Alabama Graphite Corp.
Coosa Graphite Project
Alabama, USA

Preliminary Economic Assessment

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Effective Date: 27 November 2015

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11/27/2015

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Units of Measure

Above mean sea level ... amsl
Acre ... ac
Ampere ... A
Annum (year) ... a
Billion ... B
Billion tonnes ... Bt

Billion years ago	Ga
British thermal unit	BTU
Centimetre	cm
Cubic centimetre	cm^3
Cubic feet per minute	cfm
Cubic feet per second	ft^3/s
Cubic foot	ft^3
Cubic inch	in^3
Cubic metre	m^3
Cubic yard	yd^3
Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dollar (American)	US$
Dollar (Canadian)	C$
Dry metric ton	dmt
Foot	ft
Gallon	gal
Gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Hertz	Hz
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2

Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litres per minute	L/min
Megabytes per second	Mb/sec
Megapascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres Baltic sea level	mbsl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Microns	µm
Microns	µm
Mile	mi
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million bank cubic metres	Mbm^3
Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa·s
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"

Second (time)	sec
Specific gravity	SG
Square centimetre	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Three Dimensional	3D
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per hour metre cubed	ts/hm^3
Total	T
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt

Abbreviations and Acronyms

Absolute Relative Difference	ABRD
Acid Base Accounting	ABA
Acid Rock Drainage	ARD
Alpine Tundra	AT
Atomic Absorption Spectrophotometer	AAS
Atomic Absorption	AA
British Columbia Environmental Assessment Act	BCEAA
British Columbia Environmental Assessment Office	BCEAO
British Columbia Environmental Assessment	BCEA
British Columbia	BC
Canadian Dam Association	CDA
Canadian Environmental Assessment Act	CEA Act
Canadian Environmental Assessment Agency	CEA Agency
Canadian Institute of Mining, Metallurgy, and Petroleum	CIM
Canadian National Railway	CNR
Carbon-in-leach	CIL
Caterpillar's® Fleet Production and Cost Analysis software	FPC
Closed-circuit Television	CCTV
Coefficient of Variation	CV
Copper equivalent	CuEq
Counter-current decantation	CCD
Cyanide Soluble	CN
Digital Elevation Model	DEM
Direct leach	DL
Distributed Control System	DCS

Drilling and Blasting	D&B
Environmental Management System	EMS
Flocculant	floc
Free Carrier	FCA
Gemcom International Inc.	Gemcom
General and administration	G&A
Gold equivalent	AuEq
Heating, Ventilating, and Air Conditioning	HVAC
High Pressure Grinding Rolls	HPGR
Indicator Kriging	IK
Inductively Coupled Plasma Atomic Emission Spectroscopy	ICP-AES
Inductively Coupled Plasma	ICP
Inspectorate America Corp.	Inspectorate
Interior Cedar – Hemlock	ICH
Internal rate of return	IRR
International Congress on Large Dams	ICOLD
Inverse Distance Cubed	ID3
Land and Resource Management Plan	LRMP
Lerchs-Grossman	LG
Life-of-mine	LOM
Load-haul-dump	LHD
Locked cycle tests	LCTs
Loss on Ignition	LOI
Metal Mining Effluent Regulations	MMER
Methyl Isobutyl Carbinol	MIBC
Metres East	mE
Metres North	mN
Mineral Deposits Research Unit	MDRU
Mineral Titles Online	MTO
National Instrument 43-101	NI 43-101
Nearest Neighbour	NN
Net Invoice Value	NIV
Net Present Value	NPV
Net Smelter Prices	NSP
Net Smelter Return	NSR
Neutralization Potential	NP
Northwest Transmission Line	NTL
Official Community Plans	OCPs
Operator Interface Station	OIS
Ordinary Kriging	OK
Organic Carbon	org
Potassium Amyl Xanthate	PAX
Predictive Ecosystem Mapping	PEM
Preliminary Assessment	PA
Preliminary Economic Assessment	PEA
Qualified Persons	QPs

Quality assurance	QA
Quality control	QC
Rhenium	Re
Rock Mass Rating	RMR ‘76
Rock Quality Designation	RQD
SAG Mill/Ball Mill/Pebble Crushing	SABC
Semi-autogenous Grinding	SAG
Standards Council of Canada	SCC
Stanford University Geostatistical Software Library	GSLIB
Tailings storage facility	TSF
Terrestrial Ecosystem Mapping	TEM
Total dissolved solids	TDS
Total Suspended Solids	TSS
Tunnel boring machine	TBM
Underflow	U/F
Valued Ecosystem Components	VECs
Waste rock facility	WRF
Water balance model	WBM
Work Breakdown Structure	WBS
Workplace Hazardous Materials Information System	WHMIS
X-Ray Fluorescence Spectrometer	XRF

1 SUMMARY

1.1 Summary

The Coosa Graphite Project (Project) is located in the western part of Coosa County, State of Alabama, USA. The Coosa Project is located 50 mi (80 km) south-southeast of Birmingham, Alabama, USA. The Project is located in the flake graphite belt of central Alabama, also known as the Alabama Graphite Belt.(Figure 1-1)

Alabama Graphite Corp. (AGC) acquired the Coosa project in 2012 based on the geologic setting. The Coosa Project mineral tenure comprises mineral rights leased by AGC totaling 41,964 acres (16,982 ha) or 65.6 sq. mi. AGC's subsidiary, Alabama Graphite Co. Inc. (Alabama Graphite), a company registered in Alabama, entered into a Mining Lease Agreement and Option pursuant to which it leased the mineral rights in respect to an area with the potential for graphite comprising 14,020.86 acres (5,674 ha) in township 22N, range 17E, Coosa County, Alabama; and received an option of first refusal to lease the mineral rights to adjacent areas comprising 30,756.52 acres (12,447 ha). On 5 November 2012, Alabama Graphite exercised the option and leased an additional 27,943.52 acres (11,308 ha) of the remaining available acres under the same terms as in the initial agreement. The total property under lease is now 41,964.38 acres (16,982 ha).

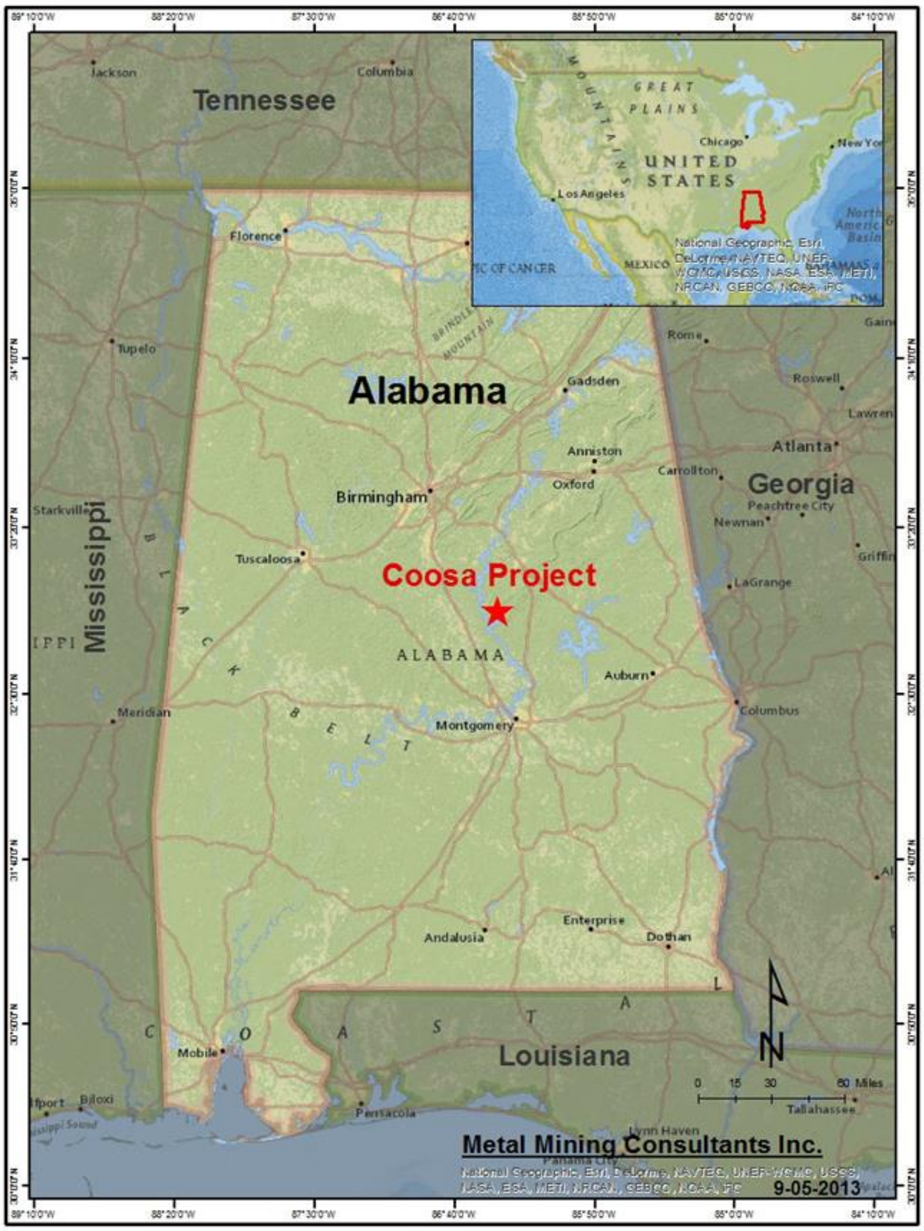
The Coosa Graphite Project PEA envisions an open pit with both primary and secondary processing plants to generate two finished (final) specialty graphite products. These are a coated spherical graphite product (CSPG) and purified micronized flake graphite (PMG). The proposed mill throughput is 5,500 tons per annum of final product for the first five years, then ramping up in Year 6 to 16,500 tons per annum in Year 7. The mine would run for 27 years in the oxide resource with the remaining resource below this pit design still available. The proposed location of required infrastructure, mining, and processing facilities for the Project are shown in Figure 1-2.

The PEA incorporates the results of AGC's recent infill drill program and a new resource estimate for Coosa Graphite Project dated 13 October 2015. Approximately 83% of the mineralization contained within the pit design is classified as Indicated Mineral Resources with the remaining 17% as Inferred Mineral Resources. The PEA also includes results of a preliminary metallurgical program, designed to characterize the metallurgical processes and recoveries in the PEA and to support the decision to advance to a future Feasibility Study (FS).

The PEA was completed by AGP Mining Consultants Inc. (AGP), an independent Canadian-based engineering firm and the updated mineral resource estimate was prepared by Metal Mining Consulting (MMC). The mineral resources conform to the Canadian Institute of

Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014) whose definitions are incorporated by reference into National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

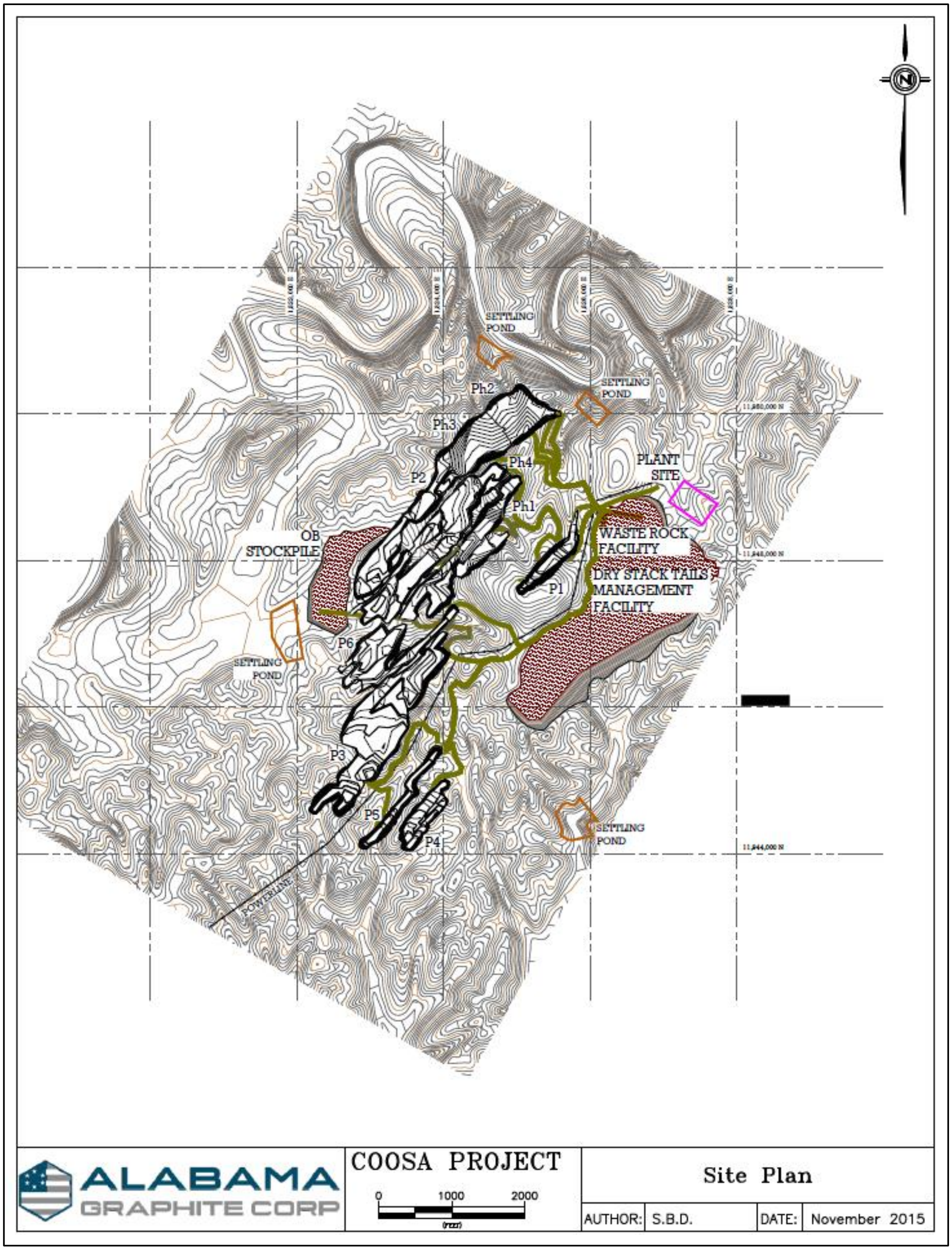
Figure 1-1 Location Map of The Coosa Project



The scenario chosen by AGP and Alabama Graphite (Base Case) in this PEA uses graphite prices of $8,165 per ton ($9,000/tonne) for coated spherical graphite (CSPG) and $1,814 per ton ($2,000/tonne) for purified micronized graphite (PMG). A 2% NSR is reflected in the economic analysis this PEA. All costs, unless otherwise noted, are in Q3 2015 United States

(US) dollars. Cost estimates were developed for all disciplines, both in operating and capital requirements.

Figure 1-2 Coosa Graphite Site Layout Plan



AGP concludes that the Coosa Graphite Project has the potential to yield a Base Case pre-tax net present value (NPV) (8% discount rate) of $444 million with an internal rate of return (IRR) of 52.2%. The Base Case post-tax NPV (8% discount rate) is estimated to be $320 million with an IRR of 45.7%. The pre-tax payback is anticipated to be 1.9 years, and the post-tax payback to be 2.0 years.

The PEA is preliminary in nature and includes Inferred Mineral Resources, which are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the results in this PEA will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

AGP recommends that the Coosa Graphite Project advance to the Feasibility level of engineering and trade-off studies, with infill and exploration trenching and drilling, engineering, and environmental field programs required to support the necessary level of engineering for a FS.

With the current level of information for the project, AGP does not foresee Mineral Resources, potential economics, or environment issues that would inhibit the project from advancing to further levels of study.

1.2 Geology and Mineralization

The Coosa Project is located at the southern end of the Appalachian mountain range, a northeast trending belt of folded and metamorphosed rocks of Neoproterozoic to Lower Paleozoic age. These are covered by the Coastal Plain Sediments of Cretaceous and younger age in the southern half of Alabama. The rocks of the southwestern end of the Appalachians are generally separated into four physiographic and geologic provinces which are, from northwest to southeast: the Interior Low Plateau province, the Appalachian Plateau province, the Valley and Ridge province, and the Piedmont province. The Alabama Graphite Belt is located in the Northern Piedmont.

The Northern Piedmont has three structural blocks: the Talladega block on the northwest side with low grade greenschist facies metasedimentary and metavolcanic rocks; the Central Coosa block with high grade, upper greenschist to kyanite and sillimanite grade metamorphic rocks and abundant pegmatite and small granitoid bodies; and the Tallapoosa block on the southeast side of high grade, middle to upper amphibolite facies metasedimentary rocks metavolcanic and metamorphosed ultramafic and mafic rocks with large areas of quartz diorite to granitic plutonic rocks.

Within the Coosa block lies the Wedowee group which contains the Higgins Ferry Formation which is the host for the Coosa project. The Higgins Ferry Formation is defined as an interbedded sequence of at least three major lithologic units: coarse to fine grained biotite-feldspar-quartz gneiss, sericite-feldspar-muscovite schist, (±biotite)-(±garnet)-muscovite schist, biotite-garnet-feldspar gneiss; (±roscoelite)-graphite-quartz schists, graphitic quartzite; and garnet quartzite and garnetiferous altered mafic rocks. Locally there are a few thin actinolite-chlorite-hornblende amphibolite and pyroxenite bodies. kyanite, sillimanite and sericite porphyroblasts common in more aluminous units, and there are scattered pegmatites.

The local Coosa project geology has been further broken down into four main lithology's:

Quartz-Muscovite-Biotite-Graphite Schist (QMBGS):

This schist is medium to coarse grained in texture and is characterized by large porphyroblasts of muscovite. The color varies from medium gray to dark gray-green. It is moderately foliated and commonly contorted, with lenticular pegmatites parallel to foliation. Pyrite and pyrrhotite are common accessory minerals as large disseminated grains. Sillimanite fibers have been observed. Graphite flakes are generally coarse in this unit although the average carbon content is generally <1% Cg.

Quartz-Graphite Schist (QGS):

This schist is both finer grained and better laminated than the QMBGS unit. The color varies from dark gray to black. Contorted foliation and pegmatites are less common. Pyrite and pyrrhotite are both finer grained and more abundant than in the QMBGS unit and form laminae parallel to foliation. Graphite is more abundant in this unit with grades >1% Cg.

Mixed QMBGS/QGS (INT):

The QMBGS and QGS are commonly interbedded at a centimeter scale forming a mixed unit with graphite grades >1% Cg.

Quartz-biotite-garnet schist (QBGS):

This unit is subordinate to both the QMBGS and QGS units. It is medium grained and medium to dark gray-green in color. The garnets are usually fine grained with a diameter of approximately 1 mm, but in places are up to 5 to 10 mm. The garnets are light pink in color, suggesting a high manganese (spessartine) content. Foliation is irregular with abundant pegmatites. Pyrite is sparse. Graphite is sparse and grades are <1% Cg.

1.3 Drilling

Alabama Graphite has conducted two drilling programs at the Coosa project totaling 20,345.4 ft. (6,201.3 m) of drilling. The first program consisted of two northwesterly trending fences of six holes each for a total of 6,003 ft. (1,829.7 m) (AGC-001 through AGC-012). All fence holes were drilled perpendicular to the stratified graphite horizons to gain information regarding the stratigraphy and graphite distribution. The two fences were approximately 1 mi (1.6 km) apart with hole spacing averaging 500 ft. (152.4 m) and drilled to a nominal depth of 500 ft. (152.4 m).

The second drilling program consisted of 57 vertical holes drilled on a 200 ft. x 200 ft. (61 m x 61 m) grid for a total of 14,342.4 ft. (4,371.6 m). The grid drilling was a combination of sonic core and HQ core. Sonic collars were drilled to increase recovery in the weathered zones near the surface. Of the 57 vertical grid holes, 44 of them were drilled with the sonic drill for a total of 1,523.5 ft. (464.4 m).

