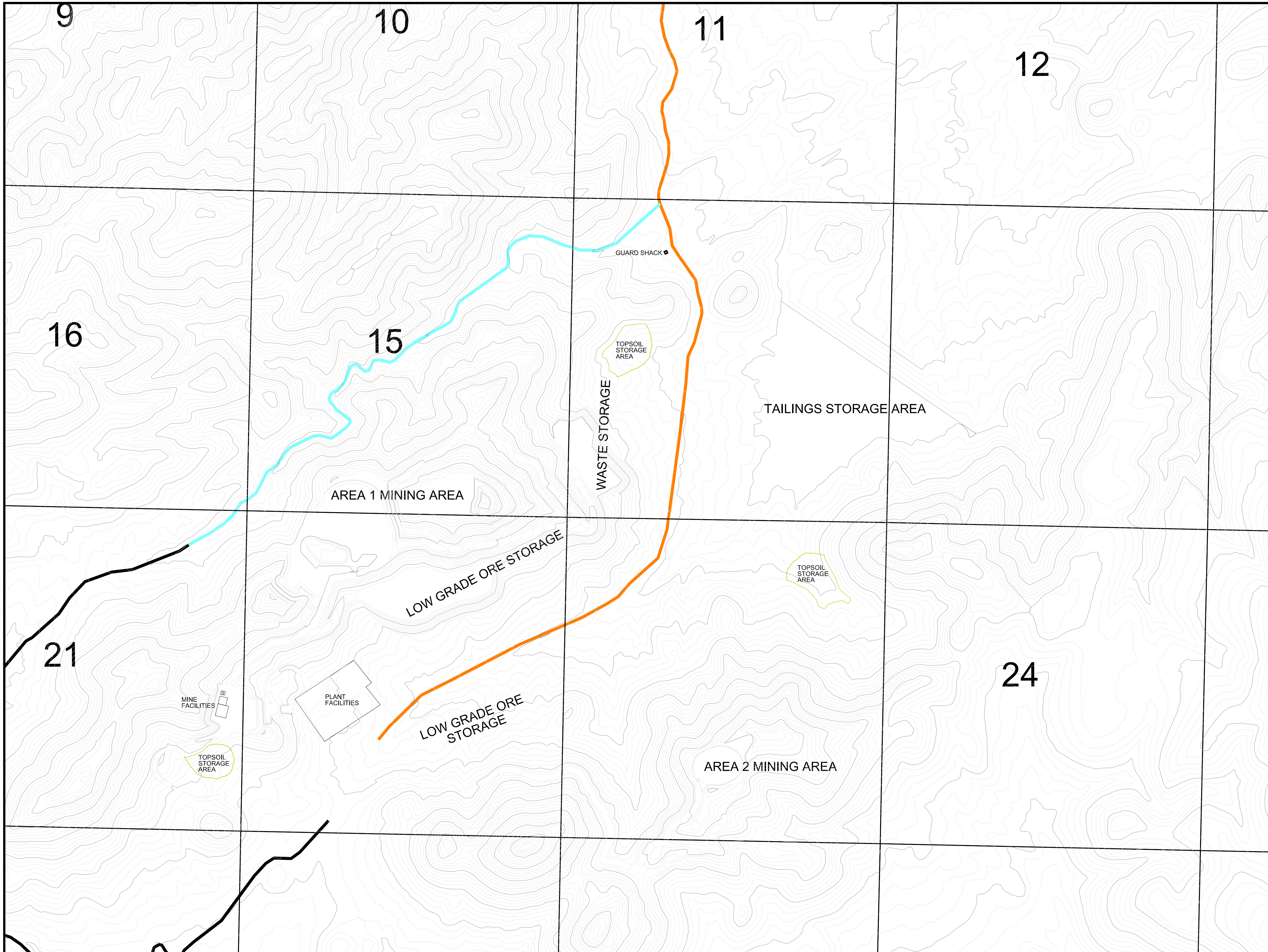
MINE PROGRESSION

YEAR 4

DATE: 11/18/2016

BY:

Millcreek Mining GROUP



POTASH RIDGE

LEGEND

SITE ACCESS ROAD
BY-PASS ROAD
BEAVER COUNTY ROAD
20 FT TOPOGRAPHY CONTOUR

0 1000 2000 3000
FEET
SCALE: 1:1500

FIGURE 16.6

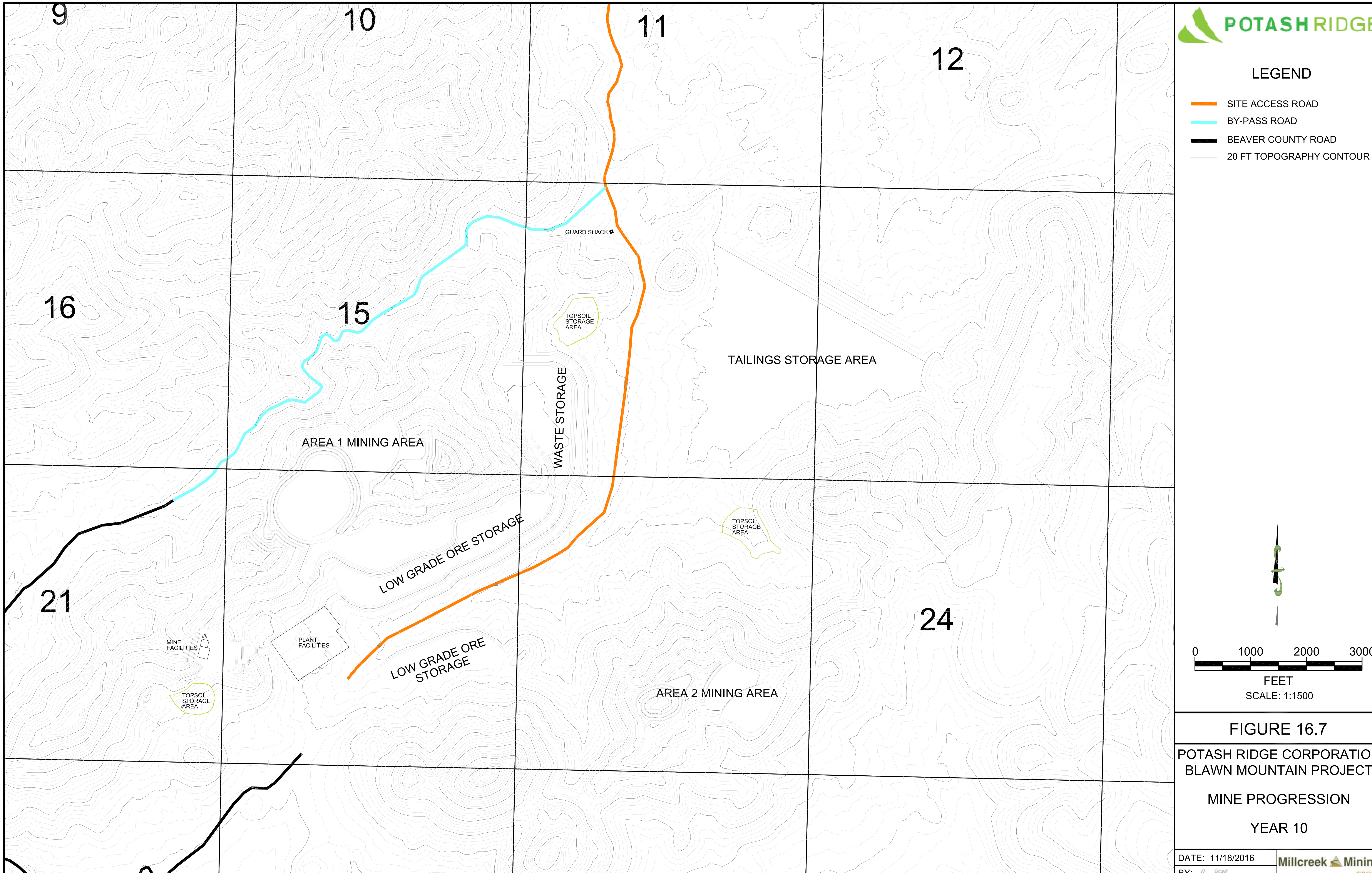
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