The two drill programs identified a higher grade quartz-graphite-schist (QGS) bound by a lower grade quartz-muscovite-biotite-graphite-schist (QMBGS) and a graphite barren quartz-biotite-garnet-schist (QBGS).

1.4 Resource Statement

The 13 October 2015 Coosa Project Indicated and Inferred Mineral Resources in Table 1-1 are reported at a graphitic carbon cut-off grade of 1% Cg. The resources are based on 109 unique drill holes totaling 25,905 ft. of drilling plus 9 trenches totaling 3,800 ft. of sampling, treated as drill holes.

Inferred Mineral Resources represent material that MMC considers to be too speculative to be included in economic evaluations at this point. Additional drilling will be required to convert Inferred Mineral Resources to Measured or Indicated Mineral Resources.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-1: Coosa Graphite Project Mineral Resource Estimate

Classification	Tonnage (tons)	Graphitic Carbon (Cg%)	Insitu/Contained Graphite (tons)
Indicated	78,488,000	2.39	1,876,000
Inferred	79,433,000	2.56	2,034,000

Notes: Effective Date: 13 October 2015, Scott Wilson CPG. 1. Mineral Resources are reported within constraining pit shell developed using Whittle™ software. Assumptions include graphite selling price of $1,450 /ton, process recovery of 82.8%, mining cost of $4.81/ton mined, processing cost of $14.56 /ton, G&A cost of $4.54/ton and pit slopes of 60 degrees.

Scott Wilson, CPG, President, Metal Mining Consultants Inc. prepared the Mineral Resource estimate. Mr. Wilson is a Qualified Person (QP) for the purposes of NI 43-101 and is independent of AGC.

The resource model for Coosa was constructed with Vulcan software using a block model. All of the required information about the deposit is stored in each individual block. This includes estimated carbon and statistical characteristics such as number of samples used in an estimate, distances to the nearest sample, number of drill holes used, etc., are stored in each individual block. Geologic triangulations were also used to identify the rock type of each block. Geologic codes stored in the block model were also used to assign the density within specific geologic boundaries. Table 1-2 outlines the framework of the Coosa Block model.

For Coosa, blocks bound inside the QGS, QBGS and QMBGS geologic triangulation were estimated only using samples also within these boundaries. Samples within the QGS were not used to estimate grade within the QMBGS or QBGS.

Table 1-2: Coosa Block Model Framework

	East	North	Elevation
Block Model Reference Point	1,824,000	11,942,000	-100
Number of Blocks	460	320	55
Parent Block Size Feet	20	20	20
Orientation (absolute bearing of x axis around Z axis)	30 degrees		

Grades for graphitic carbon were estimated using inverse distance to the second power (ID2). Search ellipsoid orientations were determined by evaluating regional orientation, foliation and plunge of the graphitic schist observed in the field. Drill Hole data was used to define a semi-variogram which was used to determine the appropriate search distance for the major axis. Semi-variogram model results for the semi major and minor axis were poor so dimensions were selected to tie together mineralization from surface trenches and drill holes (semi-major axis) and limit the distribution of grades perpendicular to the dip of the graphitic schist (minor-axis). Blocks estimated on the first pass were flagged accordingly and omitted during the second pass. A two pass estimation was performed in the main host (QGS) using different minimum and maximum sample counts, minimum number of drill holes and search ellipsoids for each pass. A single pass estimation was performed on the secondary host (QMBGS). Only samples flagged in the composite file as QGS were used during the estimation of QGS blocks and only samples flagged in the composite file as QMBGS were used to estimate QMBGS blocks. Composites with a length of less than 5 ft. were omitted from the estimation.

The Coosa Project resource was classified as Indicated Mineral Resources or Inferred Mineral Resources. In the opinion of the QP, the company has not obtained adequate information regarding metallurgy, graphite flake size, and geologic continuity to classify any part of the resource as Measured Mineral Resource. The grid holes demonstrated adequate spacing and continuity of graphite grade to be classified as Indicated as well as oxide and transitional material, to the north and south of the grid, where surface trench samples were taken and continuity is demonstrated with drill holes. Resource blocks estimated, to the north and south of the grid, in the reduced zone may demonstrate reasonable prospects of economic extraction however they warrant additional drilling at depth to prove grade continuity within the reduced graphitic schist before they can be upgraded from the Inferred to the Indicated Mineral classification.

1.5 Open Pit Mining

Open pit mining was selected as the mining method to examine the Coosa deposit based on the decision to restrict mining to the shallow oxidized horizon as well as the very low strip ratio of the deposit. As there is very little waste rock, and mining would not occur in competent rock, underground bulk mining was not considered to be a practical mining method for this deposit.

Due to the decision to only mine the oxidized mill feed initially, all waste and overburden material was to be hauled outside of potential mining areas. Therefore, no backfilling of pits was to be planned for so that future potential mining below the oxidized horizon could be preserved.

Pit designs were generally very shallow with depths up to approximately 100 feet. Four starter phases were identified in the northern area of the project, followed by six small lower grade pits to the south. Some mining areas were standalone pits and while others were pushbacks of other phases. During detailed designs, significant effort was placed on raising the grade of pits where reasonable. This resulted in somewhat smaller final pit designs than the original economic pit shells. The detailed tons and grade of the phases and pits have been shown in Table 1-3 by category.

Table 1-3: Pit Phase Tonnages and Diluted Grade by Resource Category

	Mill Feed (ktons)	Grade (DCg%)	Mill Feed (ktons)	Grade (DCg%)	Overburden (ktons)	Rock (ktons)	Total Waste (ktons)	Strip Ratio
	Indicated		Inferred					
Phase 1	663	3.26	347	3.42	60	5	65	0.06
Phase 2	955	3.16	10	3.05	52	84	137	0.14
Phase 3	1,457	3.19	54	2.98	62	48	110	0.07
Phase 4	1,323	3.06	1	3.19	59	0	59	0.04
Subtotal	**4,398**	**3.16**	**411**	**3.35**	**233**	**138**	**371**	**0.08**
Pit P1	-	-	474	3.22	31	216	247	0.52
Pit P2	476	2.79	11	2.67	28	19	47	0.10
Pit P3	2,831	2.75	973	2.76	191	376	567	0.15
Pit P4	440	2.90	198	2.64	40	33	73	0.11
Pit P5	136	2.78	181	2.75	29	145	175	0.55
Pit P6	4,302	2.60	369	2.90	210	54	264	0.06
Subtotal	**8,186**	**2.68**	**2,205**	**2.87**	**528**	**843**	**1,372**	**0.13**
Total	**12,584**	**2.85**	**2,616**	**2.95**	**761**	**981**	**1,742**	**0.11**

Mining will not include blasting, so a simple, small conventional loading and hauling fleet will be used. It will consist of loading with a 6.0 cubic yard excavator in the pit and 45 ton articulated trucks. Assistance to the excavator will be with a 400+ horsepower dozer to loosen and push material to the excavator for loading. Additional support to the mine and plant will be provided by graders and smaller dozers to dress roads and waste/overburden storage facilities.

Three different schedules were developed with annual reporting periods so that various product throughput rates could be evaluated at a cash flow level. Schedule 1 had a product throughput rate target of 5,511 tons per year (tpa). Schedules 2 and 3 had their product throughput rates ramped up to 16,535 tpa and 27,558 tpa respectively. A common cut-off grade of 1% DCg (diluted graphitic carbon) was applied in all three schedules.

Schedule 2 was selected as the final schedule based on a cash flow analysis. It contains a production rate of 5,511tpa for the initial five years, but transitions in Year 6 to a slightly higher rate, then reaches full plant product throughput in Year 7 at 16,535 tpa. After the initial four phases are mined complete, the average grade of mill feed trends downward for the remainder of the schedule where Pits 1-6 are mined. Mined tonnages are increased to consistent levels between 700 and 800 ktons per year in Schedule 2. Mined tonnages include mill feed, rock, and overburden. Mining and processing are completed in Year 27. A total of 12.6 million tons of Indicated mill feed grading 2.85% Cg (diluted) and 2.6 million tons of Inferred mill feed grading 2.95% Cg (diluted) will be processed. Mined overburden and rock tonnages will be 761 ktons and 981 ktons respectively.

1.6 Metallurgy and Process

A series of scoping level metallurgical test programs and one flowsheet development program were carried out on samples from the Coosa County mineralization. The work was conducted at SGS Minerals in Lakefield Ontario and spanned a time period from April 2013 to August 2015. The majority of the work was conducted on small trench or drill core samples to develop an understanding of the metallurgical response of the Coosa County mineralization.

1.6.1 Grinding and Flotation

Two Bond Ball Mill Grindability tests were conducted, which place the near surface material into the very soft category and the material from 100 to 150 ft. below surface into the medium hard category.

A flowsheet development program commenced in August 2015 and was completed within one month. A composite that included six sub-samples from different areas within the Coosa County resource was subjected to a series of 13 rougher and cleaner flotation tests to develop a process that generates a flotation concentrate with a minimum grade target of 95% total carbon. The program culminated in a flowsheet consisting of flash and rougher flotation, followed by polishing grinding with ceramic media and cleaner flotation. Since a single cleaner circuit failed to produce a satisfactory concentrate grade, the intermediate concentrate was classified at 100 mesh (150 microns) and the screen undersize and oversize are subjected to separate secondary grinding and cleaning circuits.

The metallurgical tests completed to date are sufficient to support a flotation performance projection for this PEA. A concentrate grade of 95% total carbon at 92% carbon recovery is indicated. A total of 20% of the mass in the final flotation concentrate reports to the greater than 80 mesh (177 microns) size fractions.

1.6.2 Purification Plant

A preliminary purification and shaping process has been defined for the PEA, with supporting metallurgical testwork for the purification circuit completed in 2015 at a North American testwork facility.

The testwork demonstrated the ability to produce 99.95% graphite purity from a variety of Coosa Project flotation concentrates, using a pyrometallurgical chlorination process. Micronization and spheronization circuits are built into the process to allow production of coated spherical purified graphite (CSPG) for Li-ion battery markets and a micronized purified flake product for use in polymer, plastic and rubber composites, powder metallurgy, energy materials, and friction materials, etc.

For the purposes of the PEA, AGP has assumed that the recovery of carbon to the final products would be 90% of that contained in flotation concentrates.

1.7 Infrastructure and Site Layout

The Coosa Graphite Project infrastructure consists of the following:

- Open pits
- Process plant, mobile equipment and maintenance shops
- Office/administration area
- Waste rock management facility
- Dry Stack tailings management facility
- Electrical substation

This infrastructure is required to support the final facility generating 16,500 tons per annum (15,000 Mt per annum) of spherical and micronized final product.

The mill and processing facility is to be located to the northeast of the open pit phases. This is at the head end of the valley where the waste rock storage and dry stack tailings management facilities will be located. The purification plant is to be located in the vicinity of

Rockford, Al. This is 19 mi from the mine site with access via County Roads 29 and 22. Access to natural gas in this location is key for the purification plants furnaces.

The small scale nature of the mine and proximity to nearby urban areas allows the size of the facilities to be minimal. Use of a maintenance contract means that warehousing requirements are minimized. The use of sea containers for immediate parts storage is considered with larger parts stored at the equipment vendors' facilities in the nearby city.

Office and dry facilities will be portable offices located near the mill.

The site access road is approximately 3.4 mi from Coosa County Road 29. It will be upgraded to allow truck traffic to carry the flake concentrate from the mine to the purification plant. This road will be maintained as a gravel road with the proper ditches and culverts for surface drainage.

Approximately 90,000 gal/d of fresh water will be required to satisfy water demand for the process plant. This will come from water collection in the settling pond below the dry stack tailings management facility as well as from a series of water wells.

Electrical power for the site will be supplied initially with diesel generation. This comes from a 1 MW unit for the plant with an additional 600 kW unit for site requirements and additional capacity for the mill should it be required.

When the production rate is increased to 16,500 tons per annum of final product, electrical grid power is brought to site. Access to the Alabama Power transmission line is possible with a 3.75 mi line connecting to the west of the project. The power line would follow existing roads to facilitate easy installation and periodic checks.

At full capacity the plant will have a connected load of 2.7 MW with an operating load of 1.5 MW. The mill will be serviced with a main substation and electrical power distributed by a combination of cable ducts and aerial lines. The existing diesel generation will be used as back-up power.

The dry stack tailings management facility is located to the south of the plant location. The material will be placed with an articulated dump truck and mixed with waste material. Final sloping of this material is at a 27-degree slope for long term stability. The principal objective of the dry stack tailings management facility is to provide secure containment of all tailings solids generated by the milling process. The facility must accommodate 8.7 million cubic yards of tailings. The tailings material stored in the facility has been assumed to be non-acid generating for the PEA study. This will need to be verified with material from the pilot plant testwork and further study.

The waste rock management facility only requires 620,000 cubic yards of volume for the full waste storage. This material is mixed with the dry stack tailings facility material to create a single dump platform for maintenance and reclamation purposes.

1.8 Capital and Operating Costs

1.8.1 Capital Costs

The capital costs for the Coosa Project are summarized in Table 1-4. The costs are based on installation of a 5,500 ton per annum concentrate operation in Year 1, followed by an expansion to 16,500 tons per annum in Year 7.

The project includes an open pit mine, a process plant with crushing/grinding/flotation and tailings dewatering operations plus a purification plant to allow production of purified micronized graphite (PMG) and coated, spherical, purified graphite (CSPG). Miscellaneous support infrastructure and a dry-stack tailings disposal facility are also included.

The mine has a 27-year processing life.

Table 1-4: Coosa Project Capital Cost Summary

Capital Category	Total Capital ($M)	Preproduction Capital Year -3 to Year -1 ($M)	Production Capital Year 1 ($M)	Sustaining Capital Year 2+ ($M)
Open Pit Mining	8.3	1.1	0.8	6.4
Milling & Flotation	33.6	11.7	0.0	21.9
Purification Plant	32.5	11.9	0.0	20.6
Infrastructure	8.9	2.4	0.3	6.3
Environmental	1.3	0.0	0.0	1.3
Indirects	24.0	10.0	0.1	14.0
Contingency	18.9	6.1	0.1	12.7
Total	**127.6**	**43.2**	**1.2**	**83.2**

Initial capital requirements (pre-production) are estimated to be $43.2 million. Production starts in Year 1 and the tail end of the start-up capital requirements will be partially offset by revenue in that year. Capital requirements for Year 1 total $1.2 million.

Leasing is applied to the overall mine capital requirements and only the 20% down payment of the initial capital cost is reflected in the mining capital cost summary. The indirect and contingency values vary by capital cost item.

1.9 Operating Costs

Operating costs were developed for an initial 550 t/d mining and milling operation with a 27-year milling life. In Year 7 the mill be expanded to a rate of 1,650 t/d.

Diesel fuel pricing is estimated at $2.00 /gal. This estimate was derived from current prices in the area for off-road diesel fuel. The price of diesel power generation for the first phase of the plant was estimated at $0.13/kWhr. The price for electrical power was set at $40 per MWh, based on current Alabama industrial pricing.

The Coosa pit will be developed using conventional open pit technology, with small scale equipment. This includes a backhoe excavator loading 45 ton articulated trucks. The open pit has a LOM strip ratio of 0.11:1. A total of 12.6 million tons of Indicated material and 2.6 million tons of Inferred material will be fed to the mill over the 27-year processing life. A further 1.7 million tons of material will be placed in the waste rock management facility.

Mill feed material will be available immediately from Year 1 due to the near surface nature of the deposit. Only minor overburden/organics are required to be removed initially and they will be stockpiled to the west of the pits. Processing will average 191,000 tons of material to the mill in Years 1-5. In Year 6 the plant will start its ramp-up to produce 16,500 tpa of final product. This will require on average 661,000 tons per annum of feed to the mill from Year 7 until the end of the mine life.

The plant will use conventional grinding and flotation to generate a flake concentrate. This concentrate will then be trucked 19 mi to the town of Rockford where the Purification plant is considered to be located. Tailings from the process plant will be filtered and stored in the dry stack tailings facility in the valley adjacent to the south of the plant.

G&A costs are based on an average of 7 people; 2 staff and 5 hourly. Additional charges for public relations, recruitment, logistics, bussing, etc. are also included in the G&A costs. Mine employees will be located in the immediate area, and no camp will be provided or required.

No costs are allowed for product transportation or shipping: the product sales are FOB customer's vehicle at the Coosa purification plant loading bay.

Table 1-5 shows a summary of all the operating cost categories on a cost per ton (or tonne) final product.

Table 1-5: Total Final Product Operating Costs – Years 1 to 27

Cost Category	Per Dry Ton Product ($/ton)	Per Dry Ton Product ($/tonne)
Open Pit Mining – Mill Feed and Waste	301	332
Processing	360	397
G&A	84	93
Subtotal On-Site Costs	746	822
Filter Cake Trucking	3	3
Purification Plant	662	730
Subtotal Off-Site Costs	665	733
Total	**1,410**	**1,555**

1.10 Economic Analysis

This PEA is preliminary in nature and it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Coosa Graphite project was examined from the perspective of a staged approach to develop the extensive resource. Industrial mineral projects are more constrained with market demand than traditional copper or gold projects. For this reason, a focus on annual tonnage produced is crucial to the success of the project to ensure the economics are not based on unrealistic market expectations.

Alabama Graphite considered a staged approach to allow a phased entry to the specialty graphite market. This provides for the opportunity to establish the operation at a much lower capital cost, then add capacity as market conditions permit. This also allows the ability to do two things; use free cash flow to fund further expansion plus provide an opportunity for site experience to be incorporated in later phase expansions.

Three different production scenarios were examined with final product production rates of 5,500 tpa (5K), 16,500 tpa (15K), and 27,500 tpa (25K). The two cases; 16,500 tpa (15K) and 27,500 tpa (25K) were scheduled as expansions to the 5K case.

Graphite prices for the comparison used the Base Case pricing (CSPG = $8,165/ton, PMG = $1,814/ton). The NPV was determined at an 8% discount rate.

The results of the analysis indicated that the 5K-15K-25K case had a higher NPV and IRR. But only marginally so over the 5K-15K case. When considering the marketing of 27,500 tons of final product, Alabama Graphite decided to approach the market in a more realistic manner and limit the project at this time to the 5K-15K case. The ability to rapidly expand production of the mine to the 27,500 tons exists if market conditions permit.

The final economics for the project were then based on the 5K-15K case producing 16,500 tons (15,000 tonnes) per year of final product.

Mill throughput over the 27-year mine life is 15.2 million tons. The breakdown of the Indicated and Inferred material utilized LOM in the analysis is:

- Indicated = 12.6 million tons grading 2.85% Cg
- Inferred = 2.6 million tons grading 2.95% Cg

The Indicated tonnage accounted for 83% of the mill feed with the Inferred at 17% of the tonnage. This is a result of the near surface mining of the resource initially.

The first ten years of mining had elevated grades as a result of the mining sequence. The tons and grade for that period of time are:

- Indicated = 3.3 million tons grading 3.18% Cg
- Inferred = 0.4 million tons grading 3.35% Cg

The results of the Base Case analysis indicate the potential for a Pre-tax NPV at 8% discount rate of $444 million with an internal rate of return (IRR) of 52.2%. The payback period is 1.9 years. The post-tax NPV at 8% discount rate is $320 million with an internal IRR of 45.7%. The Post-tax payback is 2.0 years.

The discounted cash flow results are shown in detail in Table 1-6.

Table 1-6: Discounted Cash Flow Results

Cost Category	Unit	Low Case	Base Case	High Case
Metal Prices				
CSPG	$/ton	7,257	8,165	9,072
PMG	$/ton	907	1,814	2,722
Operating Costs				
Open Pit Mining	(M$)	113.9	113.9	113.9
Processing	(M$)	136.0	136.0	136.0
G&A	(M$)	31.8	31.8	31.8
Filter Cake Trucking – to Purification Plant	(M$)	1.0	1.0	1.0
Purification Plant	(M$)	250.1	250.1	250.1
Subtotal Operating Costs	**(M$)**	**532.8**	**532.8**	**532.8**
Capital Costs				
Open Pit Mining	(M$)	8.3	8.3	8.3
Processing Plant	(M$)	33.6	33.6	33.6
Purification Plant	(M$)	32.5	32.5	32.5

Cost Category		Unit		Low Case	Base Case	High Case
Infrastructure		(M$)		8.9	8.9	8.9
Environmental Costs		(M$)		1.3	1.3	1.3
Indirect		(M$)		24.0	24.0	24.0
Contingency		(M$)		18.9	18.9	18.9
Subtotal Capital Costs		**(M$)**		**127.6**	**127.6**	**127.6**
Revenue		(M$)		2,141.9	2,233.5	2,827.3
Royalties (2.5%)		(M$)		38.2	45.1	51.9
Net Revenue(less Royalties)		(M$)		2,103.7	2,439.5	2,775.3
Pre-Tax Net Cash Flow (Revenue-Operating-Capital)		(M$)		1,443.2	1,779.1	2,114.9
Total Tax		(M$)		377.8	480.4	583.1
Post-Tax Net Cash Flow		**(M$)**		**1,065.4**	**1,298.7**	**1,531.8**
		Net Present Value (Pre-Tax)				
NPV @ 0%		(M$)		1,443	1,779	2,115
NPV @ 8%		(M$)		351	444	537
NPV @ 10%		(M$)		257	329	400
NPV @ 12%		(M$)		191	247	303
IRR		(%)		43.6	52.2	60.6
Payback Period		Years		2.4	1.9	1.6
		Net Present Value (Post-Tax)				
NPV @ 0%		(M$)		1,065	1,299	1,532
NPV @ 8%		(M$)		255	320	385
NPV @ 10%		(M$)		186	236	286
NPV @ 12%		(M$)		137	176	215
IRR		(%)		38.4	45.7	52.8
Payback Period		Years		6.0	2.0	1.7

The graphite prices used were based on the latest market research from UK based Benchmark Mineral Intelligence. Selling prices for coated spherical graphite (CSPG) ranged from $6,350 - $10,890 per ton ($7,000 - $12,000 per tonne). A median price of $8,165 per ton ($9,000 per tonne) was used for the base case CSPG price

Micronized purified graphite (PMG) has selling prices in the range of $1,630 - $2,540 per ton ($1,800 - $2,800 per tonne). A median price of $1,814 per ton ($2,000 per tonne) was used as the base case price for PMG.

The key economic parameters and cash cost calculation for various periods of the mine life have been shown in Table 1-7.

Table 1-7: Key Economic Parameters and Cash Cost Calculations – United States Units and Metric

Cost Category	Units	Year 1-5	Year 6-27	Year 1-27
CSPG Produced – Average Annual Production	tons	4,100	11,900	10,500
CSPG - Total Life of Mine Production	tons			283,300
PMG Produced – Average Annual Production	tons	1,400	4,000	3,500
PMG – Total Life of Mine Production	tons			94,400
Mining	$ /ton	574	280	301
Milling and Flotation	$/ton	572	343	360
G&A	$/ton	180	77	84
Sub-total Operating Cost	**$/ton**	**1,326**	**700**	**746**
Filter Cake Trucking	$/ton	3	3	3
Purification	$/ton	865	646	662
Total Production Cost	**$/ton**	**2,194**	**1,349**	**1,410**
CSPG Produced – Average Annual Production	tonnes	3,700	10,800	9,500
CSPG - Total Life of Mine Production	tonnes			257,000
PMG Produced – Average Annual Production	tonnes	1,200	3,600	3,200
PMG – Total Life of Mine Production	tonnes			85,700
Mining	$/tonne	632	309	332
Milling and Flotation	$/tonne	631	378	397
G&A	$/tonne	198	85	93
Sub-total Operating Cost	**$/tonne**	**1,462**	**772**	**822**
Filter Cake Trucking	$/tonne	3	3	3
Purification	$/tonne	953	712	730
Total Production Cost	**$/tonne**	**2,418**	**1,487**	**1,555**

The project sensitivity to various inputs was examined on the Base Case. The items that were varied were recovery, metal prices, capital cost, and operating cost. The results of the analysis are shown in Figure 1-3 and Figure 1-4.

Figure 1-3: Spider Graph of Sensitivity to NPV at 8% (Post-Tax)

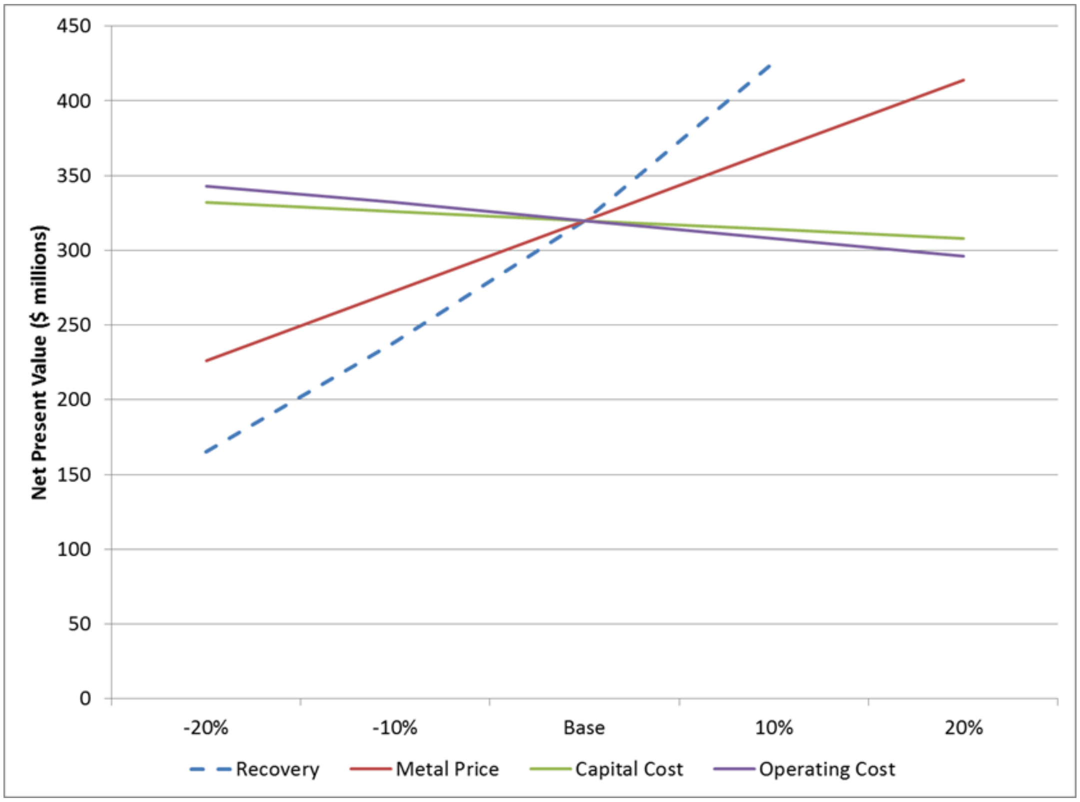
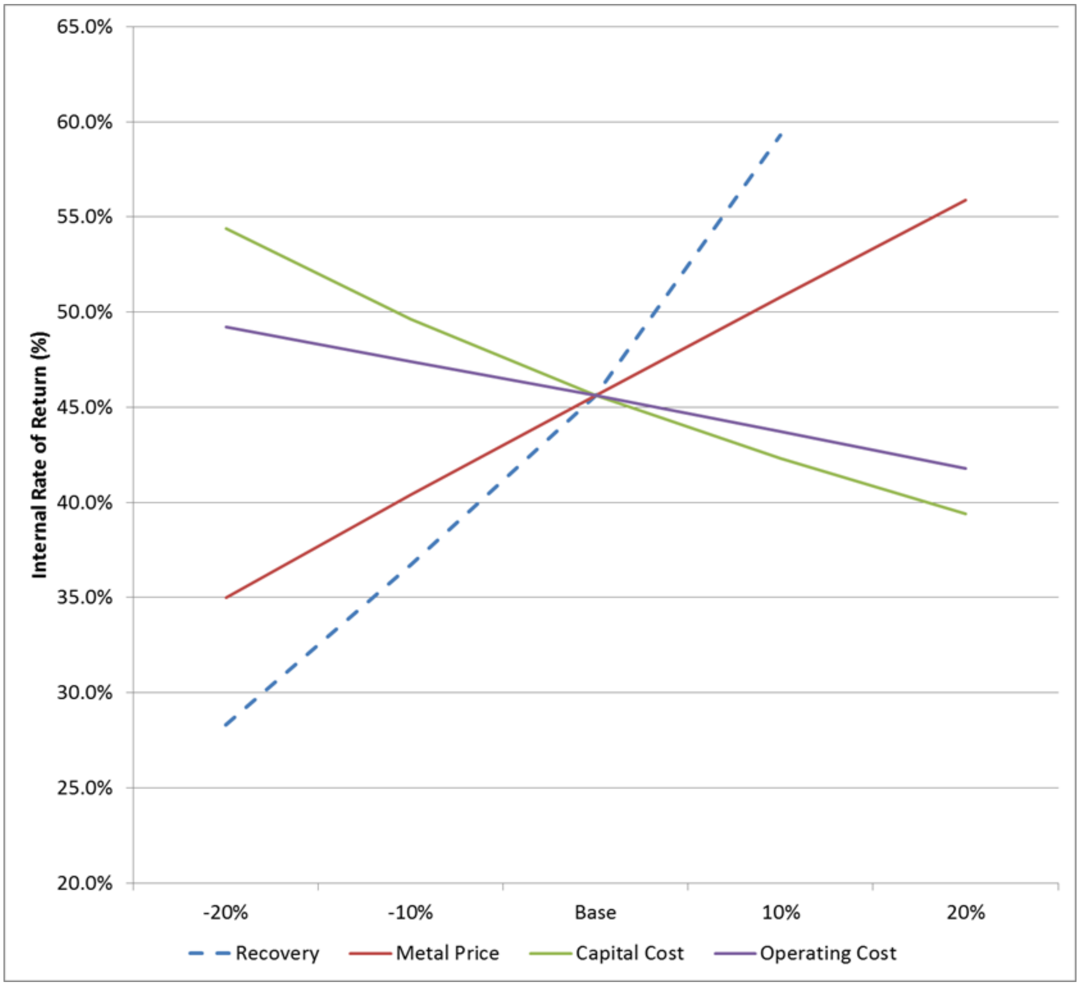


Figure 1-4: Spider Graph of IRR Sensitivity (Post-Tax)



The greatest sensitivity in the project is to recovery. To calculate the sensitivity to recovery, a percentage factor was applied to each graphite recovery in the same proportion. Therefore, while sensitivity exists, actual practice may show less fluctuation than is considered in this analysis. The +10% and +20% values are somewhat meaningless as they show recoveries greater than 100% due to the already elevated recovery. Recoveries on the negative side may also not be practical as the testwork has shown less fluctuation that the percentage variations here would indicate. But they do indicate a steeper slope showing a greater sensitivity. This large a swing in recovery has the obvious effect of influencing the project, but may not be realistic.

The next most sensitive item is the graphite price. While the project still maintains a positive NPV and elevated IRR, the reduction in the graphite price has a dramatic impact. A 20% drop in the metal price drops the IRR by 10% and the NPV by $94 million.

Operating costs are the next most sensitive. A 20% increase in these results in the $24 million decrease in the NPV.

The sensitivity to capital cost is the least sensitive. The cost of the plant is where this fluctuation may occur as it represents the majority of the capital expenditure. An increase in the capital costs by 20% drops the NPV by $12 million.

Federal and Alabama State taxes have been considered applying, among other matters, the appropriate depreciation and depletion calculations. The Federal income tax rate is approximately 35% in accordance with Internal Revenue Service Publication 542. For the purposes of the PEA, the depletion deduction has been applied to the purification plant in addition to the mining.

A 2% NSR royalty is payable by Alabama Graphite to the lessor from the commercial production and sale of graphite from the properties. A smaller royalty has also been included for an independent contractor with initial cash payments subject to certain conditions of $100,000, $150,000 and a further 0.5% royalty up to $150,000.

1.11 Environmental

With the PEA completed and some understanding of the potential size and scope of the Project now understood, the data collection and testwork can begin in earnest to prepare for the next stage of study and ultimately permit applications.

Basic data collection needs to commence immediately covering a wide range of diverse subjects, including: weather, water flows, vegetation, wildlife, and socio-economic. A comprehensive program will need to be established to collect the required information necessary to comply with the respective agency permit application requirements. This is of critical importance to ensure that the permits may be issued in a timely manner. Baseline environmental studies, including Biology (vegetation and wildlife), Cultural Resources, and Waters of the United States & Wetland Delineation will need to be completed.

ARD and metal leaching testwork needs to be developed and completed for proper waste rock characterization, dry stack tailings and development of storage options. Ongoing environmental work to advance the project includes waste effluent analysis for water chemistry, tailings solids, acid base accounting (ABA), and toxicity testing for local aquatic species. Material for this testwork is coming from the 2015 Q3 Pilot Plant Study at SGS Lakefield Research. This testwork is expected to be complete by the end of 2015.

Monthly baseline water quality data has been collected from four locations at the Coosa site. Two sample locations are upstream and downstream of the project on Weogufka Creek and two locations are from ephemeral streams. Analysis is conducted at ALS Environmental in Middleton, Pennsylvania.

The schedule for total permitting, which is a largely Alabama State requirement, is typically 6-8 months in duration upon submission of the appropriate documentation.

1.12 Proposed Budget

1.12.1 Exploration

Based on the exploration and results from the Coosa project to date, the authors recommend the Coosa project is of sufficient merit to warrant further exploration. An additional surface trenching program totaling 15,000 ft. is recommended for the Coosa project. Costs to execute this program are estimated to total $230,000.

All trenches should be constructed between existing drill hole grids and perpendicular to the known strike of the graphitic schist. The program can be completed in three phases as follows:

- Phase 1: Central/grid area
- Phase 2: North of central/grid area.
- Phase 3: South of central/grid area.

1.13 Feasibility Study

AGP recommends that Alabama Graphite proceed with activities that will allow completion of a Feasibility Study (FS). AGP estimates that the cost of a FS for the Coosa Graphite Project would be in the range of $3-6 million to complete, although the scope of the FS remains to be thoroughly reviewed and a detailed budget determined. The FS would cover areas such as:

- resource estimate update
- metallurgical testwork, pilot plant, product testing
- geotechnical and hydrogeological studies
- condemnation drilling
- dry stack tailings management facility design, material characterization and site geotechnical
- environmental management studies and data collection
- graphite marketing and sales studies
- capital and operating cost estimation
- infrastructure evaluation and costing
- financial evaluation
- project management and administration.

Aspects of the study can take place concurrent with the exploration budget. Environmental and social work and some of the geotechnical and metallurgical work would also occur at this time.

2 INTRODUCTION AND TERMS OF REFERENCE

2.1 General

Alabama Graphite Corp. (AGC) is a Canadian-based flake graphite exploration and development company. The Company operates through its wholly owned subsidiary, Alabama Graphite Company Inc. (a company registered in the state of Alabama). Alabama Graphite Corp. is focused on the exploration and development of its flagship Coosa Graphite Project in Coosa County, Alabama, and its Bama Mine Project in Chilton County, Alabama.

Alabama Graphite Corp. holds a 100% interest in the mineral rights for these two U.S.-based graphite projects, which are both located on private land. The two projects encompass more than 43,000 acres and are located in a geopolitically stable, mining-friendly jurisdiction with significant historical production of crystalline flake graphite in the flake graphite belt of central Alabama, also known as the Alabama Graphite Belt (source: U.S. Bureau of Mines). A significant portion of the Alabama deposits are characterized by graphite-bearing material that is oxidized and has been weathered into extremely soft rock.

2.2 Terms of Reference

This National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) Technical Report on the Coosa Graphite Project (Technical Report or Report) was independently prepared by AGP Mining Consultants Inc. (AGP), and Metal Mining Consultants Inc. (MMC). This Technical Report was prepared for AGC to provide a Preliminary Economic Assessment (PEA) of the Coosa Graphite Project based on the updated Mineral Resource Estimate dated 13 October 2015 completed by Metal Mining Consultants. The Technical Report will be filed with Canadian Securities Administrators on SEDAR.

All measurement units used in this report are in United States Customary Units (unless otherwise noted), and currency is expressed in United States dollars unless stated otherwise.

2.3 Qualified Persons

The Qualified Persons (QPs) responsible for the preparation of this Report, as defined in NI 43-101, are the following:

- Scott E. Wilson, CPG, President (Metal Mining Consultants)
- Scott D. Redwood, FIMMM (Consultant)
- Gordon Zurowski, P. Eng., Principal Mining Engineer (AGP)
- Willie Hamilton, P.Eng., Senior Associate Mining Engineer (AGP)
- Andy Holloway, CEng., P.Eng., Principal Process Engineer (AGP)

- Phillip Mackey, P.Eng., B.Sc. (Hons), Ph.D., President (P.J. Mackey Technology)
- Lyn Jones, P.Eng., Senior Associate Process Engineer (AGP)
- Oliver Peters, P.Eng., Senior Associate Metallurgist (AGP)

All QPs of Metal Mining Consultants, AGP and P.J. Mackey Technology are independent of Alabama Graphite or of any company associated with Alabama Graphite.

2.4 Site Visits and Responsibility

Metal Mining Consultants and AGP Mining Consultants have conducted site visits to the project as shown in Table 2-1.

Table 2-1: Date of Site Visits and Areas of Responsibility

QP Name	Site Visit Dates	Area of Responsibility
Scott E. Wilson	No site visit	Responsible for Section 14, and those portions of the Summary, Interpretations and Conclusions, and Recommendations that pertain to those sections of the Technical Report.
Stewart D. Redwood	June 8 – 10, 2015 August 23 – 27, 2014 February 21-22, 2013	Responsible for Sections 4 to 12, and those portions of the Summary, Interpretations and Conclusions, Recommendations and References that pertain to those sections of the Technical Report.
Gordon Zurowski	July 9 – 10, 2015	Responsible for Sections 1,2,3,15,18,20,21.1.1, 21.1.2,21.1.5 to 21.1.8, 21.2.1 to 21.2.2, 21.2.5 to 21.2.7,22,23,24, 25.2, and those portions of the Interpretations and Conclusions, and Recommendations that pertain to those sections of the Technical Report.
Willie Hamilton	No site visit	Responsible for Sections 16, and those portions of the Summary, Interpretations and Conclusions, and Recommendations that pertain to those sections of the Technical Report.
Andy Holloway	No site visit	Responsible for Sections 17.5, 21.1.4, 21.2.4, 25.5, 26.2 and those portions of the Summary, Interpretations and Conclusions, and Recommendations that pertain to those sections of the Technical Report.
Phillip Mackey	No site visit	Responsible for Sections 13.6, 13.7.2 and those portions of the Summary, Interpretations and Conclusions, and Recommendations that pertain to those sections of the Technical Report.
Lyn Jones	No site visit	Responsible for Sections 17.1 to 17.4, 21.1.3, 21.1.9, 21.2.3, 25.5, 26.2 and those portions of the Summary, Interpretations and Conclusions, and Recommendations that pertain to those sections of the Technical Report.

QP Name	Site Visit Dates	Area of Responsibility
Oliver Peters	No site visit	Responsible for Sections 13.1 to 13.5, 13.7.1 and those portions of the Summary, Interpretations and Conclusions, and Recommendations that pertain to those sections of the Technical Report.

2.5 Effective Dates

The effective date of the PEA for the Coosa Graphite project is 27 November 2015.

The effective date of the Mineral Resource estimate for the Coosa Graphite Project is 13 October 2015.

2.6 Previous Technical Reports

Two previous NI 43-101 technical reports were completed on the Coosa Graphite project:

1) Wilson, Scott E., Redwood, Stewart D., 2014. Technical Report, Alabama Graphite Corp., Coosa Project, Revised March 26,2014
2) Wilson, Scott E., Redwood, Stewart D., 2014. Technical Report, Alabama Graphite Corp., Coosa Project, Effective Date: 13 October 2015

These reports are on file on the SEDAR website (www.sedar.com). Background information and a portion of the technical data for this Technical Report were obtained from these reports.