MINE PROGRESSION

YEAR 5

DATE: 11/18/2016
BY:

Millcreek Mining GROUP


9
10
11
12
16
15
21
24
GUARD SHACK
TOPSOIL STORAGE AREA
TAILINGS STORAGE AREA
WASTE STORAGE
AREA 1 MINING AREA
LOW GRADE ORE STORAGE
TOPSOIL STORAGE AREA
MINE FACILITIES
PLANT FACILITIES
LOW GRADE ORE STORAGE
TOPSOIL STORAGE AREA
AREA 2 MINING AREA
POTASH RIDGE
LEGEND
SITE ACCESS ROAD
BY-PASS ROAD
BEAVER COUNTY ROAD
20 FT TOPOGRAPHY CONTOUR
0
1000
2000
3000
FEET
SCALE: 1:1500
FIGURE 16.7
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT
MINE PROGRESSION
YEAR 10
DATE: 11/18/2016
BY:
Millcreek Mining GROUP

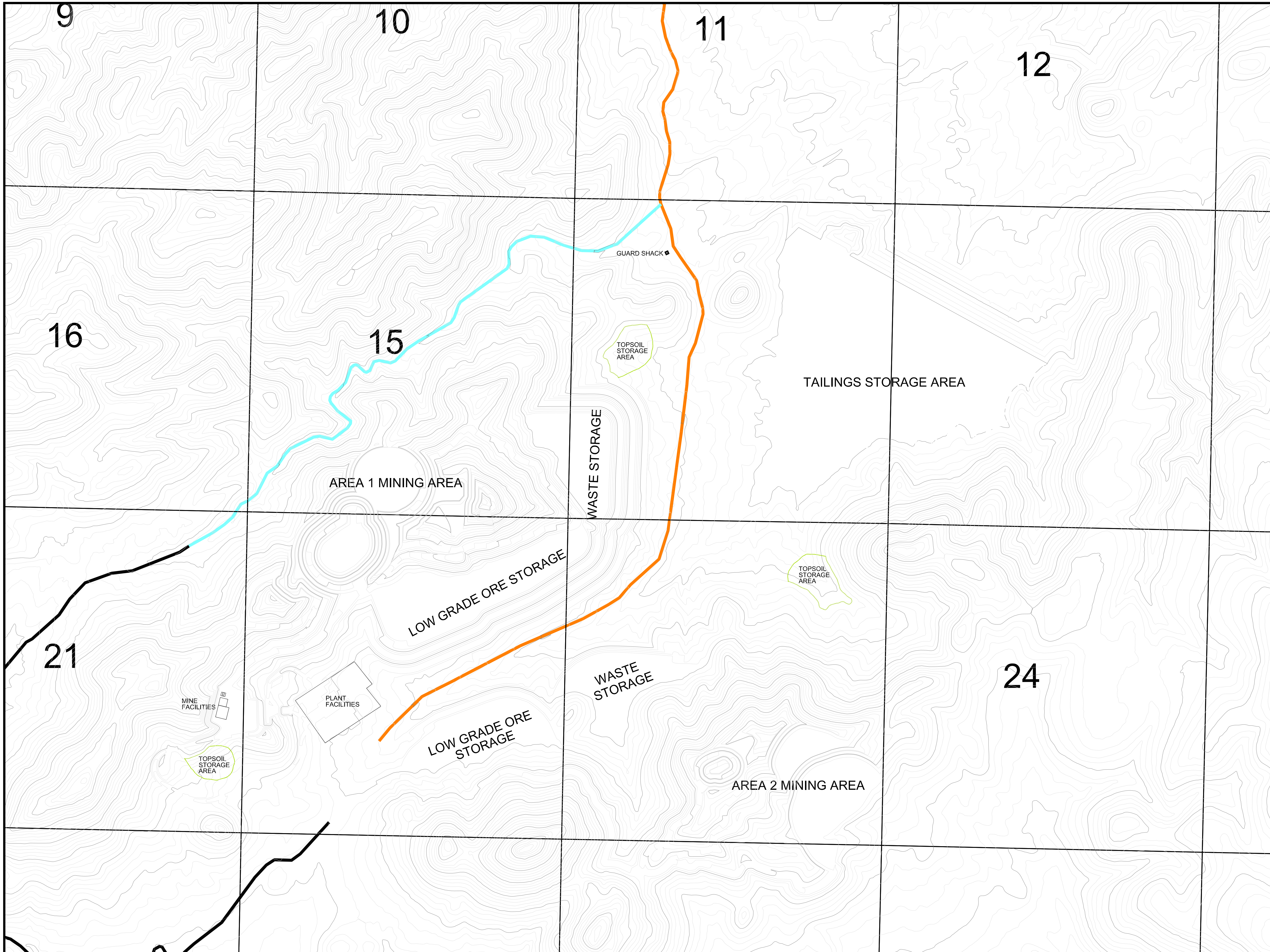


FIGURE 16.8

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

MINE PROGRESSION

YEAR 15

DATE: 11/18/2016

BY:

Millcreek Mining GROUP

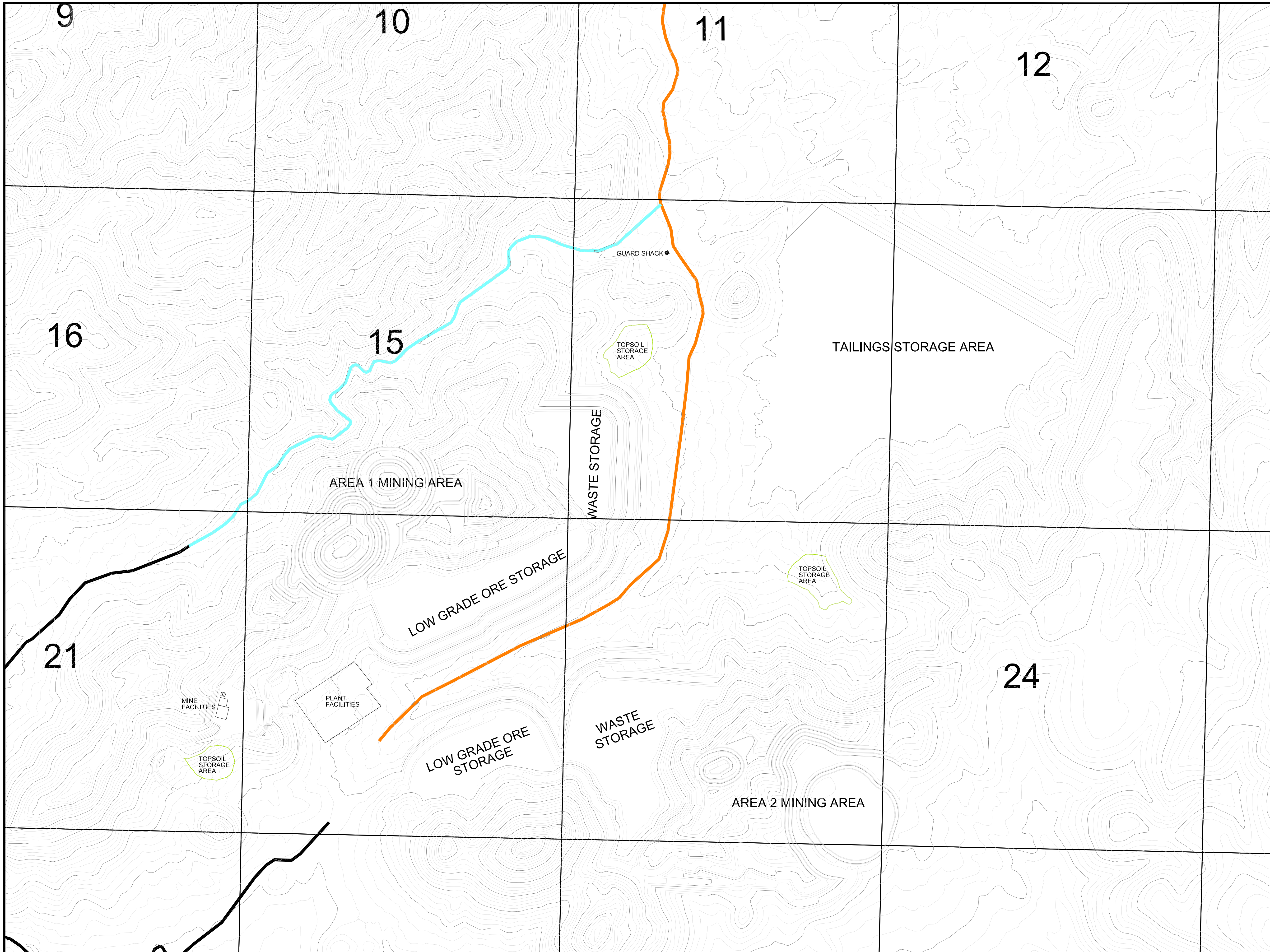


FIGURE 16.9

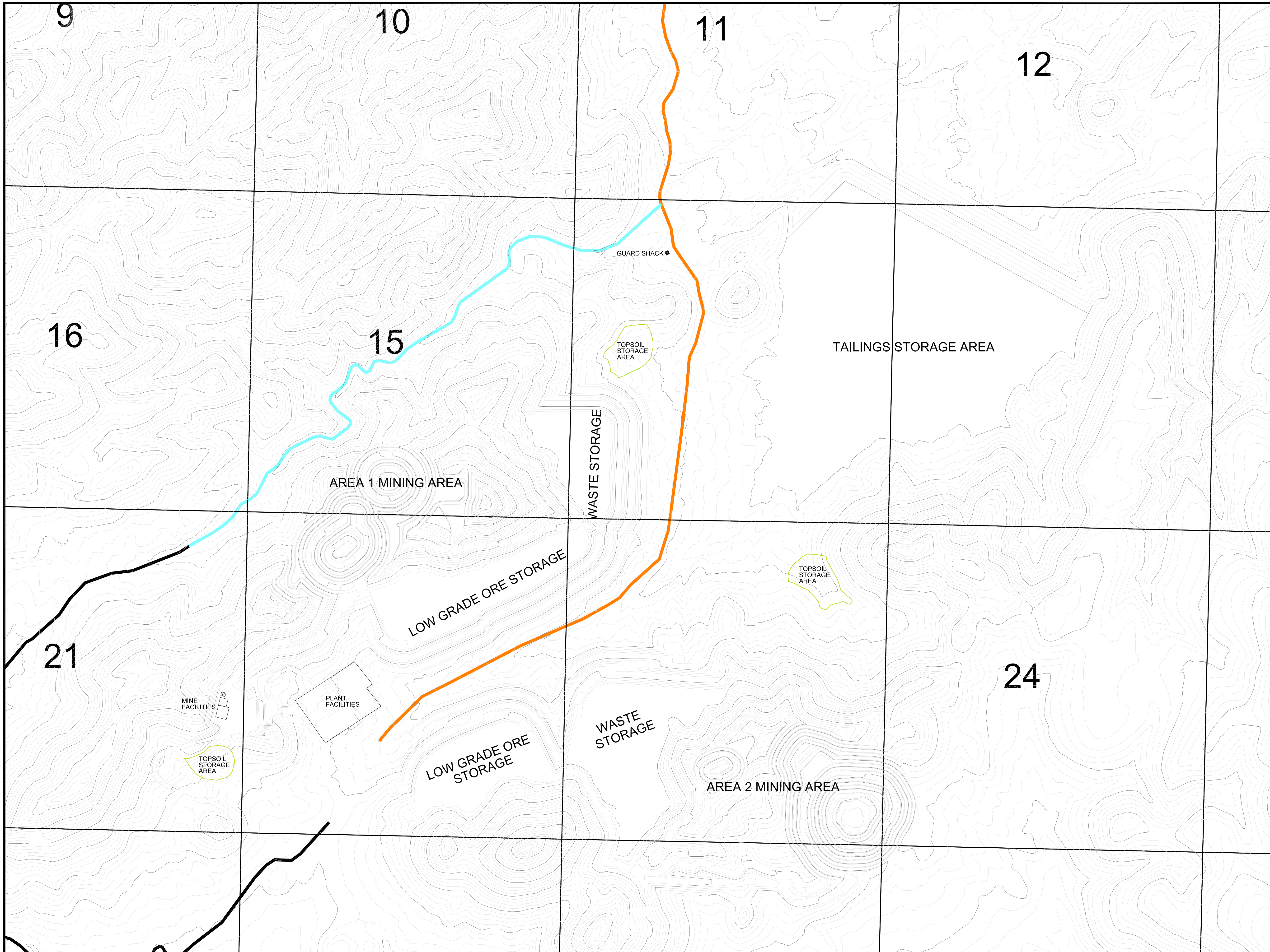
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

MINE PROGRESSION

YEAR 20

DATE: 11/18/2016
BY:

Millcreek Mining GROUP



POTASH RIDGE

LEGEND

SITE ACCESS ROAD
BY-PASS ROAD
BEAVER COUNTY ROAD
20 FT TOPOGRAPHY CONTOUR

0 1000 2000 3000
FEET
SCALE: 1:1500

FIGURE 16.10

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

MINE PROGRESSION

YEAR 28

DATE: 11/18/2016
BY:

Millcreek Mining GROUP

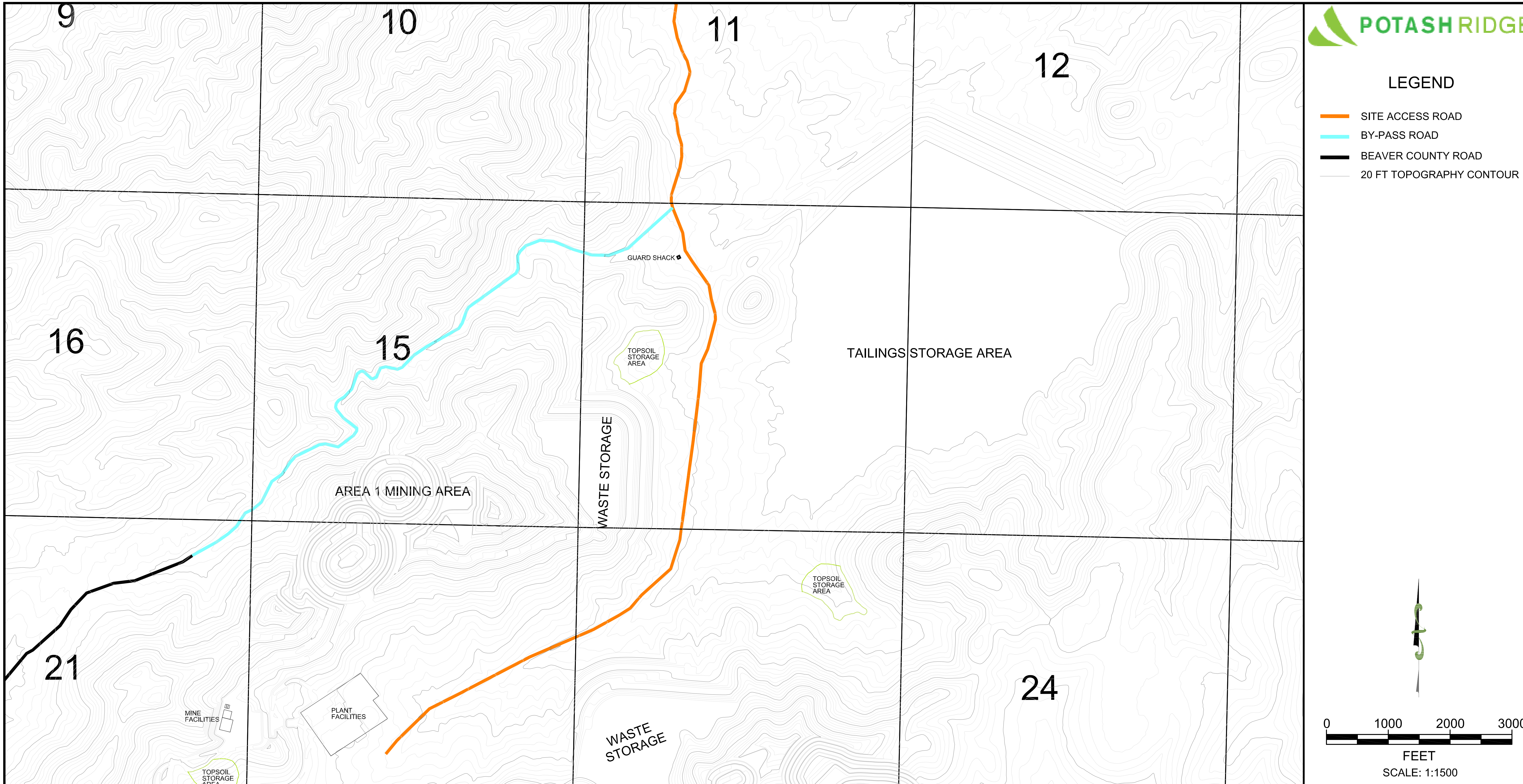


FIGURE 16.11

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

MINE PROGRESSION

YEAR 45

DATE: 11/18/2016
BY:

Millcreek Mining GROUP

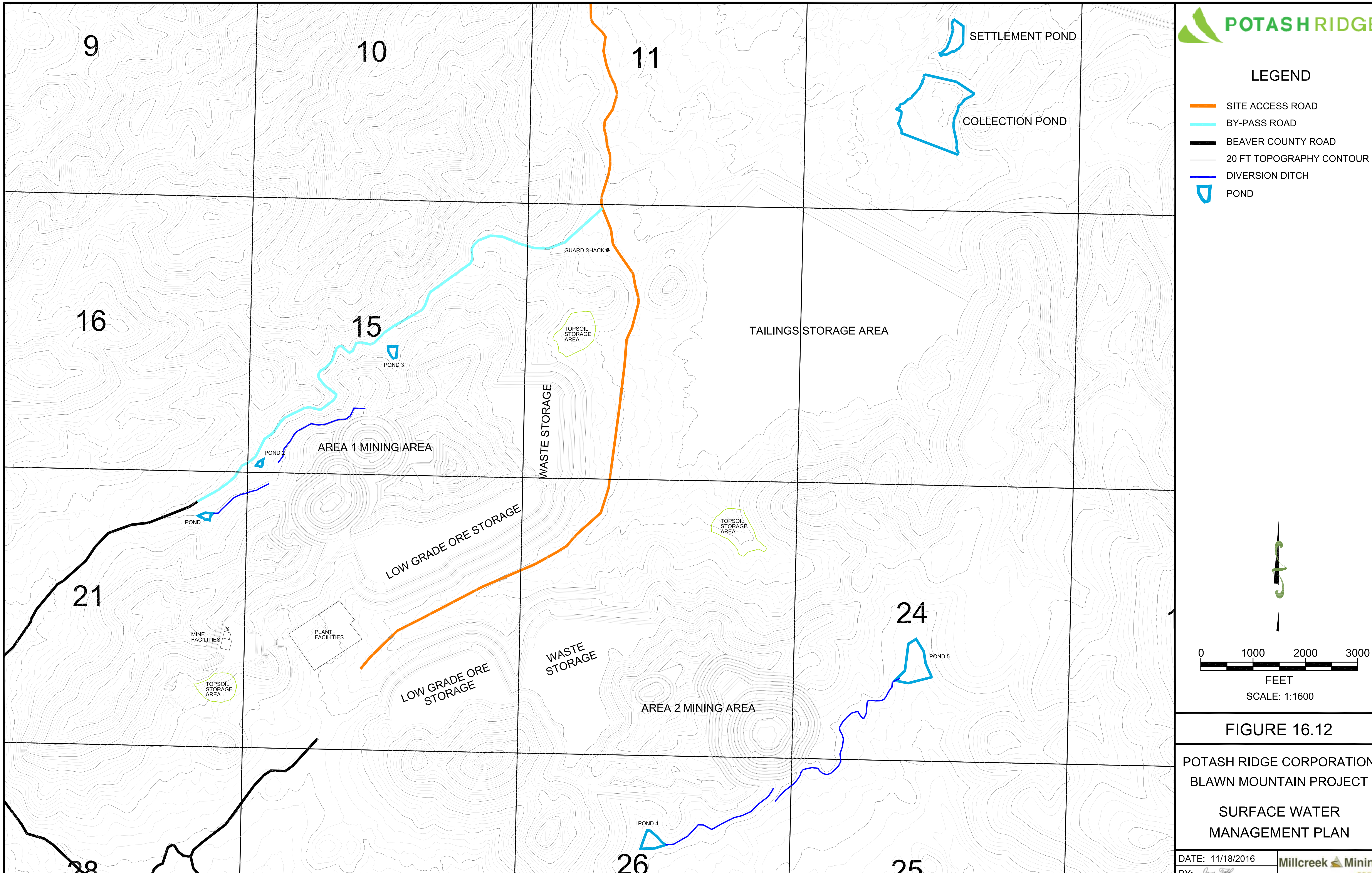


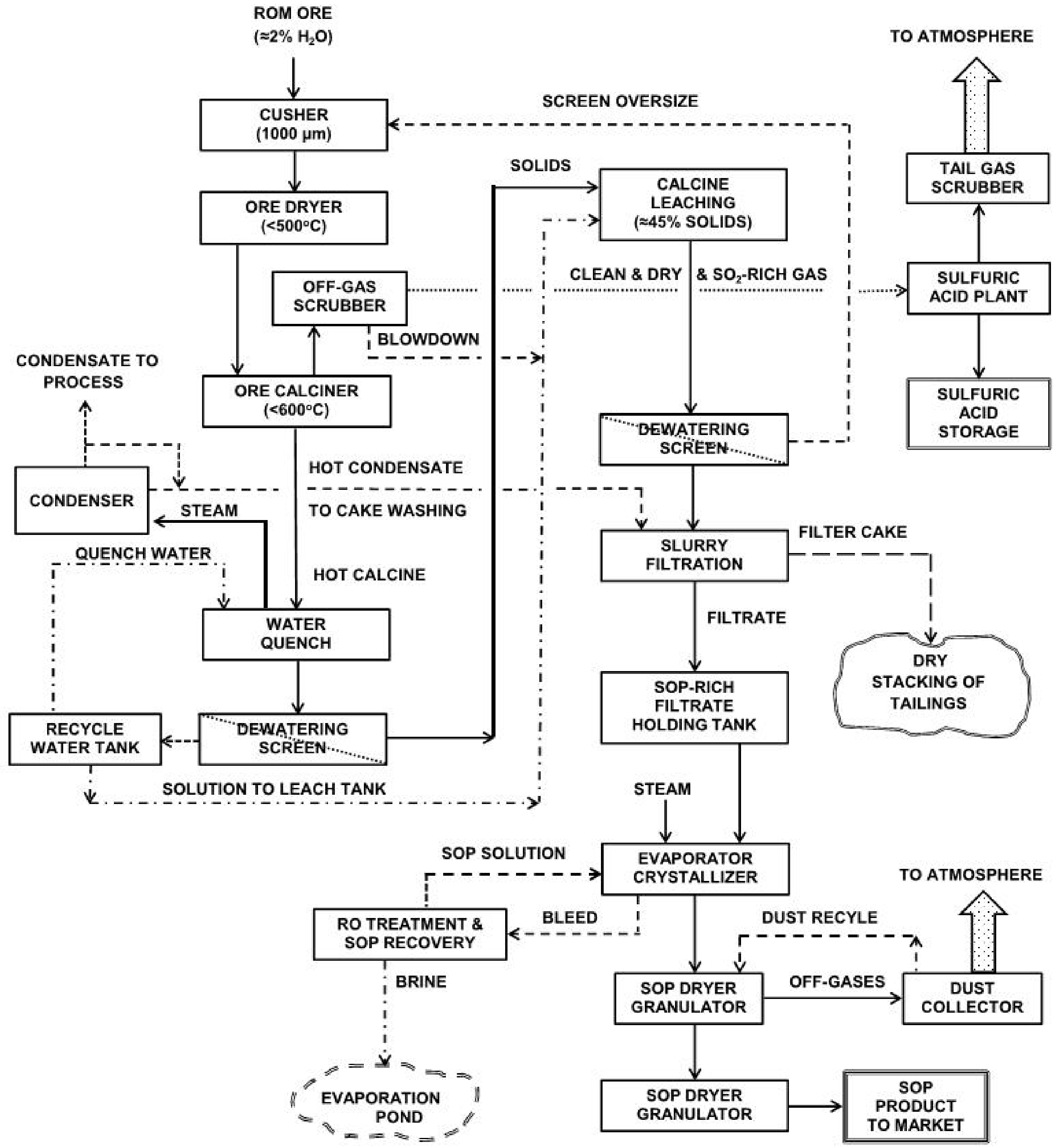
FIGURE 16.12

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

SURFACE WATER MANAGEMENT PLAN

DATE: 11/18/2016
BY:

Millcreek Mining GROUP



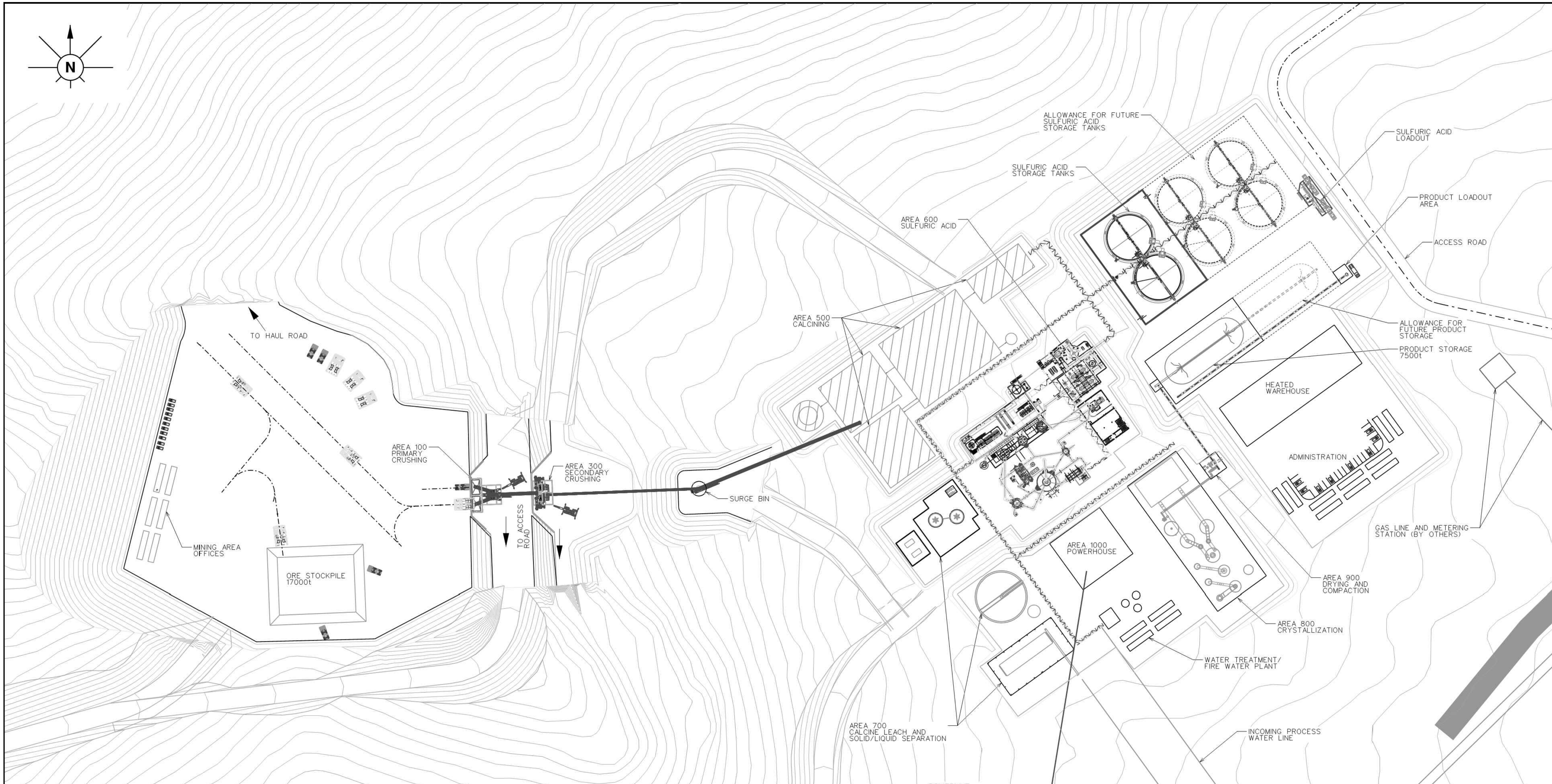
POTASH RIDGE

FIGURE 17.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

MAJOR UNIT OPERATIONS
BLOCK FLOW DIAGRAM

DATE: 01/05/2017
BY:



POTASH RIDGE

Scale in Feet

FIGURE 17.2

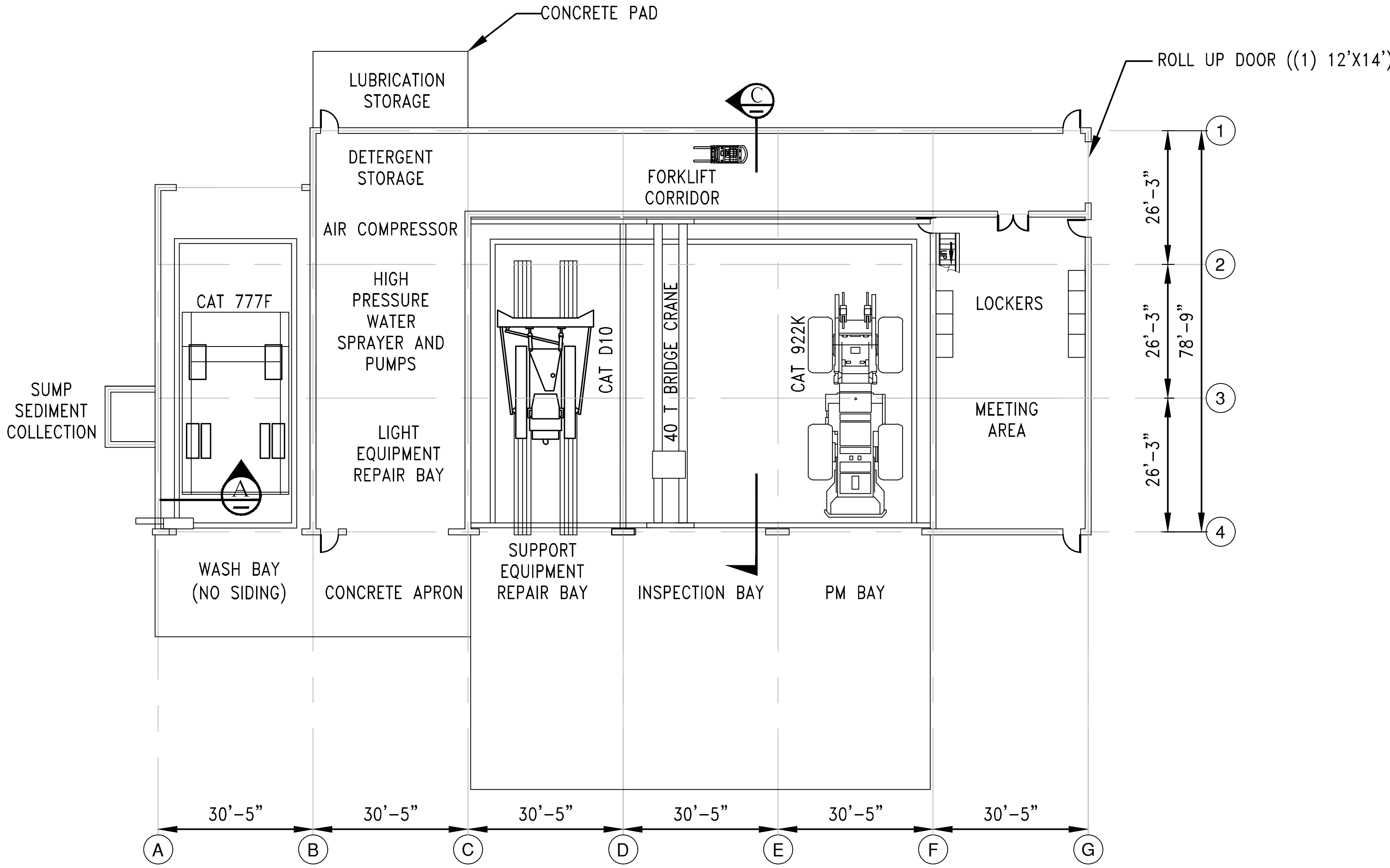
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