3 RELIANCE ON OTHER EXPERTS

The authors have relied on information supplied by the Law Office of John Foster Tyra, PC, Tuscaloosa, Alabama, as the legal counsel to AGC, for information about the mining titles described in Section 4.2.1 of this report, in reports to AGC dated 17 July 2012 ("Preliminary Mining Title Opinion"), and 26 October 2012 ("Alabama Graphite Option"), and on information from AGC from which Figure 4-2, Figure 4-3 and Table 4-1 were prepared. The authors have not carried out independent verification of the mineral rights.

3.1 Mineral Tenure

The QPs have not reviewed the mineral tenure, nor independently verified the legal status or ownership of the Project area or underlying property agreements. The QP's have relied upon information provided by Alabama Graphite.

3.2 Metallurgy and Processing

The authors have relied on work completed in 2015 for Alabama Graphite by a confidential North American testwork facility. Specifically, an order of magnitude engineering study and capital/operating cost estimate for the purification section of an integrated micronization, shaping, purification and coating plant.

The identity of the testwork facility has been kept anonymous at Alabama Graphite's request, for reasons of commercial confidentiality.

3.3 Permitting

The QP's have relied on information regarding the status of the current surface rights, road access and permits through opinions and data supplied by Alabama Graphite.

3.4 Taxation

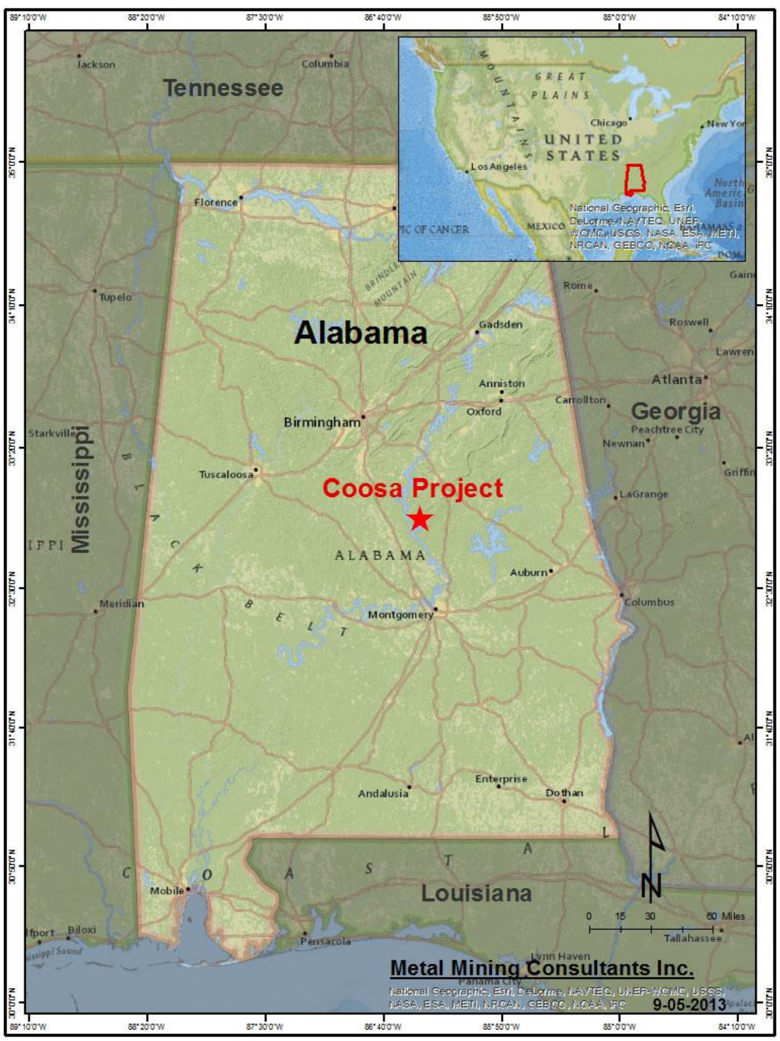
The QPs have relied on information regarding taxation supplied by the accounting firm MNP LLP and incorporated in the cash flow. This information is used in Sections 22 of this Report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Coosa Property Location

The Coosa Graphite Project is located in the western part of Coosa County, State of Alabama, USA. The property covers parts of townships T. 21 N., T. 22 N., T. 23 N. and T. 24 N. and ranges R. 16 E., R. 17 E., R. 18 E. and R. 19 E. The western boundary is approximately the Coosa River. The center of the drill grid is at 32°54'30''N, 86°24'00" W. The location is shown in Figure 4-1.

Figure 4-1: Location Map of the Coosa Project



4.2 Coosa Mineral Tenure

4.2.1 Ownership

The Coosa Project mineral tenure comprises mineral rights leased by AGC totaling approximately 41,964 acres (16,982 ha) or 65.6 sq. mi (169.8 sq. km).

On 1 August 2012, AGC's subsidiary, Alabama Graphite, a company registered in Alabama, entered into a Mining Lease Agreement and Option with Eugenia W. Dean (since 2014 the estate of Eugenia W. Dean), Birmingham, Alabama pursuant to which it: (a) leased the mineral rights in respect to an area with the potential for graphite comprising 14,020.86 acres (approximately 5,674 ha) in township 22N, range 17E, Coosa County, Alabama; and (b) received an option of first refusal to lease the mineral rights to adjacent areas comprising 30,756.52 acres (12,447 ha). On 5 November 2012, Alabama Graphite exercised the option and leased an additional 27,944.02 acres (approximately 11,308 ha) of the remaining available acres under the same terms as in the initial agreement. The total property under lease is now 41,964.88 acres (approximately 16,982 ha). Maps of the mineral rights are shown in Figure 4-2 and Figure 4-3, and a list of the mineral rights is given in Table 4-1.

Under the terms of the agreement, the lease is for successive renewable 5-year terms (not to exceed 70 years) in consideration of an initial cash payment of $30,000 and annual advance royalty payments of $10,000, starting on 1 July 2015 (paid). Alabama Graphite also paid $1,000 for the Option and was required to make annual payments of $1,000 to keep the Option in good standing. Alabama Graphite made a payment of $48,537 on 5 November 2012 to exercise the option as an initial 3--year payment through 4 November 2015, and issued 25,000 shares to the lessor.

Alabama Graphite is also obliged to pay the lessor a net smelter returns royalty of 2% from the commercial production and sale of graphite from the properties, as well as royalties for any precious metals, mica, iron, magnetite, manganese, calcium carbonate, copper, tantalum, and rare earths commercially produced and sold from the properties.

In connection with entering into the Mining Lease Agreement and Option, Alabama Graphite engaged Charles Merchant ("Merchant"), an arm's length party, as an independent contractor to assist it with establishing its graphite operations in Alabama. As consideration for his services, the Company paid Merchant the cash sum of $320,000 and is to issue 500,000 common shares of the Company (the "Finder's Shares") to him once Alabama Graphite has secured the surface rights to the Coosa Graphite property.

Alabama Graphite is obliged to pay Merchant an additional $100,000 upon receipt by Alabama Graphite of a bankable FS in respect of the Leased Property and a further $150,000 upon full permitting of the Leased Property. The Company is also obliged to pay Merchant net smelter royalties of 0.5% up to an aggregate amount of $150,000 if and when graphite mining operations commence on the Leased Property.

The mineral rights are patents of private land issued between 1842 and 1860 (for the four with title opinion). The surface land rights were subsequently sold separately from the mineral rights and are now held by a different owner. The ownership of the mineral rights is a matter of public record in the Probate Records of Coosa County, Alabama. The authority for the State of Alabama non-fuel minerals surface mining is the Alabama Department of Labor, Mining Division. No assessment work is required to hold mineral rights on private land. The mineral rights were granted in perpetuity. The Coosa mineral rights are identified by their township, range, section and, where relevant, quarters, in the same manner as land rights, and do not have an identification number or name. The mineral rights have not been surveyed.

The mineral rights are patents of private land that were issued before the introduction of the General Mining Law of 1872, which governs mining on Federal public lands, hence this law is not applicable to the Coosa Project and the mineral rights are not held as either patented or unpatented lode claims. Thus the mineral rights do not have claim names and are not defined by metes and bounds.

Townships, ranges, sections and quarters are numbered as follows in the United States:

- Township measures the distance north or south of a base line which is a designated parallel. A township usually measures 6 mi. The first 6 mi north of the base line is township one north, written T.1 N.
- Range measures east or west from the principal meridian which is a designated meridian. Ranges are also usually 6 mi in size. The first 6 mi west of the principal meridian is written range one west, R. 1 W.
- Townships are subdivided into sections. Each township is 6 mi x 6 mi and contains 36 sections of 1 mi x 1 mi. The northeast corner is section 1. They are numbered to the west in the north row, then to the east in the second row, and so on in a snake-like pattern.
- Sections are subdivided into northeast, northwest, southeast and southwest quarters of 160 acres, for example NW ¼ (northwest quarter). There are further subdivided into quarters of 40 acres and are designated by the smallest quarter followed by the largest quarter, for example, NE ¼, SW ¼ means the northeast quarter of the southwest quarter).

The drilling carried out by AGC is on the mineral rights of township 22 N., range 17 E, sections 5, 7 and 8. John Foster Tyra, PC, Tuscaloosa, Alabama, carried out a title search in the public records in the Office of the Probate Judge of Coosa County and delivered a title opinion on Sections 5, 7, 8 and 9, township 22 N, range 17 E in reports to AGC dated 17 July 2012 (“Preliminary Mining Title Opinion”) and 26 October 2012 (“Alabama Graphite Option”).

Figure 4-2: Map of the Mineral Rights of the Coosa Project

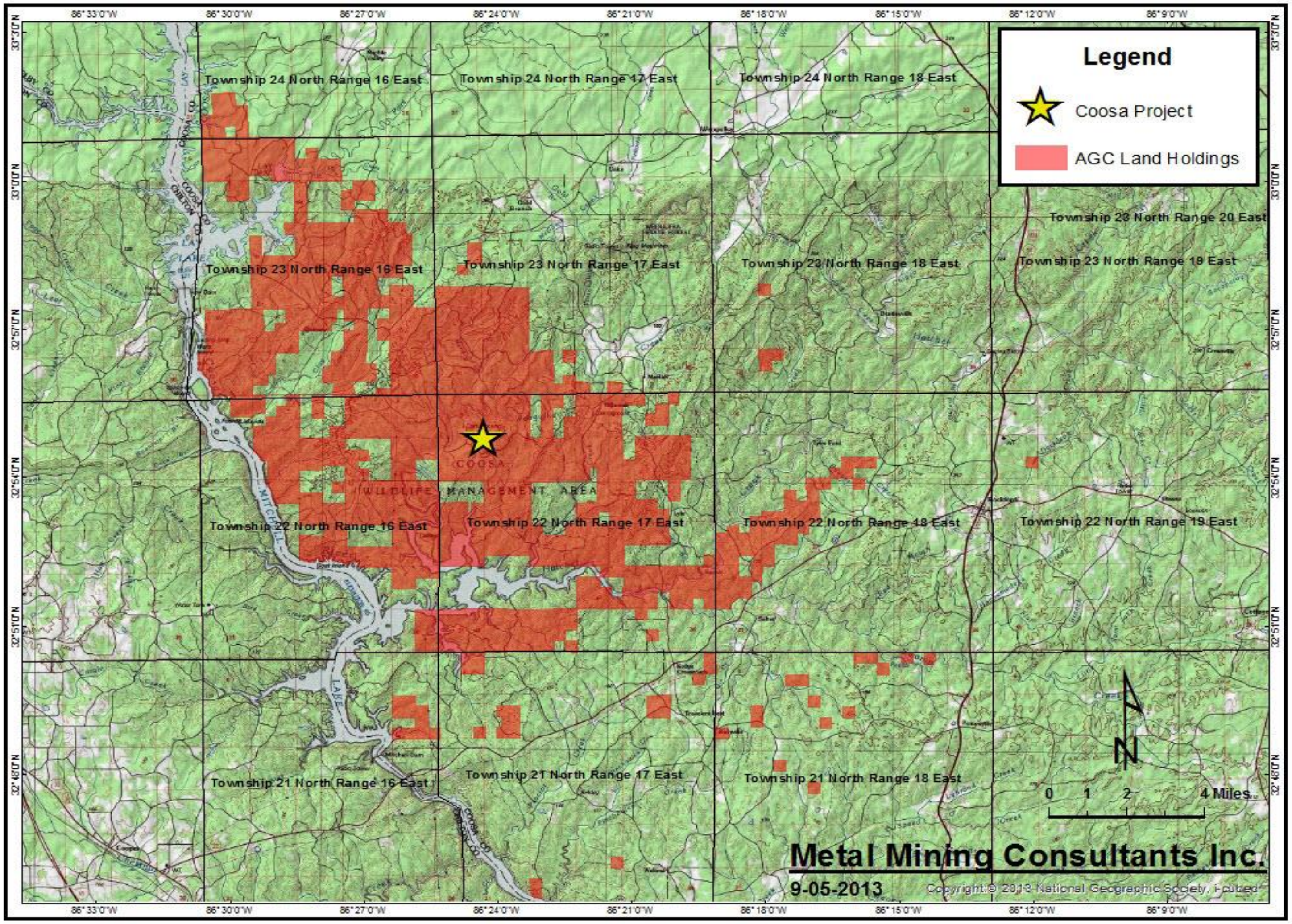


Figure 4-3: Detailed Map of Mineral Rights and Drill Hole Locations, Coosa Project

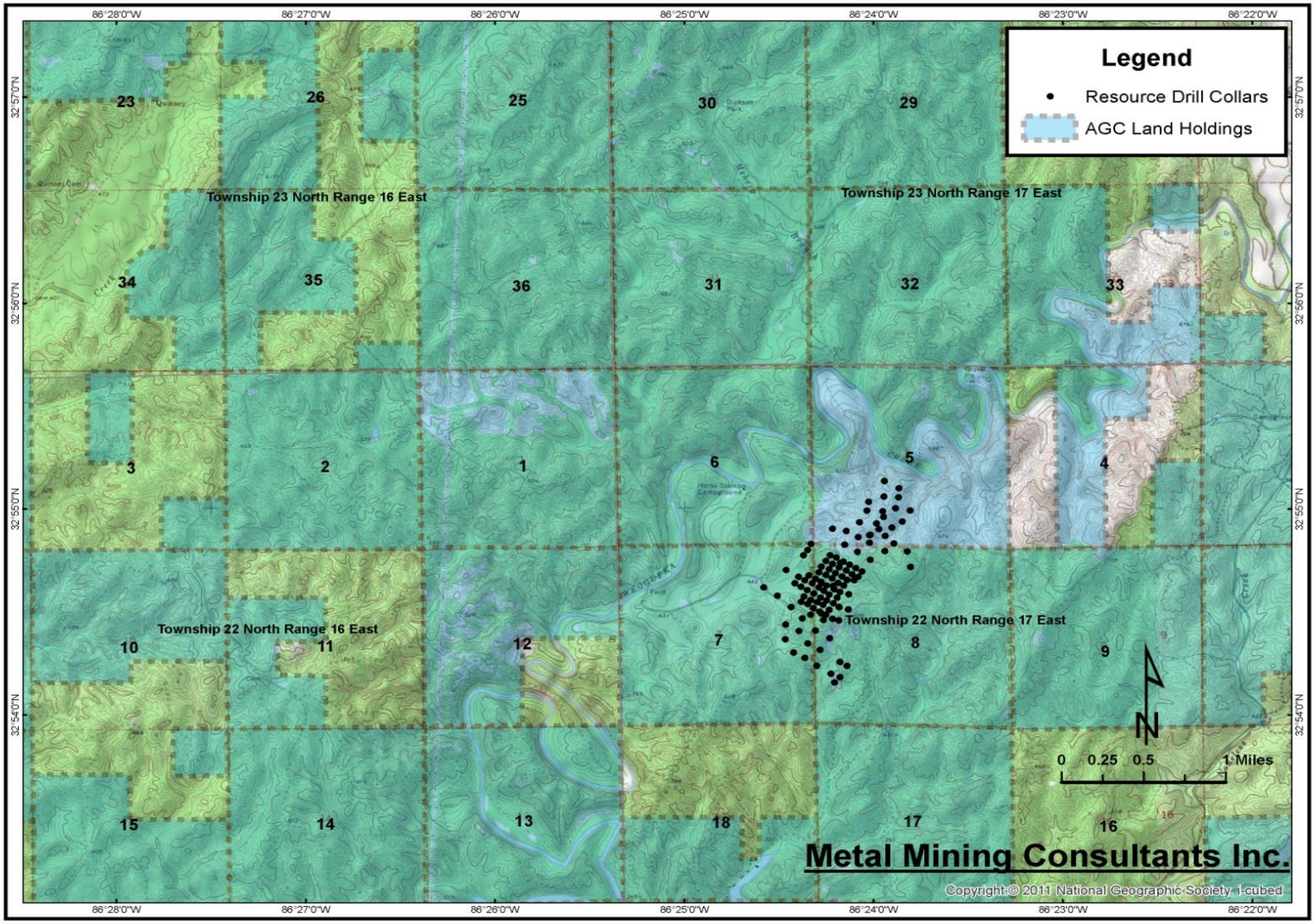


Table 4-1: List of Mineral Rights in the Coosa Project, Coosa County, Alabama

Township	Range	Section	Quarter	Area (acres)
T. 21 N.	R. 16 E.	Section 12	SW1/4 NE1/4, NW1/4, S1/2 SW1/4, NW1/4 SE1/4, S1/2 SE1/4	400
T. 21 N.	R. 17 E.	Section 01	E1/2 NE1/4, NW1/4 SE 1/4	120
T. 21 N.	R. 17 E.	Section 06	NE1/4	160
T. 21 N.	R. 17 E.	Section 07	SE1/4 SE1/4	40
T. 21 N.	R. 17 E.	Section 08	SW1/4 NE1/4, SE1/4 NW1/4, E1/2 NE1/4 SW1/4, SE1/4 SW1/4, W1/2 SE1/4	220
T. 21 N.	R. 17 E.	Section 11	NE1/4	160
T. 21 N.	R. 17 E.	Section 27	SE1/4 SE1/4	40
T. 21 N.	R. 17 E.	Section 33	SW1/4 NW1/4, fractional NW1/4 SW1/4	69
T. 21 N.	R. 18 E.	Section 02	NW1/4 NW1/4; ALSO 10 acres off the East side of the NE1/4 NW1/4; ALSO all that part of the NW1/4 NE1/4 lying North of the creek Containing 20 acres, more or less	70
T. 21 N.	R. 18 E.	Section 03	SW1/4 NE1/4, N1/2 NW1/4	120
T. 21 N.	R. 18 E.	Section 04	A tract of land on the East side of the NE1/4 NE1/4, Containing 6.5 acres, more or less	6.5
T. 21 N.	R. 18 E.	Section 05	N1/2 SE1/4	80
T. 21 N.	R. 18 E.	Section 07	SE1/4 NE1/4, NE1/4 SE1/4, SW1/4 SW1/4	120
T. 21 N.	R. 18 E.	Section 09	SE1/4 NE1/4, NW/4 NW1/4, NE1/4 SW1/4	120
T. 21 N.	R. 18 E.	Section 17	NE1/4 SW1/4	40
T. 21 N.	R. 18 E.	Section 21	NW1/4 NW1/4	40
T. 22 N.	R. 16 E.	Section 01	All	640
T. 22 N.	R. 16 E.	Section 02	All	640
T. 22 N.	R. 16 E.	Section 03	E1/2 NW1/4, SE1/4 SE1/4	120
T. 22 N.	R. 16 E.	Section 04	NE1/4 NE1/4, S1/2 NE1/4, NW1/4 NW1/4, N1/2 SW1/4 NW1/4, E1/2 SW1/4, SW1/4 SW1/4, SE1/4	460.1
T. 22 N.	R. 16 E.	Section 05	NE1/4	160
T. 22 N.	R. 16 E.	Section 08	Lot A	14.1
T. 22 N.	R. 16 E.	Section 09	N1/2, SE1/4, E1/2 SW1/4	554.2
T. 22 N.	R. 16 E.	Section 10	N1/2, N1/2 SW1/4, NW1/4 SE1/4, NE1/4 SE1/4	481.1
T. 22 N.	R. 16 E.	Section 11	S1/2 NW1/4, S1/2 SW1/4, NW1/4 SW1/4	200
T. 22 N.	R. 16 E.	Section 12	N1/2, SW1/4	480
T. 22 N.	R. 16 E.	Section 13	All	641.2
T. 22 N.	R. 16 E.	Section 14	All	640
T. 22 N.	R. 16 E.	Section 15	S1/2, S1/2 NW1/4, NE1/4 NE1/4	440.5
T. 22 N.	R. 16 E.	Section 16	All of fractional N1/2	196.7
T. 22 N.	R. 16 E.	Section 22	NE1/4, S1/2 NW1/4, NE1/4 NW1/4, all of fractional S1/2	580
T. 22 N.	R. 16 E.	Section 23	All of fractional S1/2	318.3
T. 22 N.	R. 16 E.	Section 24	All	640
T. 22 N.	R. 16 E.	Section 25	N1/2 NE1/4, NW1/4, E1/2 SE1/4; also 10 acres in the Northeast Corner of the NW1/4 SW1/4; also a fractional part containing 250 acres, more or less, as described in that certain deed from the trustees under the last will and testament of John R. Hegeman to Charles A. Dean, dated 11 November 1929, and recorded in Deed Book 6, page 582, in the land records of the Probate Judge's Office, Coosa County, Alabama.	580
T. 22 N.	R. 16 E.	Section 36	E1/2	320
T. 22 N.	R. 17 E.	Section 01	NW1/4 NW1/4	40

Township	Range	Section	Quarter	Area (acres)
T. 22 N.	R. 17 E.	Section 02	SE1/4 NE1/4, N1/2 NW1/4, N1/2 SE1/4	200
T. 22 N.	R. 17 E.	Section 03	All	640
T. 22 N.	R. 17 E.	Section 04	NW1/4 NE1/4, E1/2 SE1/4, E1/2 NW1/4, E1/2 SW1/4	280
T. 22 N.	R. 17 E.	Section 05	All	640.96
T. 22 N.	R. 17 E.	Section 06	All	640
T. 22 N.	R. 17 E.	Section 07	All	640
T. 22 N.	R. 17 E.	Section 08	All	640
T. 22 N.	R. 17 E.	Section 09	All	640
T. 22 N.	R. 17 E.	Section 10	All, except E1/2 SE1/4 SW1/4	620
T. 22 N.	R. 17 E.	Section 11	All, except E1/2 NW1/4	560
T. 22 N.	R. 17 E.	Section 12	W1/2 except 8 acres in the SE corner of the SE1/4 SW1/4	312
T. 22 N.	R. 17 E.	Section 13	E1/2 NW1/4	80
T. 22 N.	R. 17 E.	Section 14	W1/2 NW1/4, SW1/4, W1/2 SE1/4, NE1/4 SE1/4	360
T. 22 N.	R. 17 E.	Section 15	NE1/4, SW1/4, N1/2 SE1/4, SE1/4 SE1/4	440
T. 22 N.	R. 17 E.	Section 17	All	640
T. 22 N.	R. 17 E.	Section 18	SW1/4, NE1/4 SE1/4, W1/2 NW1/4 SE1/4, S1/2 SE1/4	300
T. 22 N.	R. 17 E.	Section 19	All	640
T. 22 N.	R. 17 E.	Section 20	N1/2, N1/2 S1/2	480
T. 22 N.	R. 17 E.	Section 21	NE1/4 NE1/4, S1/2 NE1/4, NW1/4 NW1/4, S1/2 NW1/4, S1/2	560
T. 22 N.	R. 17 E.	Section 22	All	642
T. 22 N.	R. 17 E.	Section 23	N1/2, ALSO the South 15 acres of the SE1/4 SE1/4	335
T. 22 N.	R. 17 E.	Section 24	SE1/4 SE1/4	40
T. 22 N.	R. 17 E.	Section 25	All, except SW1/4 SE1/4	600
T. 22 N.	R. 17 E.	Section 26	N1/2, SW1/4, W1/2 SE1/4, all NE1/4 SE1/4 North of Hatchett Creek	580
T. 22 N.	R. 17 E.	Section 27	N1/2 NE1/4, S1/2 NW1/4, SW1/4, W1/2 SE1/4, SE1/4 SE1/4	440.9
T. 22 N.	R. 17 E.	Section 31	All	640
T. 22 N.	R. 17 E.	Section 32	All	640
T. 22 N.	R. 17 E.	Section 33	N1/2, SW1/4, NW1/4 SE1/4, SE1/4 SE1/4	560
T. 22 N.	R. 17 E.	Section 34	S1/2 NW1/4, NE1/4 SW1/4	120
T. 22 N.	R. 17 E.	Section 35	NW1/4 NE1/4, NE1/4 NW1/4 except 10 acres off the West side	70
T. 22 N.	R. 18 E.	Section 09	SW1/4 SE1/4, E1/2 SE1/4	120
T. 22 N.	R. 18 E.	Section 10	N1/2 SW1/4	80
T. 22 N.	R. 18 E.	Section 15	NW1/4 NW1/4	40
T. 22 N.	R. 18 E.	Section 16	NE1/4 NW1/4, W1/2 W1/2	200
T. 22 N.	R. 18 E.	Section 17	SW1/4 NE1/4, SE1/4, S1/2 SW1/4	280
T. 22 N.	R. 18 E.	Section 18	S1/2 SE1/4 SE1/4	20
T. 22 N.	R. 18 E.	Section 19	NE1/4 NE1/4, S1/2 NE1/4, S1/2 NW1/4, SW1/4, N1/2 SE1/4; ALSO 25 acres lying South of Hatchett Creek in the NE1/4 NW1/4; ALSO 20 acres lying East of Cox Mill Road in the SE1/4 SE1/4	485
T. 22 N.	R. 18 E.	Section 20	N1/2 NE1/4; SW1/4 NE1/4 less 10 acres in the Southwest corner thereof; E1/2 NW1/4 less 2 acres now or formerly owned by J.D. Hardy, and less 10 acres East of the Old Road in the Southeast corner of the SE1/4 NW1/4; W1/2 NE1/4 SW1/4 less 1 acre at the church; 5 acres in the Northwest corner of the NE1/4 SE1/4; W1/2 W1/2	362
T. 22 N.	R. 18 E.	Section 21	NW1/4 NW1/4	40
T. 22 N.	R. 19 E.	Section 07	NE1/4 SE1/4 except 15 acres now or formerly owned by G.W. Miller	25
T. 23 N.	R. 16 E.	Section 03	S1/2, SE1/4 NW1/4	360
T. 23 N.	R. 16 E.	Section 04	E1/2, NW1/4, N1/2 SW1/4, N1/2 S1/4 SW1/4	600

Township	Range	Section	Quarter	Area (acres)
T. 23 N.	R. 16 E.	Section 05	All	643
T. 23 N.	R. 16 E.	Section 07	Lot A	42
T. 23 N.	R. 16 E.	Section 08	NE1/4, NW1/4 less 13 acres covered by waters of the Coosa River	307.1
T. 23 N.	R. 16 E.	Section 10	W1/2 W1/2	160
T. 23 N.	R. 16 E.	Section 11	W1/2 NE1/4, NE1/4 NW1/4, S1/2 NW1/4, NW1/4 SW1/4, SE1/4 SW1/4, S1/2 SE1/4	361.1
T. 23 N.	R. 16 E.	Section 13	S1/2 NW1/4, NW1/4 NW1/4, W1/2 NE1/4 NW1/4, SW1/4	300
T. 23 N.	R. 16 E.	Section 14	All	641.1
T. 23 N.	R. 16 E.	Section 15	All	641.1
T. 23 N.	R. 16 E.	Section 16	NE1/4 NE1/4, SW1/4 NW1/4, W1/2 SW1/4, SE1/4 SW1/4, S1/2 SE1/4	280
T. 23 N.	R. 16 E.	Section 17	NE1/4 SW1/4, S1/2 S1/2 except that part covered by the waters of the Coosa River	193.1
T. 23 N.	R. 16 E.	Section 20	W1/2 NE1/4, N1/2 NW1/4, SE1/4 NW1/4, E1/2 SE1/4	280
T. 23 N.	R. 16 E.	Section 21	All, except 10 acres in the Southwest corner of the NW1/4 NW1/4	630
T. 23 N.	R. 16 E.	Section 22	All	640
T. 23 N.	R. 16 E.	Section 23	W1/2, W1/2 E1/2, NE1/4 NE1/4, SE1/4 SE1/4	560
T. 23 N.	R. 16 E.	Section 24	SW1/4, NW1/4 NW1/4	200.1
T. 23 N.	R. 16 E.	Section 25	All	640.1
T. 23 N.	R. 16 E.	Section 26	E1/2 NW1/4, NW1/4 NW1/4, 5 acres in the SW1/4 NW1/4, SE1/4 NE1/4, NE1/4 SE1/4, SW1/4	365.5
T. 23 N.	R. 16 E.	Section 27	NW1/4, W1/2 NE1/4, NW1/2 NE1/4 NE1/4 (the Northwest half of NE1/4 NE1/4)	260
T. 23 N.	R. 16 E.	Section 28	S1/2 NE1/4, NE1/4 NE1/4, W1/2 NW1/4, the North 30 acres of the NE1/4 NW1/4, S1/2	531.5
T. 23 N.	R. 16 E.	Section 29	All	641.6
T. 23 N.	R. 16 E.	Section 31	All of fractional section	10
T. 23 N.	R. 16 E.	Section 32	All of fractional section	620.32
T. 23 N.	R. 16 E.	Section 33	NW1/4, NW1/4 SW1/4, W1/2 NE1/4 SW1/4	220
T. 23 N.	R. 16 E.	Section 34	E1/2 NE1/4, N1/2 SE1/4, SE1/4 SE1/4, all of the SW/14 NE1/4 lying East of Clay Creek	230.8
T. 23 N.	R. 16 E.	Section 35	SW1/4 NE1/4, NW1/4, N1/2 SW1/4, SW1/4 SW1/4, NW1/4 SE1/4, S1/2 SE1/4 SE1/4	380
T. 23 N.	R. 16 E.	Section 36	All	640
T. 23 N.	R. 17 E.	Section 18	E1/2 SE1/4	80
T. 23 N.	R. 17 E.	Section 19	S1/2, NW1/4 NE1/4	360
T. 23 N.	R. 17 E.	Section 20	S1/2	320
T. 23 N.	R. 17 E.	Section 28	N1/2 N1/2, SW1/4 NE1/4, SE1/4 NW1/4, N1/2 SE1/4, SE1/4 SE1/4 E1/2 SW1/4, SW1/4 SW1/4	480.8
T. 23 N.	R. 17 E.	Section 29	All	641.1
T. 23 N.	R. 17 E.	Section 30	All	640
T. 23 N.	R. 17 E.	Section 31	All	640
T. 23 N.	R. 17 E.	Section 32	All	640
T. 23 N.	R. 17 E.	Section 33	NE1/4 NE1/4, NW1/4, SW1/4, S1/2 SE1/4, NE1/4 SE1/4	480
T. 23 N.	R. 17 E.	Section 34	SW1/4 SW1/4, SE1/4 SE1/4	80
T. 23 N.	R. 18 E.	Section 32	W1/2 NW1/4, NE1/4 NW1/4	120
T. 23 N.	R. 19 E.	Section 20	NW1/4 SW1/4	40
T. 24 N.	R. 16 E.	Section 32	NW1/4 NE1/4, S1/2 NE1/4, E1/2 NW1/4, SW1/4, N1/2 SE1/4, SE1/4 SE1/4	480

4.2.2 Royalties

A 2% NSR royalty is payable by AGC to the lessor from the commercial production and sale of graphite from the properties, as well as royalties for any precious metals, mica, iron, magnetite, manganese, calcium carbonate, copper, tantalum and rare earths commercially produced and sold from the properties.

A NSR of 0.5% up to a total of $150,000 is also payable to Merchant if and when graphite mining operations commence on the Leased Property.

4.2.3 Legal Access and Surface Rights

AGC has negotiated a Surface Use Agreement with the holder of the surface rights in an area of 3,481 acres (1,409 ha), Headwaters Investments Corporation, Atlanta, Georgia. The first period of the agreement was until 1 September 2013, and it was renewed for a second period until 30 September 2015. The agreement covers the area of the drill grid and allows the company to carry out exploration including sampling, trenching and drilling. Payments of $53,000 were made for the first period and $50,000 for the second period plus a guarantee of $20,000 placed in escrow. The payments were based on the amount of anticipated surface disturbance. This provides unrestricted access to the Property with the stipulation that AGC is responsible for reclamation of any surface disturbance caused by its exploration activities.

Headwater Investments Corp. has fee simple ownership of the surface rights. They are required to pay an annual property tax of $1.13 per acre. The property taxes have been paid in full to date and the property is in good standing.

4.2.4 Environmental Regulations and Permits for Exploration

Exploration, road access, and drilling at the Coosa project is subject to environmental permits from the Alabama Department of Environmental Management (ADEM) including a Construction storm water permit under the National Pollutant Discharge Elimination System (NPDES), and General Permit #ALR1000000 authorizing discharges. A Construction Best Management Practices Plan (CBMPP) was prepared for storm water management, erosion and sediment control. The authors know of no other environmental liabilities for which the Property is subject to.

4.2.5 Other

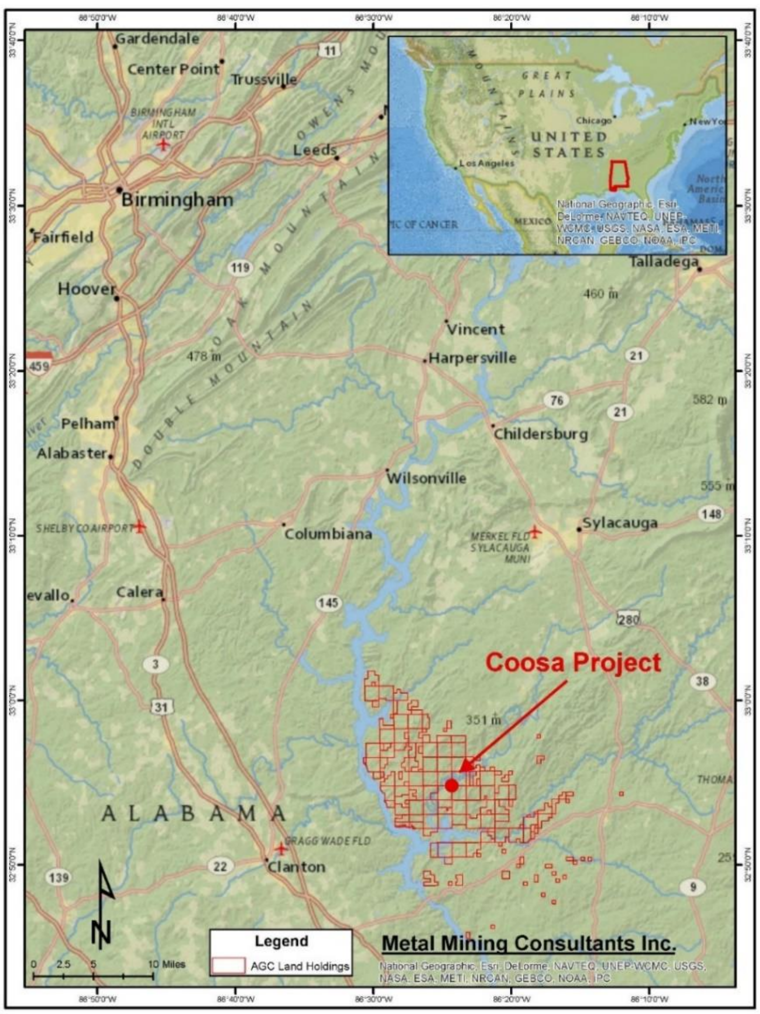
MMC knows of no other known significant factors and risks that may affect access, title or the right or ability to perform work on the property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Describe Access and Infrastructure

The Coosa Project is located in Western Coosa County, State of Alabama, USA. The property encompasses portions of townships T21N, T22N, T23N and T24N and ranges R16E, R17E, R18E and R19E. The western boundary is approximately the Coosa River. The center of the drill grid on the property is at 32°54’30’’ N, 86°24’00” W. The location and access are shown in Figure 5-1.

Figure 5-1: Access Map to the Coosa Project Drill Grid



The project is 50 mi (80 km) south-southeast of Birmingham (population 1.1 million in the metropolitan area). Access to the project is by driving southeast from Birmingham Airport on Highway US 280 for approximately 52 mi (84 km) to Sylacauga (population 12,749 at the 2010 census) which is the closest small city with hotels and services. AGC has its field office and core store here. It is called the "Marble City" due to large marble quarries and processing facilities nearby. From Sylacauga it is approximately 24 mi (39 km) by road to the Coosa project. The driving time from Birmingham to the project is approximately 90 to 120 minutes, and from Sylacauga to the property it is 45 to 60 minutes. The access route is given in Table 5-1.

Table 5-1: Access from Birmingham to Coosa Project

From	To	Highway	Distance (mi)	Distance (km)
Birmingham Airport	Sylacauga	US 280	52	84
Sylacauga	Stewartville	US 231	7	11
Stewartville	Weogufka	County Road 56	6	10
Weogufka	Mt. Moriah Church	County Road 29	7.5	12
Mt. Moriah Church	Mt. Moriah Cemetery	County Road 15	0.25	0.4
Mt. Moriah Cemetery	Drill grid	Forest roads	3	5
Total			75.75	122.4

Note: Conversions and totals may not be exact due to rounding.

The nearest major airport with scheduled flights is in Birmingham, and there is an airfield at Sylacauga.

The property is close to centers of population which could supply the workforce and logistical needs of a graphite mine. The area within 10 mi (16 km) of the property is very sparsely populated, so a mine would directly affect very few people. Mining has been a traditional industry in the area, and marble quarries are still active.

There is no infrastructure in the immediate area of the property, other than a network of well-maintained logging access roads. An electrical transmission line occurs approximately 1 mi (1.6 km) west of the drill grid. Water is abundant in small streams and in Mitchell Lake, a large impoundment on the Coosa River at the western edge of the property.

5.2 Physiography

The property is located in an area of ridges and valleys with elevations of approximately 350 to 1200 ft. (107 to 370 m). The area of the drill grid is a peneplain surface dipping gently south at an elevation of 600 to 700 ft. (183 to 213 m), and incised by a meandering river system to give a local relief of up to 300 ft. (91 m). The elevations of the drill collars vary from approximately 351 to 766 ft. (107 to 233 m).

The land use is forestry with thick mixed hardwoods and pines, and significant areas of pine plantations which are harvested every 20 to 25 years. There are clearings at towns and for agriculture. Due to extensive weathering, natural outcrops are sparse and most useful exposures are in road cuts, stream drainages, or old mine workings.

The Weogufka Creek runs from northeast to southwest on the north side of the drill grid, and joins Swamo Creek, the Coosa River, and the Mitchell Lake dam. This is a tributary of the Alabama River which flows southwest, becomes the Mobile River, and then flows into the Gulf of Mexico at Mobile.

Figure 5-2: Overview of the Physiography of the Drill Grid Looking Southwest from near Hole AGC-011, Showing Dissected Peneplain and Pine Forest

Photo: S. Redwood

5.3 Climate

The climate zone of Alabama is classified as humid subtropical (Cfa) under the Koppen Climate Classification. The Holdridge Life Zones Climate Classification is subtropical moist forest.

The nearest climatic data available is for Sylacauga, 20 mi (32 km) northeast of the project, which is summarized in Table 5-2. It has an average temperature of 61.7°F (16.5°C). The warmest month is July with an average temperature of 78.7°F (25.9°C). The coolest month on average is January with an average temperature of 43.2°F (6.2°C). The maximum average high is 90.6°F (32.6°C) in July, and the minimum average low is 31.4°F (-0.3°C) in January. Average annual precipitation is 55.2 in (140.2 cm). It rains all year with the wettest months being March with an average of 5.9 in (15.0 cm). Snow is rare with an average of 0.8 in (2.0

cm). It is humid in the summer. The state is prone to tropical storms, hurricanes, thunderstorms, and tornadoes. Due to the moderate climate of this region of United States, the operating season is year round.