GENERAL ARRANGEMENT

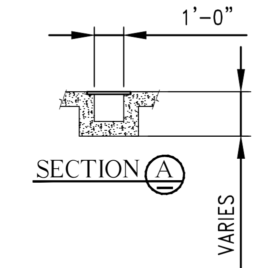
PROCESS AREA SITE PLAN

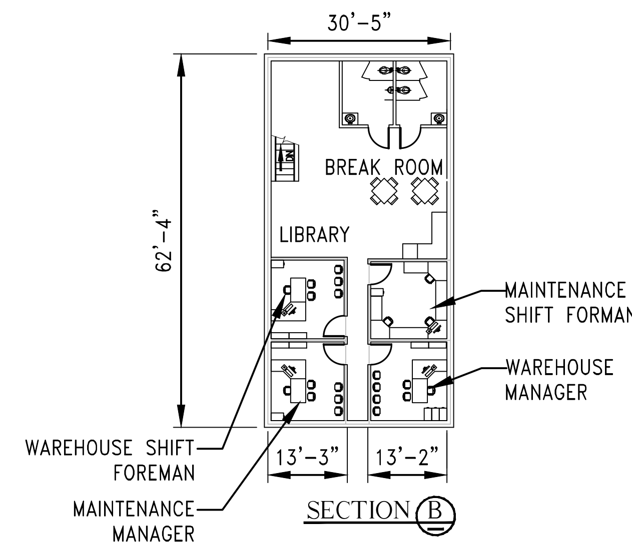
DATE: 01/06/2017

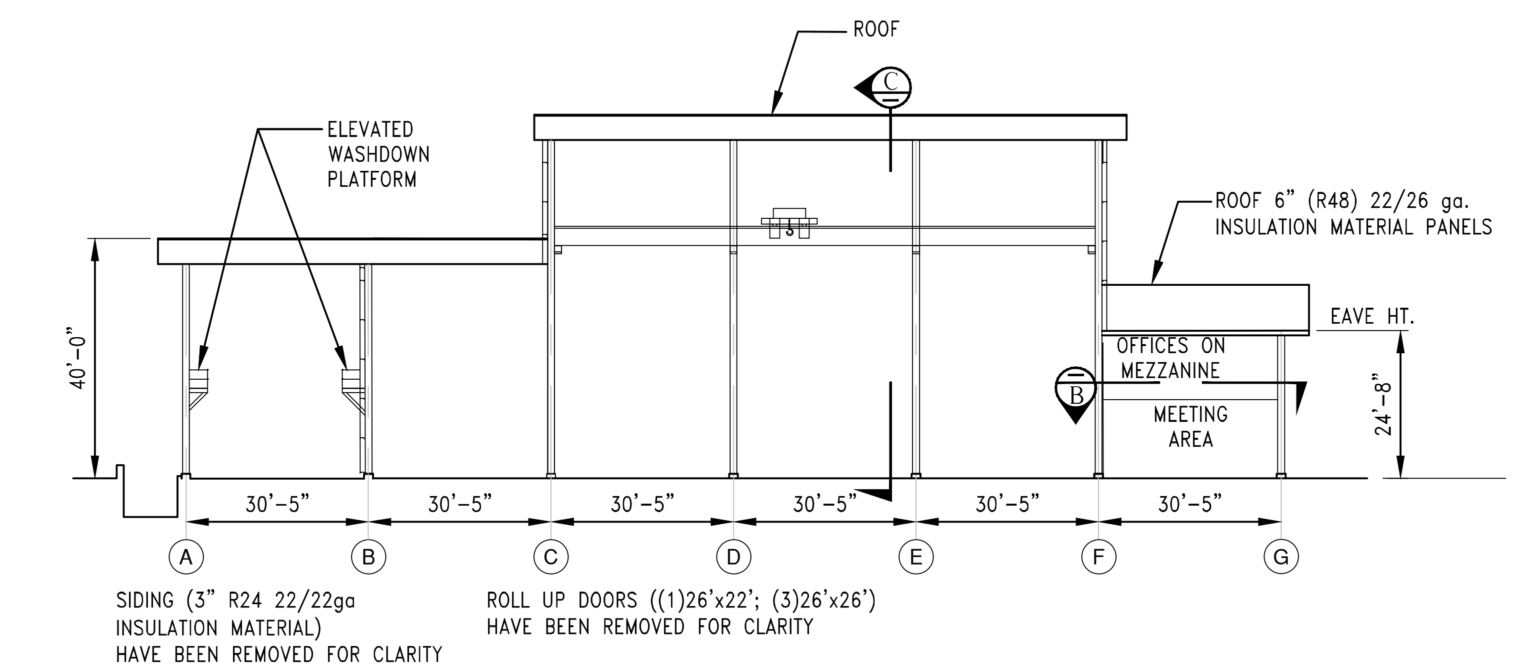
BY:



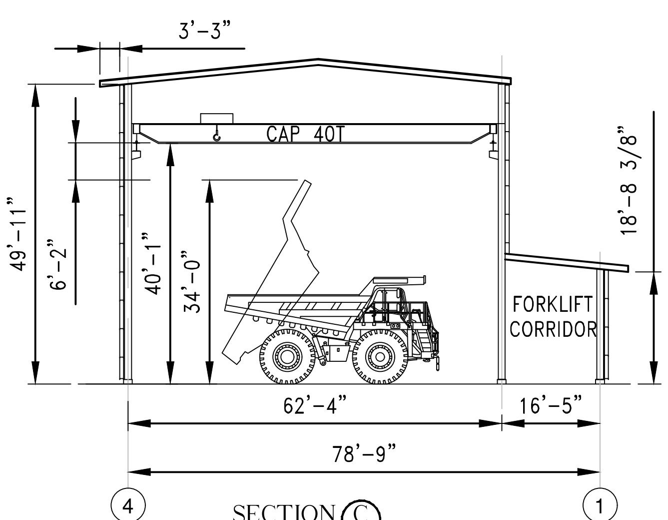
PLAN
TRUCK SHOP SCALE: 1/32" = 1'-0"







ELEVATION
TRUCK SHOP SCALE: 1/32" = 1'-0"



POTASH RIDGE

0 10 20 40
FEET
SCALE: AS NOTED

FIGURE 18.1

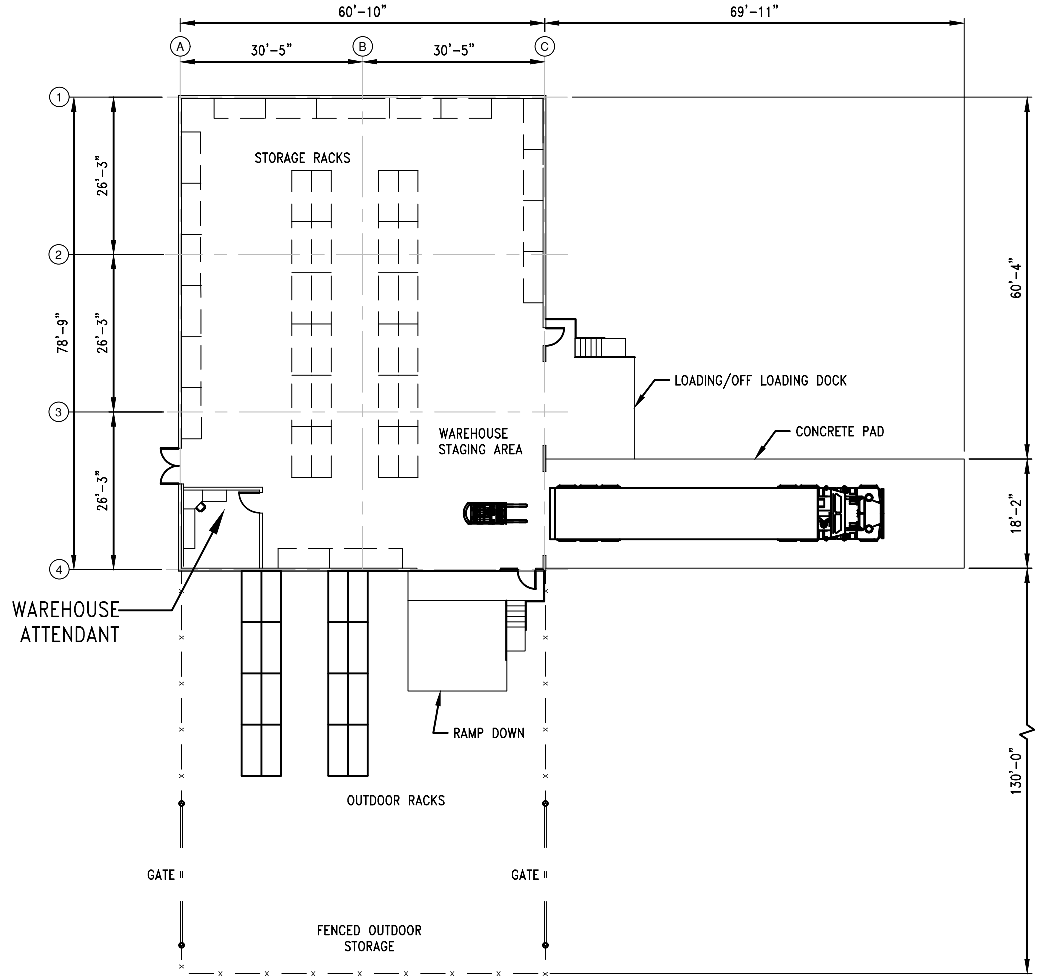
BLAWN MOUNTAIN PROJECT

MINE TRUC□ SHOP PLAN & ELEVATION VIEWS

DATE: 12/10/2016
BY:

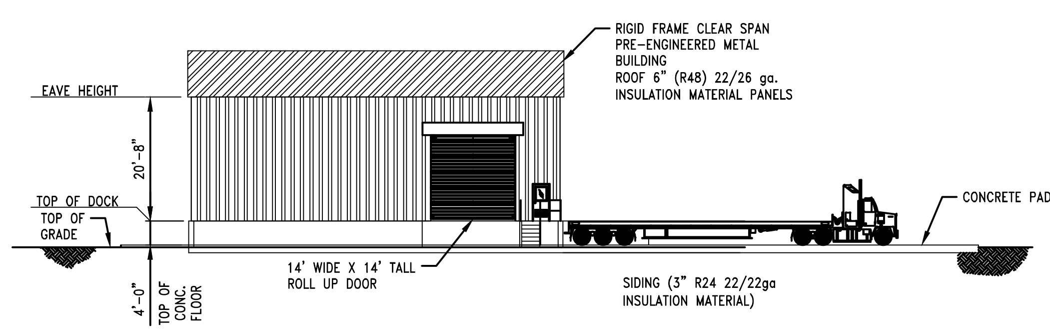
Millcreek Mining GROUP

POTASH RIDGE



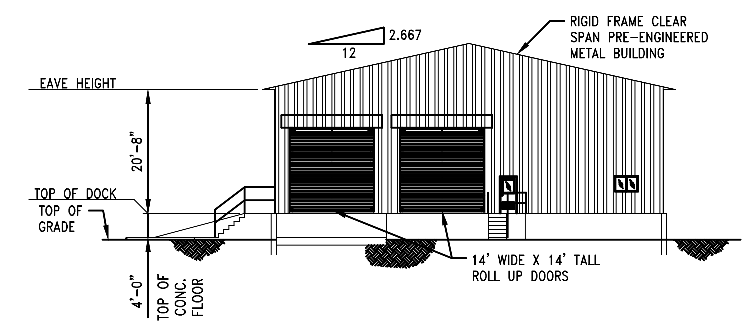
PLAN

MINE WAREHOUSE SCALE: 1/32" = 1'-0"



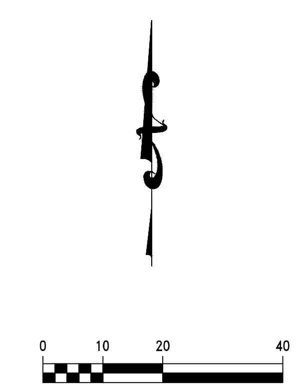
ELEVATION

MINE WAREHOUSE SCALE: 1/32" = 1'-0"



SIDE VIEW

MINE WAREHOUSE SCALE: 1/32" = 1'-0"



SCALE: AS NOTED

FIGURE 18.2

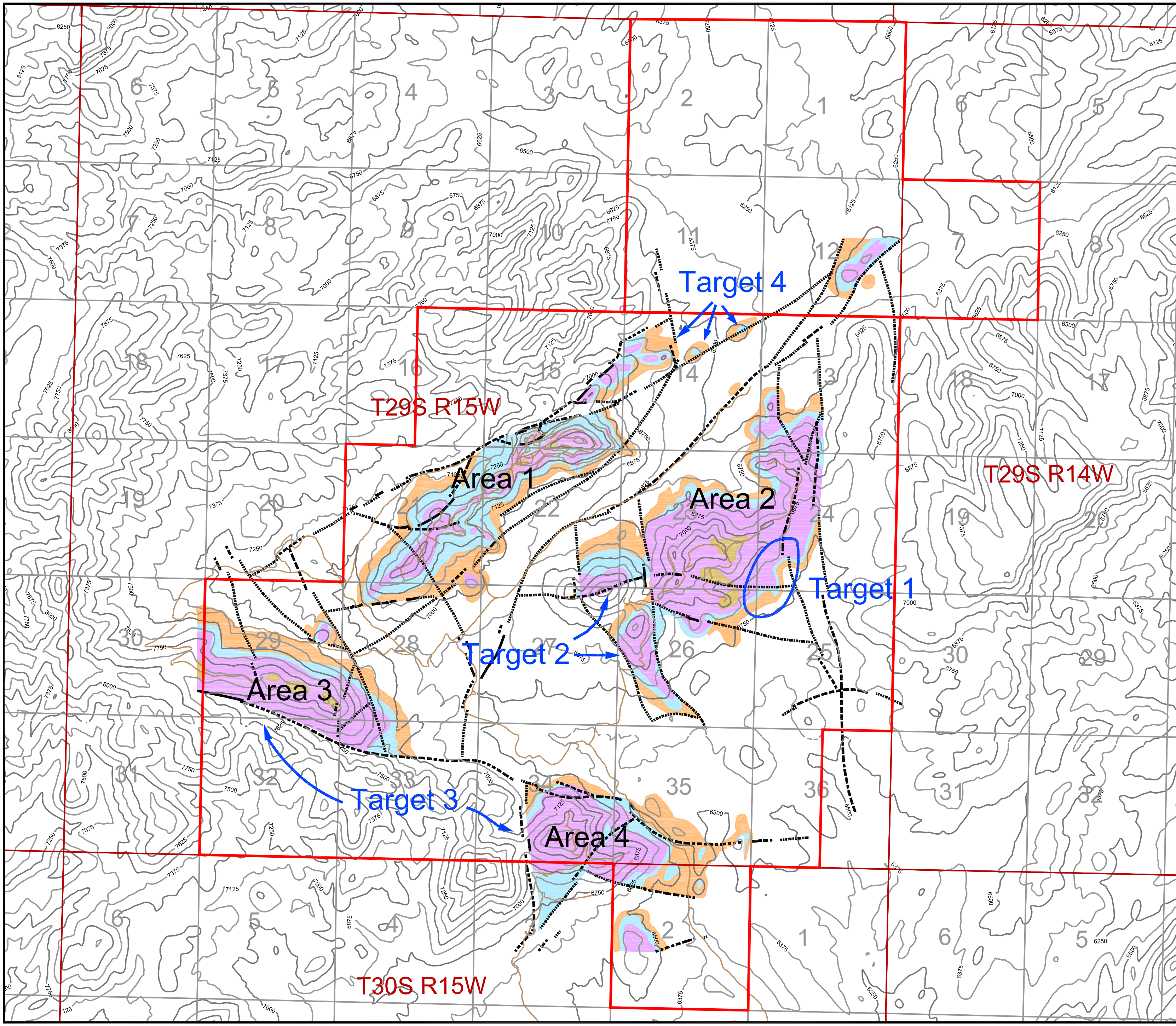
BLAWN MOUNTAIN PROJECT

MINE WAREHOUSE
PLAN & ELEVATION VIEWS

DATE: 06/28/2013

BY:

Millcreek Mining GROUP



POTASH RIDGE

LEGEND

- CONTOURS
- TOWNSHIP-RANGE
- SECTION
- LEASE BOUNDARY
- HEMATITE CLAY ZONE
- HIGH PROPYLITIC ZONE
- QUARTZ ALUNITE ZONE
- SILICA ZONE
- FAULT LOCATED
- FAULT APPROX.
- FAULT CONCEALED
- DRILLING ROAD
- EXPLORATION TARGETS

0 1000 2000 3000
FEET

SCALE: 1"=5200'

FIGURE 26.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

EXPLORATION
TARGETS

DATE: 11/02/2016
BY:

Millcreek Mining GROUP