Table 5-2: Average Annual Climatic Data for Sylacauga

Month	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Maximum recorded temperature (°F)	81	83	90	91	99	100	102	104	98	92	86	79	104
Average high temperature (°F)	55.0	59.3	67.3	75.5	82.7	87.9	90.6	90.3	85.3	76.2	66.2	57.4	74.5
Average temperature (°F)	43.2	46.6	53.9	61.5	69.4	75.5	78.7	72.8	72.8	62.2	52.7	45.4	61.7
Average low temperature (°F)	31.4	34.0	40.5	47.3	56.1	63.1	66.7	66.0	60.3	48.2	39.0	33.4	48.8
Minimum recorded temperature (°F)	-4	4	10	25	33	39	50	45	32	23	14	3	-4
Precipitation (inches)	5.1	5.2	5.9	4.7	4.1	4.3	5.2	4.1	4.1	3.1	4.3	5.0	55.2
Days with rain	10	9	9	8	8	9	11	9	7	6	8	9	104

Source: www.weatherbase.com.

6 HISTORY

6.1 The Alabama Graphite Belt

The following Sections 6.1 and 6.2 describe the history of graphite mining and exploration in the Alabama Graphite Belt and is provided for background information on the regional setting of the Coosa Project. The information is taken from published sources as summarized by Pallister & Thoenen (1948) and Durgin (2013). These are not necessarily an indication of the mineralization that occurs on the Coosa Project.

The Alabama Graphite Belt is a 70 mi (112 km) long, northeast-trending belt in Clay, Coosa and Chilton Counties, shown in a historic map in Figure 6-1. Most of the historical mines were in the northeast segment around Ashland, Clay County (Area A of Pallister & Thoenen, 1948), approximately 18 mi (29 km) long, separated by an 11 mi (18 km) gap from the 40 mi (64 km) long, southwest segment extending from Goodwater, Coosa County, to Verbena, Chilton County (Areas B and C). The geology of the graphite deposits at Ashland was described by Brown (1925). The Coosa Project occurs beside the historic Fixico Mine in the southwestern part of the belt (Area C), shown in a historic map in Figure 6-2.

Figure 6-1: Historic Map of the Alabama Graphite Belt (Cameron & Weiss, 1960, after Pallister & Thoenen, 1948) Showing the Location of the Coosa Project

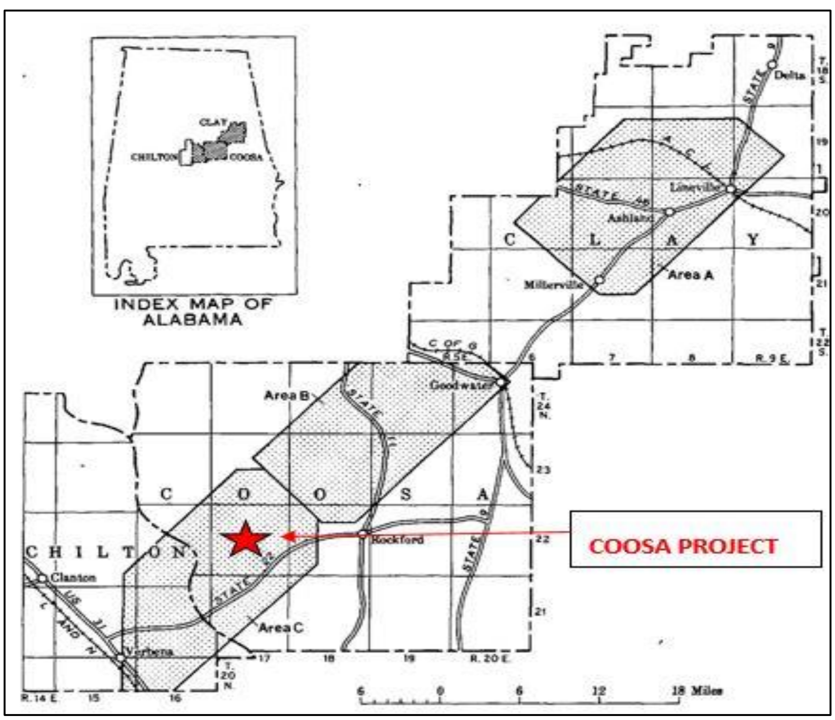
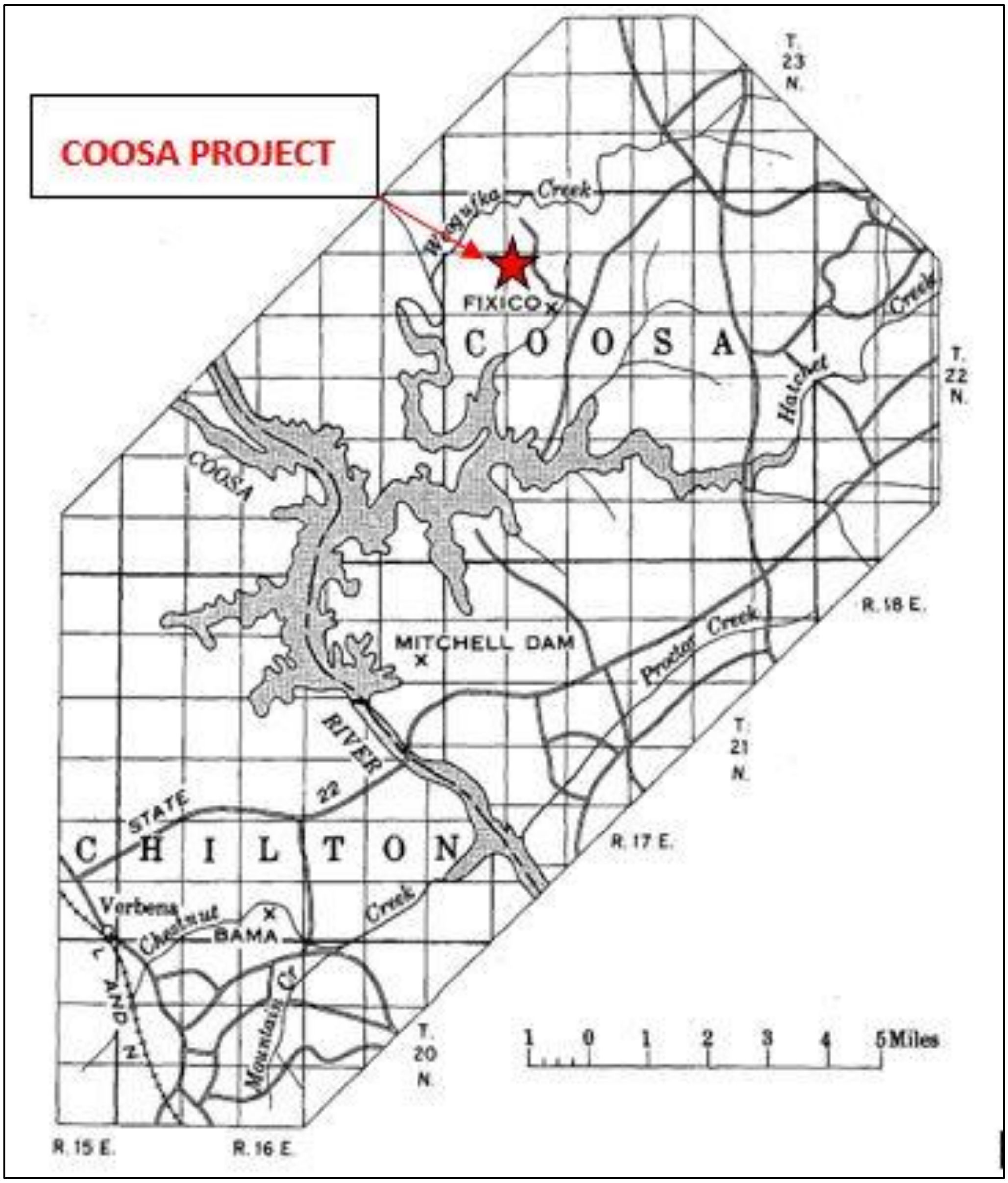


Figure 6-2: Historic Map of the Graphite Mines of Area C in the SW Part of the Alabama Graphite Belt (Cameron & Weiss, 1960, after Pallister & Thoenen, 1948) Showing the Location of the Coosa Project



The presence of graphitic schists in Alabama was recognized by M. Tuomey before the Civil War (1861-65; Jones, 1929). Dr. Gessner, employed by the Confederate Government to recover sulfur from the pyrite deposit at Pyriton, is credited with the first discovery of flake graphite in Alabama (Clemmer et al., 1941). The first attempt at development was in 1888, but an experimental mill using water for flotation was unsuccessful. The first commercial graphite operation was in 1899 when the Allen Graphite Company built a mill using a patented oil flotation process and produced the first refined graphite in Alabama, at a mill near the Quenelda Deposits in Clay County.

Shortly after that, the Fixico Mine in Coosa County, adjacent to the Coosa project, and the Dixie Mine in Chilton County began operations. By 1906 there were several mines in operation and by 1913 the graphite industry was well established in central Alabama. World War I caused the disruption of foreign graphite imports, leading to significantly higher prices and the Alabama graphite industry boomed. By 1918 there were 25 flotation plants operating in the district with a total production in 1918 of 7.8 million pounds (3.5 million kg) of graphite (Pallister & Thoenen, 1948).

However, the end of the war and the resumption of foreign imports depressed prices and the Alabama graphite industry dwindled to seven operating plants in 1920. In 1929 the last two mines in the district, the Ceylon Graphite Company in Coosa County and the Superior Flake Plant in Clay County, were closed.

At the start of World War II in 1939, C. J. Johnson rebuilt the mill at the Ceylon Mine and began production of flake graphite on a small scale. In 1940, as a result of the interruption of graphite imports from Madagascar, the US Bureau of Mines made a preliminary survey of the Alabama Graphite deposits to determine the viability of resumption of mining, with a report by Clemmer et al. (1941). As the demand for graphite rose, the War Production Board, the Metals Reserve Company, and the Reconstruction Finance Corporation (all federally funded) turned their attention to the district, and a detailed study of the district was carried out between 1942 and 1944 which resulted in a report by Pallister & Thoenen (1948). A field laboratory was established in Ashland with the capacity for pilot crushing-milling-flotation testing and graphite analyses. In 1943 the Crucible Flake Mill of Haile Mines, Inc. and the Alabama Flake Graphite Co. plant (Gisler, 1943) began to produce; the three plants produced 8.1 million lbs (3.7 million kg) of graphite, with Alabama again ranking first in graphite production in the US. The Crucible Flake Mill was closed at the end of 1943, leaving only two producers in the district.

A thorough review was made of all the known graphite producing areas in the district in 1942-44 (Pallister & Thoenen, 1948). The graphitic deposits were found to contain significant amounts of green vanadium-bearing mica (roscoelite). A total of 11 mining areas including 49 graphite deposits were mapped and studied in detail for graphite and vanadium. More than 9 mi (14.5 km) of access roads were built; 17,930 ft. (5,457 m) of bulldozer trenches, 2,670 ft. of power scoop trenches, 5,234 ft. (1,596 m) of hand trenches and 3,279 ft. (1,000 m) of existing trenches were dug and sampled. Diamond drilling totaled 5,453 ft. (1,662 m) in 84 holes (Pallister & Thoenen, 1948). This resulted in a historical mineral resource estimate of 25,910,000 tons in all classes. This included 11,059,000 tons of "measured weathered ore reserves", 3,351,000 tons of "Inferred weathered ore", and 11,500,000 tons of "unweathered ore" in "Measured", "Indicated" and "Inferred" categories, with an average recoverable grade in the "Measured weathered ore" estimated at 3% graphite (Pallister &

Thoenen, 1948, p. 77). These estimates are historical in nature as they predate the introduction of the CIM standards and NI 43-101, and are quoted for information purposes only. These are mineral resource estimates rather than mineral reserve estimates. The assumptions, parameters and methods used to make these estimates are not stated in the original publication. The historical resources include multiple deposits throughout the Alabama Graphite District. They were made to evaluate the potential of the whole district. A qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves, and the Company is not treating the historical resource estimate as current mineral resources or mineral reserves.

After World War II, production again declined rapidly due to the resumption of imports and subsequent decline in prices. By 1950 only the Pocahontas Mine near Ashland and the Bama Mine in Chilton County were still in production, and they closed in 1953. The Alabama graphite industry has been idle since that time. The processing plants have all been dismantled, burned down, or overgrown, and the graphite workings are hidden under more than 60 years' vegetation. With the current resurgence of interest in graphite deposits, attention is being turned again to the central Alabama graphite district.

Historic production figures are incomplete. There is little data for the period 1898 to 1912. The Allen Graphite Company in 1899 "was able to produce several hundred tons of graphite" (Jones, 1929). In 1917-19 Alabama ranked first in production of flake graphite in the USA, and again in 1943 (Pallister & Thoenen, 1948). Graphite production between 1913 and 1920 is shown in Table 6-1.

Table 6-1: Alabama Graphite Production 1913-1920 (Clemmer et al., 1941)

Year	No. Producers	Pounds produced	Value $
1913	3	2,020,910	87,336
1914	3	2,410,200	118,000
1915	4	3,374,800	204,572
1916	4	5,226,940	492,407
1917	14	6,223,095	719,575
1918	30	7,795,475	999,152
1919	10	3,569,030	272,413
1920	7	4,894,648	306,977

Between 1920 and 1929 production figures are not available. In 1929 three mills produced 3.5 million lbs (1.6 million kg) of graphite, but all were closed by 1930. In 1943 a total of 8.1 million lbs (3.7 million kg) were produced, and two mills continued to produce until 1953. Production statistics are unavailable.

Mineable grade graphite bodies were greatly elongated along strike and were referred to as "leads", and often included selvages of waste rock. Cameron & Weiss (1960, p 263) described the leads as follows:

"The total length of some of the larger leads may be measureable in miles, but the full length of a lead is not necessarily ore, for there is much variation in graphite content both along and across strike. A typical section shows alternating layers of rocks of low graphite content, of high graphite content, and of intermediate graphite content. The individual layers range from fractions of an inch (<1 cm) to 10 feet (3 m) in thickness, with aggregate thickness in some cases greater than 100 feet (30 m). Exposures outside the mine workings are poor and none of the individual leads is exposed. The Pocahontas lead was traced by trenching along strike for approximately 4,000 feet (1,220 m) and Brown (1925) reported that he was able to trace a lead on the Griesemer property for a like distance."

Two good examples of mines in the Ashland area are the Quenelda Mine and the Pocahontas Mine. At the Quenelda there were seven pits spaced along the strike of the leads for approximately 2,000 ft. (610 m). The pits were almost entirely in weathered mineralization along two leads 20 to 40 ft. (6.1 to 12.2 m) apart. The southeast lead was 40 to 65 ft. (12.2 to 19.8 m) thick and the northwest lead was 50 to 80 ft. (15.2 to 24.4 m) thick. The strike was N55-75°E and the dip is 55-85° to the southeast. Graphite grades averaged approximately 3%.

The Pocahontas Mine of the Alabama Flake Graphite Company was one of the last operating mines in Alabama. The workings were three parallel open cuts in which the leads were approximately 50 ft. (15.2 m) wide and several hundred feet (a few hundred m) long. The depth of weathering could exceed 80 ft. (24.4 m). Pallister & Thoenen (1948) indicated that the lead was shown to extend for another 2,400 ft. (732 m) southwest from the third cut. The grade was reported to average 4 to 5% graphite in later years.

The two most significant mines in the southwestern portion of the Alabama Graphite Belt were the Ceylon and Bama mines. The Ceylon deposit is located approximately 8 mi (12.9 km) west of the town of Goodwater. One of the largest mines in the district, it operated from 1916 to 1929 and from 1939 to 1947. The principal working developed during the later period was 925 ft. (282 m) long, up to 300 ft. (91 m) wide and up to 70 ft. (21.3 m) deep on a series of leads striking N45°E and dipping 55°SE. The grade ranged from 2 to 6% graphite and averaged 3%.

The Bama Mine in Chilton County near the southwest end of the graphite belt operated from 1925 to 1930 when the mill burned down. The main pit was 625 ft. (190 m) long, 150 ft. (45.7 m) wide and 40 to 80 ft. (12.2 to 24.4 m) deep. Two smaller pits approximately 200 ft. (61 m) long were mined along strike between the main pit and the mill. The deposits trend N20-25°W and dip 50-60°S, due to the presence of a large fold.

Figure 6-3: Crucible Flake Graphite Mill, Haile Mines, Inc., Clay County in the 1940s (Pallister & Thoenen, 1948)

Figure 6-4: Ceylon Graphite Co. Mill, Coosa County, in the 1940s (Pallister & Thoenen, 1948)

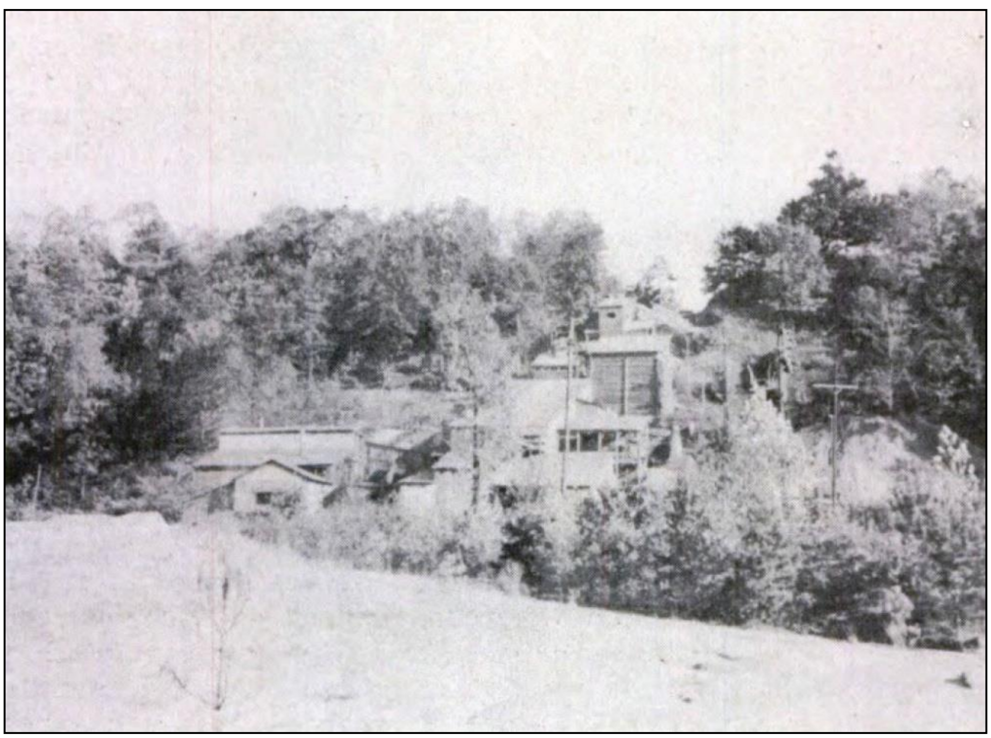


Figure 6-5: Ceylon Graphite Co. Pit, Coosa County, in the 1940s (Pallister & Thoenen, 1948)

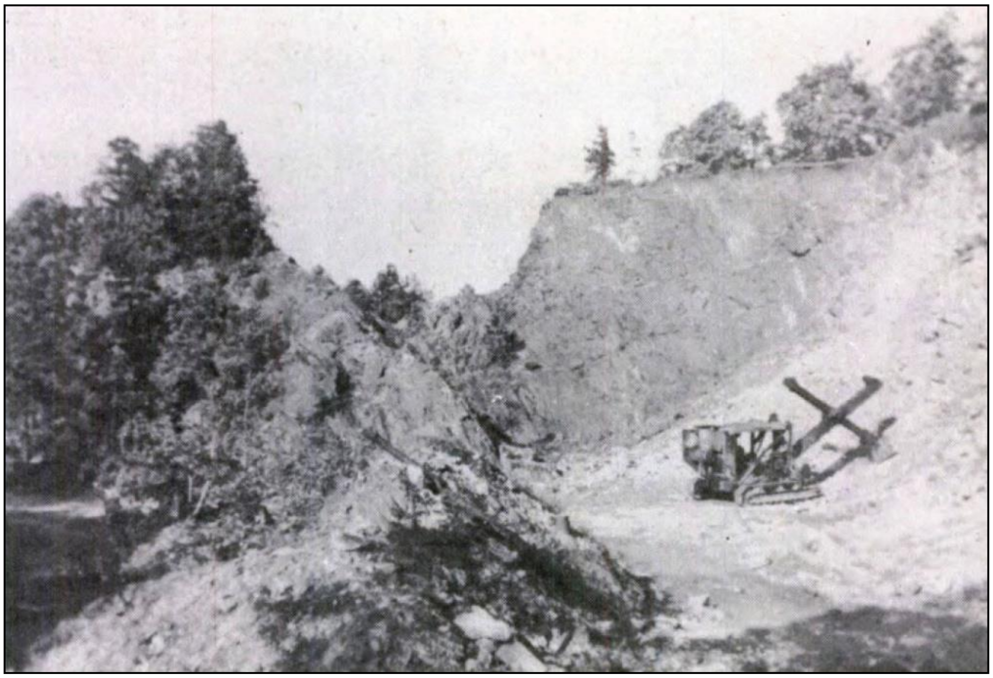
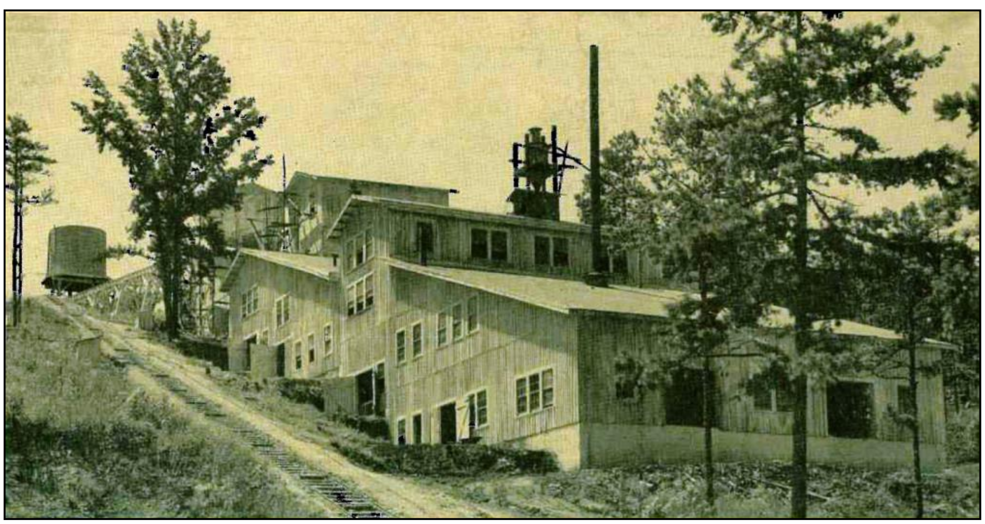


Figure 6-6: Alabama Flake Graphite Company Mill, Clay County, in 1945 (Gisler, 1945)



6.2 Coosa History

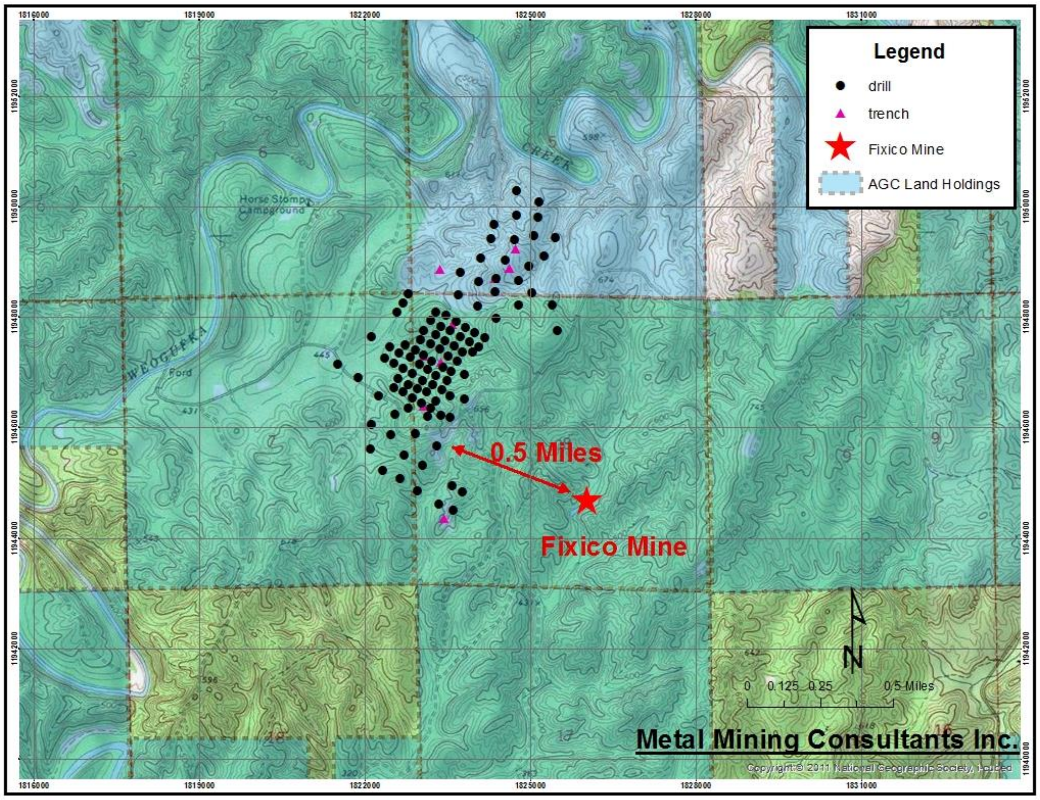
The Fixico Mine was located approximately 0.5 mi (850 m) southeast of the Coosa Project drill grid and operated from 1902 to 1908. The location is shown in Figure 6-7. There is no record of the amount of graphite and grade produced. AGC geologists have located the old pits and the old wooden dam described below. The mine was described by Pallister & Thoenen (1948, p. 72-3):

> *"The Fixico graphite mine, named for the Indian chief Fixico, who lived near the mine, is now owned by C. A. Dean (Alexander City, Ala.). It is in the southern part of sec. 8, T. 22 N., R. 17 E., 11 miles west of Rockford. C. F. Whoolock, who first used a "flotation" process to separate graphite in 1898 at the A. A. Allen plant in Clay County, operated the Fixico mine with his son Kennard. About 1902, they built a log dam, which is still intact, a mill, and houses for their employees and operated the mine until about 1908. It is thus probably the oldest mine in Coosa County.*
>
> *"The graphitic schist that was mined is exposed on the western slope of a tributary of Hatchett Creek, the backwater of which, caused by the Mitchell Dam, is not far from the property. The western slope rises for over 100 feet (30 m) above the creek level, and the pits open off a small cross valley. Mine pits were opened at various levels and running at angles from northeast through west to southwest. The largest pit is about 10 feet (3 m) above the creek and extends northeast for a distance of 150 feet (45 m). It is 75 feet (23 m) wide and 50 feet (15 m) deep at the face. Sample A-73 was taken across 40 feet (12 m) of this face. A short distance to the northwest are two parallel pits each 100 feet long (30 m), 25 to 30 feet (8 to 9 m) wide, and 30 feet deep (9 m) with a narrow "horse" between them. The floor of these pits is 40 feet (12 m) higher than the floor of the largest pit. Sample A-74 was taken across 25 feet (8 m) of the face of the southeast of these two pits. Four additional pits cut across the top of the slope towards the west, and two more extend southwest at a lower level opposite the northeast pits.*
>
> *"The schist has a general N45°E strike and 45°SE dip. The beds of graphitic schist extend for 150 to 200 feet (46 to 61 m) in thickness, with barren layers between the schist beds. The country to the northeast rises for 400 to 500 feet (122 to 152 m) before it drops off, giving room for large possible reserves."*

Sample A-73 had a grade of 119.04 lbs per ton graphite (5.95% Cg), with 23.08 lb/t (1.15%) +50 mesh, 46.84 lb/t (2.34%) -50+100 mesh, and 49.12 lb/t (2.46%) -100 mesh. Sample A-74 had a grade of 50.92 lb/t (2.55%) with 5.56 lb/t (0.28%) +50 mesh, 18.04 lb/t (0.90%) -50+100 mesh, and 27.32 lb/t (1.37%) -100 mesh (Pallister & Thoenen (1948, p. 72-3).

Sample A-73 gave 8.3% mica concentrate with a grade of 0.31% V_2O_5, and sample A-74 gave 8.8% mica concentrate with 0.08% V_2O_5 (Pallister & Thoenen (1948, p. 72-3).

Figure 6-7: Location Map of the Historic Fixico Mine and the Coosa Project Drill Holes



The long and complicated history of ownership of the mineral and surface rights of the Coosa Project is described in a title opinion made by John Foster Tyra, PC, Tuscaloosa, Alabama, for Sections 5, 7, 8 and 9, Township 22 N, range 17 E, where AGC has carried out exploration, in reports to AGC dated 17 July 2012 ("Preliminary Mining Title Opinion") and 26 October 2012 ("Alabama Graphite Option"). A brief summary is given here. The mineral and surface rights of these four sections are patents of private land issued between 1842 and 1860. Most of them were acquired by the Fixico Mining Company in 1901-02. Several other companies and individuals owned other parts. Following bankruptcy of Fixico in 1910, their mineral and surface rights were acquired by John R. Hegeman. In 1929 the mineral rights were acquired by Charles A. Dean (senior). He acquired more mineral rights over the years and on his death in 1952, they passed to multiple heirs who sold a half interest to Robert A. Russell. In 1980 Charles A. Dean, Junior, acquired all of the mineral rights from the other Dean heirs and the Russell heirs. On the death of Charles A. Dean, Junior, the mineral rights were inherited by his wife, Eugenia W. Dean, in 2008, and following her death in 2014, they were inherited by the Estate of Eugenia W. Dean.

Figure 6-8: Old Log Dam at the Fixico Mine Built in 1902

Photo: S. Redwood

Figure 6-9: One of the Overgrown Open Pits at the Fixico Mine Exploited in 1902-08

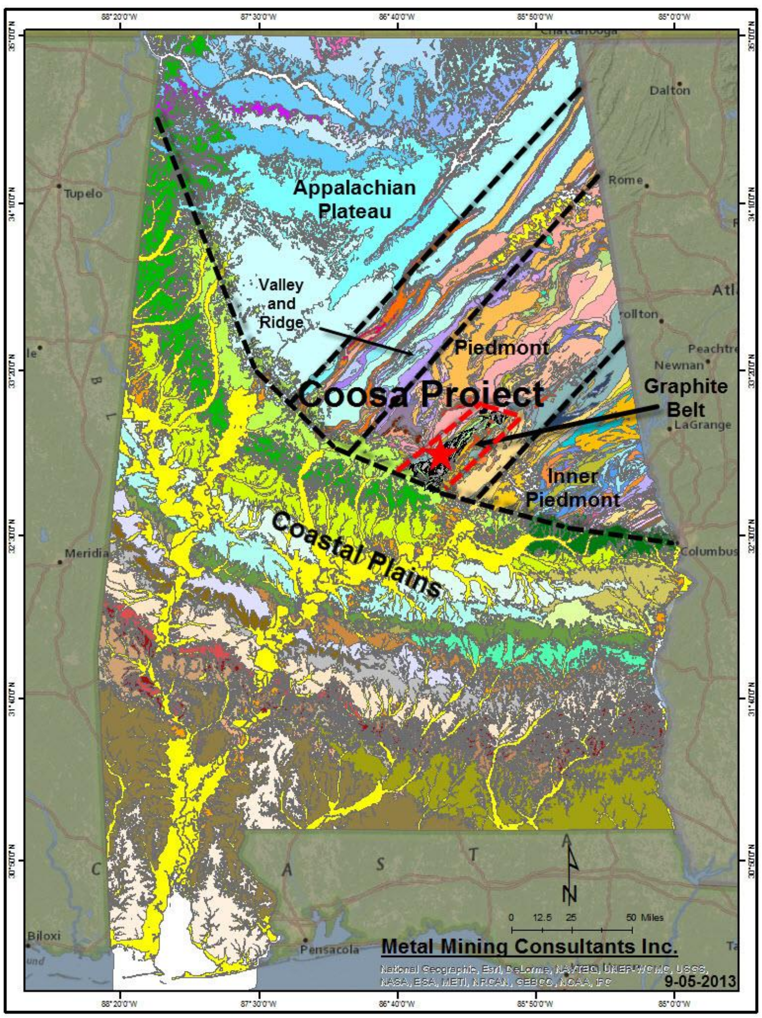
Photo: S. Redwood

7 GEOLOGICAL SETTING

7.1 Regional Geology

The Coosa Project is located at the southern end of the Appalachian mountain range, a northeast trending belt of folded and metamorphosed rocks of Neoproterozoic to Lower Paleozoic age. These are covered by the onlap Coastal Plain Sediments of Cretaceous and younger age in the southern half of Alabama. The geology of Alabama is shown in Figure 7-1. The rocks of the southwestern end of the Appalachians are generally separated into four physiographic and geologic provinces which are, from northwest to southeast: The Interior Low Plateau province, the Appalachian Plateau province, the Valley and Ridge province, and the Piedmont province. The Alabama Graphite Belt is located in the Northern Piedmont.

Figure 7-1: Geological Map of Alabama (Alabama Geological Survey)



The regional geology is described by Raymond et al. (1988) in “Alabama Stratigraphy”. The Interior Low Plateau is a Paleozoic limestone plateau of moderate relief. The Appalachian Plateau is underlain by a thick series of carbonates overlain by sandstones and shales of Cambrian to Pennsylvanian age. The rocks have open folding and are moderately dissected resulting in sandstone and shale synclinal plateaus and, in the eastern part, three linear anticlinal limestone valleys. The Valley and Ridge Province is a fold-thrust belt, with east-dipping thrusts, of carbonates, sandstones, and shales of Cambrian to Pennsylvanian age, similar to the stratigraphy of the Appalachian and Interior Low Plateaus. It is separated from the Appalachian Plateau by a large scale east-dipping thrust fault, and consists of a series of subparallel ridges and valleys.

The Piedmont Province is formed of Neoproterozoic to early Paleozoic metamorphic rocks. The metamorphic grade increases across the Piedmont from lower greenschist facies in the northwest to high grade migmatite facies in the southeast. It is divided into three lithotectonic provinces which are, from northwest to southeast: The Northern Piedmont, Inner Piedmont, and Southern Piedmont, each bounded by major faults. The physiography of the Northern Piedmont is characterized by prominent ridges with peaks up to 2,407 ft. (734 m), becoming lower to the southeast. The Inner and Southern Piedmonts have much more subdued topography.

The Northern Piedmont has three structural blocks:

- The Talladega Block on the northwestern side with low grade greenschist facies metasedimentary and metavolcanic rocks (marble, phyllite, sandstone, chert, quartzite, greenstones).
- The central Coosa Block with high grade, upper greenschist to kyanite and sillimanite grade metamorphic rocks (phyllite, schist, graphite schist, gneiss, migmatitic gneiss, quartzite, amphibolites), including the Alabama Graphite Belt, and abundant pegmatite and small granitoid bodies.
- The Tallapoosa Block on the southeastern side of high grade, middle to upper amphibolite facies metasedimentary rocks (phyllite, gneiss), metavolcanic (amphibolite) and metamorphosed ultramafic and mafic rocks (pyroxenite, gabbro), with large areas of quartz diorite to granitic plutonic rocks.

The Coosa block is thrust over the younger, lower-greenschist facies metamorphic rocks of the Talladega Block along the Hollins Line Fault, located 3 mi (4.7 km) northeast of the Coosa graphite deposit.

The Inner Piedmont includes two groups of high grade metamorphic rocks, including schists, gneisses and amphibolites, the Dadeville (schist, amphibolite, gneiss, granitic gneiss) and Opelika (schist, gneiss) complexes, with pyroxenite lenses and deformed granites.

The Southern Piedmont occupies the southeastern corner of the region. It is also underlain by high grade metamorphic rocks of the Pine Mountain, Wacoochee and Uchee complexes.

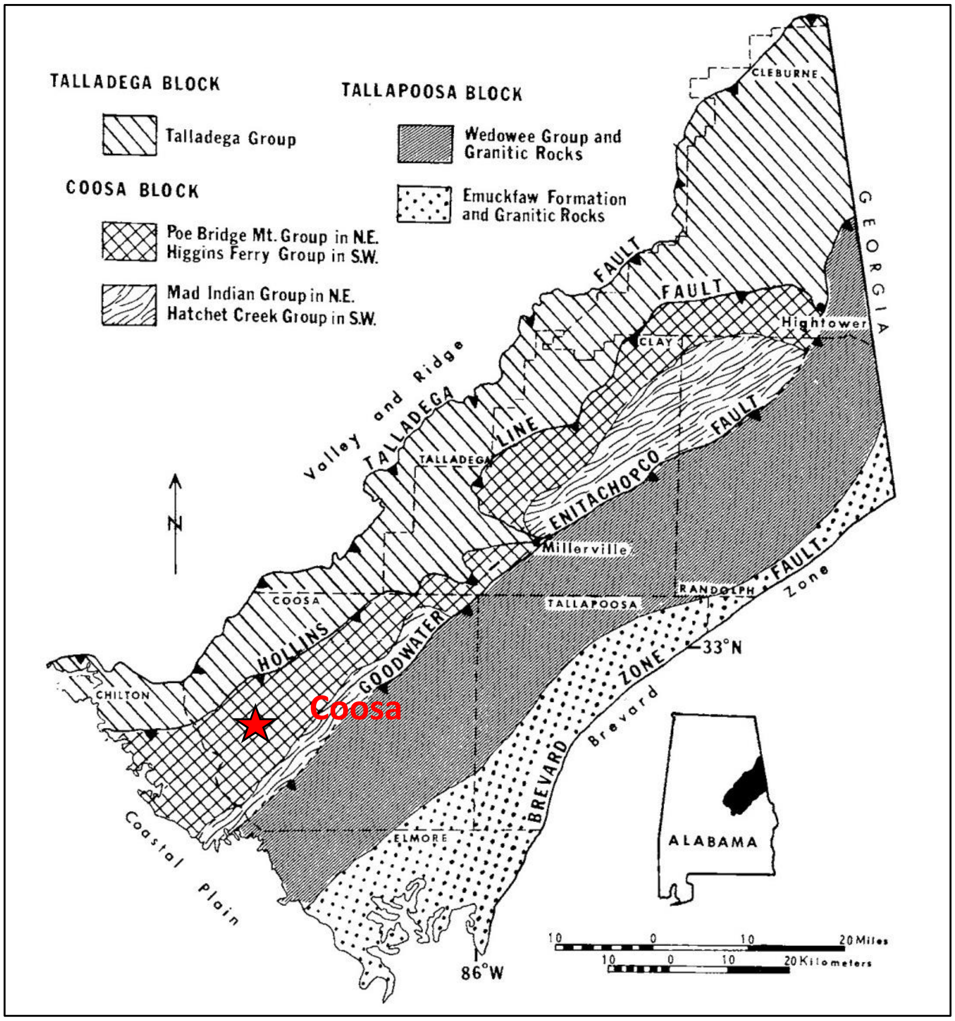
The Pine Mountain Block contains quartzite, quartzitic schists, and dolomitic marble. The Wacoochee is largely granitic gneiss and feldspathic muscovite-biotite schist. The Uchee contains a dioritic gneiss and a leucocratic quartz diorite. Folding is much less evident there.

Regionally, the Coosa block is interpreted to be part of the Eastern Blue Ridge terrane which formed on the rifted margin of Laurentia on the breakup of the Rodinia super-continent in the Neoproterozoic, and consists of rifted margin metasedimentary and rift-related volcanic rocks (Hatcher, 2010). The southern part underwent metamorphism to upper amphibolite facies in the Taconian orogeny at 460 to 455 Ma (Upper Ordovician) and is bounded on the west side by the Taconian suture (Hollins Line Fault) which can be traced for the length of the Appalachians, with different names. The Western Blue Ridge terrane including the Talladega belt is a Laurentian margin terrane of Neoproterozoic to Lower Carboniferous (Mississippian) age which was deformed and accreted in the Devonian to Mississippian Neoacadian orogeny (Hatcher, 2010).

7.2 Alabama Graphite Belt

The Alabama Graphite Belt is located in the Coosa block of the Northern Piedmont province in the Higgins Ferry Group and Poe Bridge Mountains Group of schists which are shown in Figure 7-2.

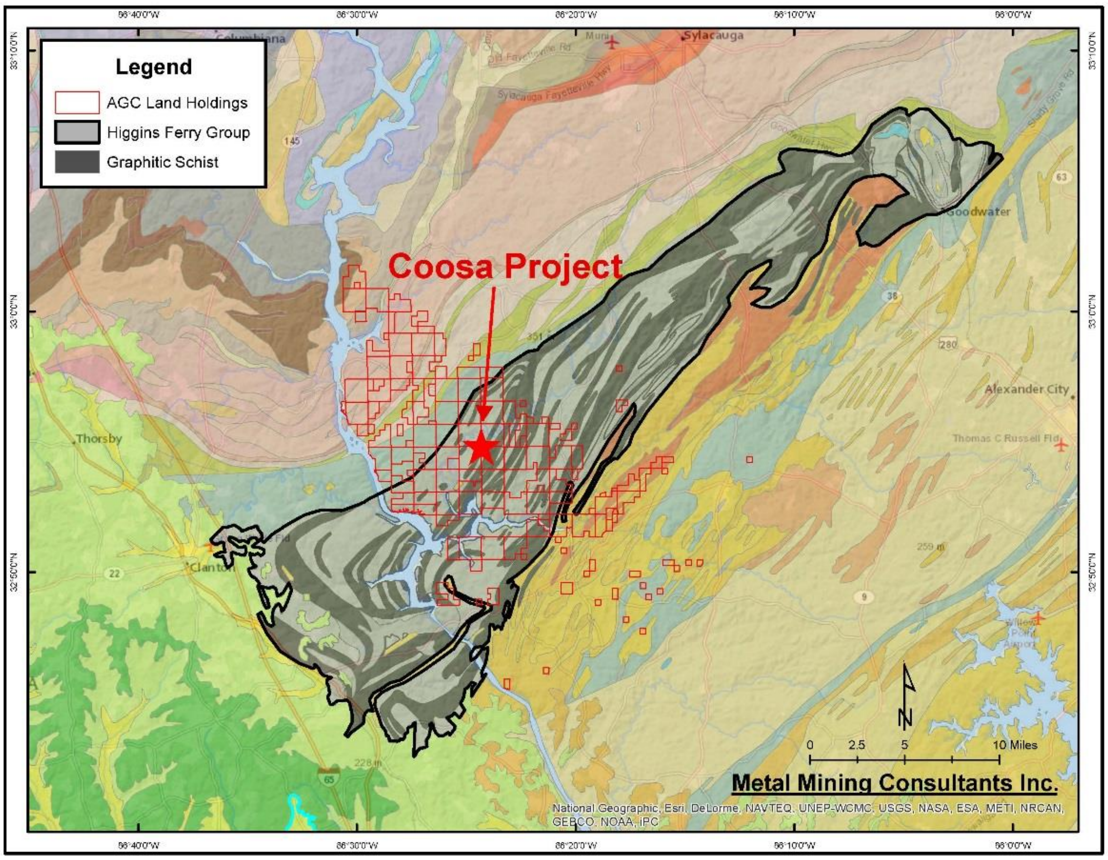
Figure 7-2: Northern Piedmont of Alabama Showing the Coosa Block and the Coosa Project (Tull, 1978)



The Coosa Block is divided into two zones or salients by the Goodwater - Enitachopco Fault, each with similar lithological units and comparable structural and metamorphic styles. The northeast salient is divided into the Poe Bridge Mountain Group to the northwest and the Mad Indian Group to the southeast. The southwest salient is divided into the Wedowee Group, the Higgins Ferry Group, and the Hatchet Creek Group. The stratigraphy was defined by mapping by the Alabama Geological Survey in the late 1960s and early 1970s (unpublished maps by Neathery, 1969, 1970, 1972) and is described in publications by Neathery (1975), Tull (1975, 1978) and Neathery & Tull (1975). These maps were used to compile a regional geological map at 1:250,000 scale published in 1988, the “Geologic Map of Alabama, Northeast Sheet” published by the Geological Survey of Alabama (Special Map No. 220, Osborne et al., 1988), part of which is shown in Figure 7-3, which is the only published geological map of the area.

The district scale geology of the Coosa project has been mapped at 1:24,000 scale on an unpublished map of the Mitchell Dam NW Seven Minute Quadrangle by D. Allison, University of South Alabama (Mobile, Alabama). He is collaborating with Alabama Graphite to integrate company geological data and carry out further detailed mapping in order to gain a better understanding of the geology of the district.

Figure 7-3: The Geology of the Southern Zone or Salient of the Alabama Graphite Belt Showing the Location of Coosa Project (Based on Alabama Geological Survey Special Map No. 220 at 1:250,000 Scale)



The Wedowee Group is formed of a thick sequence of quartz-graphite-sericite phyllite to fine-grained schist and chlorite-sericite phyllite to fine grained schist. The Poe Bridge Mountain and Higgins Ferry Groups contain distinctive interbedded sequences of interlayered coarse grained, commonly retrograded graphitic feldspathic mica schist, graphitic and garnetiferous quartzite, garnet mica schist, fine grained biotite gneiss (metagraywacke), and quartzite. There are two major amphibolite sequences associated with these groups, the Ketchepedrakee Amphibolite with the Poe Bridge Mountain Group, and the Mitchell Dam Amphibolite with the Higgins Ferry Group. Chemically these resemble tholeiitic basalts, suggesting formation in a back-arc basin. Hornblende from the Mitchell Dam Amphibolite near the Coosa River gave a K-Ar date of 348 Ma (Lower Mississippian), which was interpreted as the age of metamorphism (Russell, 1975). Highly manganiferous exhalites are associated with the amphibolites. The Mad Indian and Hatchet Creek Groups are much more homogenous in composition and consist of feldspathic garnet-quartz-muscovite schist, minor biotite-(garnet) schist and gneiss, micaceous quartzite, migmatitic gneiss and rare amphibolite. The two sequences contain abundant pegmatite and small granitoid bodies.

The stratigraphic description of the Higgins Ferry Group is an interbedded sequence of at least three major lithologic units:

- coarse to fine grained biotite-feldspar-quartz gneiss, sericite-feldspar-muscovite schist, (±biotite)-(±garnet)-muscovite schist, and biotite-garnet-feldspar gneiss;
- (±roscoelite)-graphite-quartz schists and graphitic quartzite;
- garnet quartzite and garnetiferous altered mafic rocks.

Locally there are a few thin actinolite-chlorite-hornblende amphibolite and pyroxenite bodies (Mitchell Dam Amphibolite). Kyanite, sillimanite and sericite porphyroblasts are common in more aluminous units. There are scattered pegmatites. (Neathery, 1975; Raymond et al., 1988).

The metamorphic grade ranges from kyanite to sillimanite, with local areas of retrograde upper greenschist facies mineral assemblages. The metamorphic rocks have a polyphase deformational history which was defined by Tull (1978) and is summarized in Table 7-1.

Table 7-1: Summary of the Multiple Phases of Deformation in the Northern Piedmont of Alabama (Tull, 1978)

Deformation	Metamorphism	Foliation	Folds	Lineations	Faults
D_1 Prograde dynamothermal metamorphism	M_1 Middle to upper amphibolite facies	S_1 Pervasive foliation schistosity parallel to F_1 axial planes. Parallel to compositional layering S_0 (bedding) except in F_1 fold hinges.	F_1 Tight to isoclinal flow folds defined by compositional layering S_0 (bedding) ranging from fine laminations to units 100s m thick, and folding of dikes and pegmatites.	L_1 Mineral lineation L_{1m} in plane of S_1 foliation. Defined by biotite, hornblende, quartz, graphite or other minerals. Parallel to F_1 fold axes.	
D_2 Retrograde dynamothermal metamorphism	M_2 Greenschist facies (phyllites)	S_2 Local transposition of S_1 into S_2 phyllonitic foliation	F_2 Microscopic crenulation foliation and shearing		
D_3 Dynamic deformation		S_3 Crenulation cleavage locally	F_3 S_1 and S_2 folded into tight flexural slip folds F_3, vary from crenulations to megascopic folds >1 km wavelength		
D_4 Faulting					Hollins Line Fault, reverse fault, dips to SE
D_5 Folding			F_4 Open flexural slip folds deform Hollins Line Fault		
D_6 Faulting					Goodwater-Enitachopco Fault, reverse fault, dips to SE

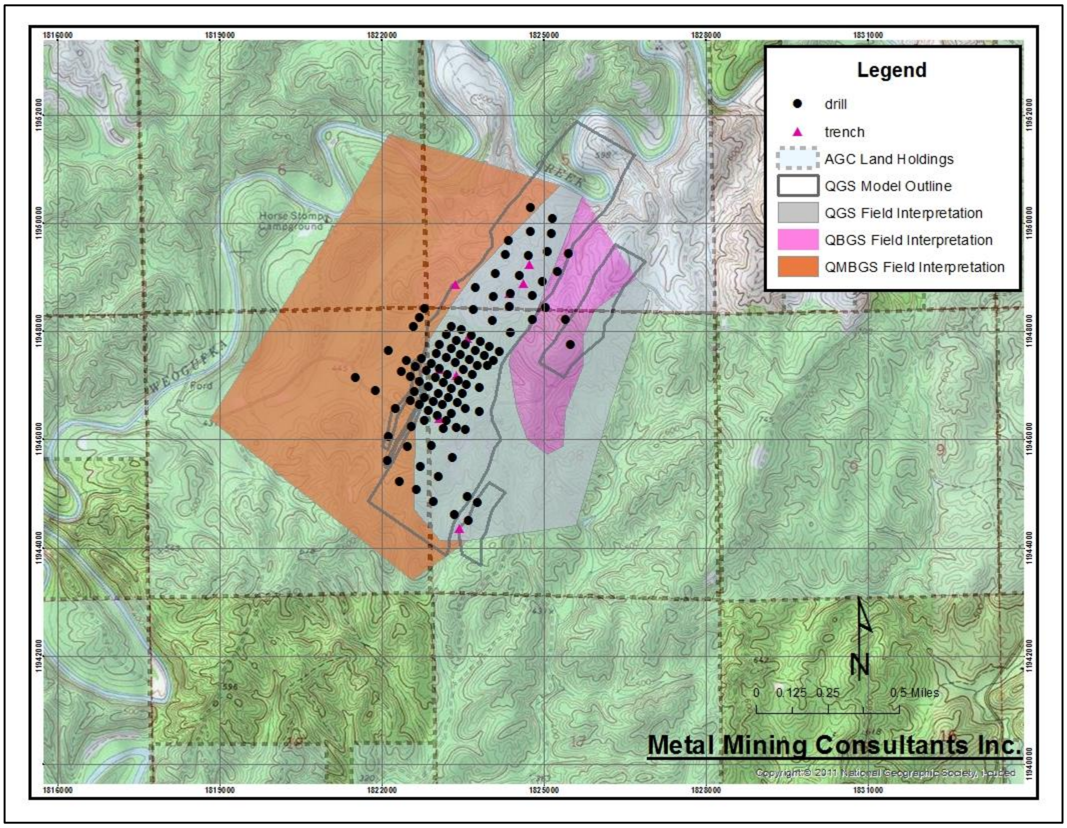
The Alabama Graphite Belt is 70 mi (112 km) long and covers the Higgins Ferry Group and the equivalent Poe Bridge Mountain Group, with an 11 mi (18 km) gap between the two where they are faulted out. The belt varies in width from 2 to 10 mi (3.2 to 16 km), with an average of 4 to 5 mi (6.4 to 8 km). Up to 10 distinct bands of graphitic schist were mapped within the Higgins Ferry Group by Neathery (1975). The individual mapped graphitic schist bands vary from a few hundred feet (tens of meters) up to 1 mi (1.6 km) in width, with continuity for up to 15 mi (24 km) or more along strike. Short narrow bands are shown as less than 1 mi (1.6 km) long.

The most common composition of the graphitic schist-bearing belts is biotitic-feldspathic-muscovite schists and gneisses. Pegmatite dikes are common, ranging from a few up to several feet (a few meters) in width to numerous dikelets less than 1 in (2.5 cm) wide. The graphite deposits are within wide bands of graphite-roscoelite-quartz-muscovite schists and graphitic micaceous quartzites. Graphite deposits of potentially economic grades occur as discontinuous elongated lenses within these zones. Coarser graphite flakes are often spatially associated with the margins of the pegmatites. Garnet-bearing biotitic schists are commonly present as well.

7.3 Coosa Geology & Mineralization

Geological mapping of the Coosa resource grid was carried out by Alabama Graphite geologists based on stream sections, road cuts, drill pads, and trenches. The results are shown in Figure 7-4. This shows that the main graphite schist trends northeast and to the south changes to north-south, and joins with another unit of graphite schist to the east. There is garnet schist on the east side of the graphite schist, which dies out to the south. These features may be interpreted as an anticline plunging at a low angle to the south with a core of garnet schist overlain structurally by graphite schist. The graphite schist is underlain structurally on the west side by quartz-muscovite-biotite-graphite schist.

Figure 7-4: Geological Map of the Coosa Project by AGC, with Drill Hole Locations



The main schist compositions and logging units defined by Alabama Graphite in drill core are:

Quartz-Muscovite-Biotite-Graphite Schist (QMBGS):

This schist is medium to coarse grained in texture and is characterized by large porphyroblasts of muscovite. The color varies from medium gray to dark gray-green. It is moderately foliated and commonly contorted, with lenticular pegmatites parallel to foliation. Pyrite and pyrrhotite are common accessory minerals as large disseminated grains. Sillimanite fibers have been observed. Graphite flakes are generally coarse in this unit although the average carbon content is generally less than 1% Cg.

Quartz-Graphite Schist (QGS):

This schist is both finer grained and better laminated than the QMBGS unit. The color varies from dark gray to black. Contorted foliation and pegmatites are less common. Pyrite and pyrrhotite are both finer grained and more abundant than in the QMBGS unit and form laminae parallel to foliation. Graphite is more abundant in this unit with grades higher than 1% Cg.

Mixed QMBGS/QGS (INT):

The QMBGS and QGS are commonly interbedded at a centimeter scale forming a mixed unit with graphite grades higher than 1% Cg.

Quartz-Biotite-Garnet Schist (QBGS):

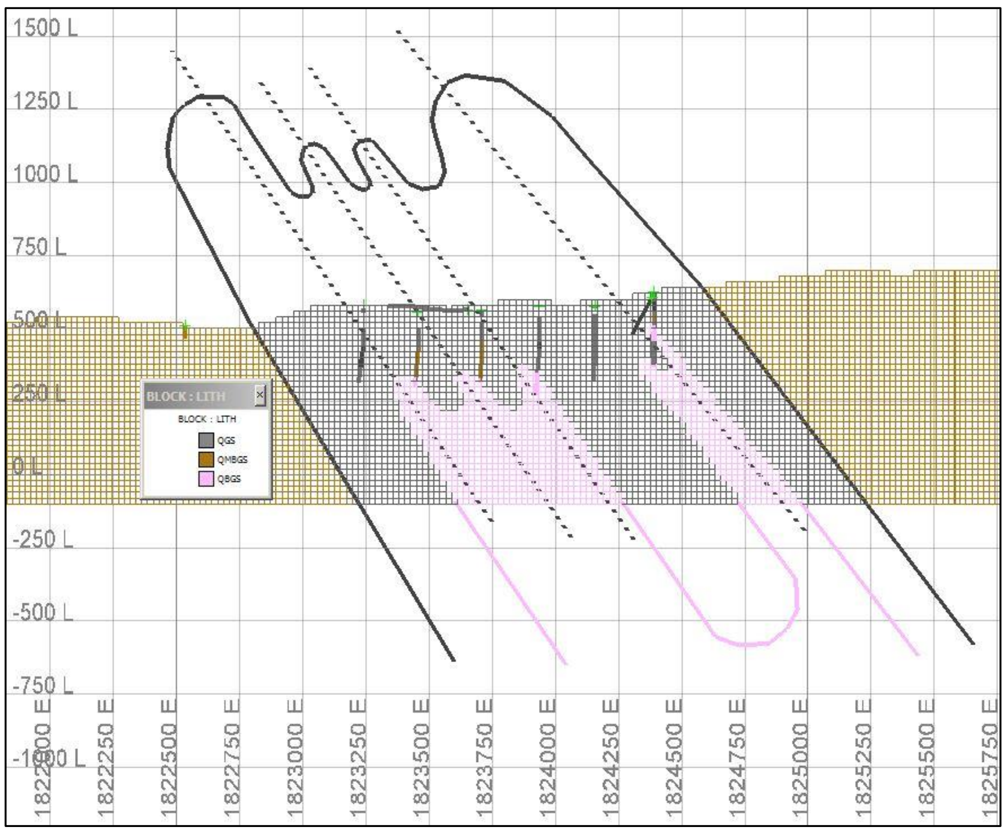
This unit is subordinate to both the QMBGS and QGS units. It is medium grained and medium to dark gray-green in color. The garnets are usually fine grained with a diameter of approximately 1 mm, but in places are up to 5 to 10 mm. The garnets are light pink in color, suggesting a high manganese (spessartine) content. Foliation is irregular with abundant pegmatites. Pyrite is sparse. Graphite is sparse and grades are less than 1% Cg.

In core the contacts between the major schist units are gradational rather than clear-cut. The dominant fabric is a strong schistosity (correlated regionally as S_1) which is affected by open, upright folds (F_3) seen in outcrop, and by less common, tight to isoclinal folds (F_1), seen in drill core.

An interpretative geological model was constructed on sections and in Vulcan from the drill hole data, geological mapping and geophysical data in order to constrain the block modeling process for resource estimation. The quartz-graphite schist (QGS) and intermediate quartz-graphite schist to quartz-muscovite-biotite-graphite schist (INT) were modeled as a single lithological unit with a strike length of 9,500 ft. (2,895 m) and an outcrop width of 1,700 ft. (518 m). These generally have grades >1% Cg. This includes some quartz-muscovite-biotite-graphite schist (QMBGS). They are overlain and underlain by QMBGS. This usually has grade of <1% Cg, although locally it can have higher grades. The contacts dip at approximately 60° east-to-southeast. The contacts are poorly constrained and can only be interpreted on 3 or 4 sections where they are forced between drill holes. Two lenses of intermediate quartz-graphite schist (INT) were modeled in the footwall of the main unit in the southwest part of the grid, and pinch out to the northeast. These are parallel to the main footwall contact and give added confidence to the attitude of the footwall contact and the geological model.

Short intersections of quartz-biotite-garnet schist (QBGS) were modeled as lenses or tight anticlines within or under the graphite schist domain. The QBGS has <1% Cg. A long section through these shows a plunge to the anticlinal axes of 20° southwest. There is a separate QBGS unit at the southeast end of the North Fence of drill holes which lies above graphite schist and is overlain structurally to the east by a second, structurally higher graphite schist unit which was intersected in hole AGC-007. One interpretation is that these are different strata and lenses in a layer-cake stratigraphy inclined to the southeast. An alternative interpretation is that these are tight to isoclinal folds of a sequence comprising garnet schist, overlain by graphite schist and in turn by quartz-muscovite-biotite-graphite schist. The axial planes of the folds and the strata dip at approximately 60° southeast, and the fold hinges plunge at 20° southwest. This interpretation is consistent with tight to isoclinal S_1 folds in core, geological mapping by AGC, and with the district scale geology. In this model, the tight to isoclinal folding has structurally thickened the graphite schist to give a much wider outcrop than the original stratigraphic thickness, which is not known but is interpreted to be much lower than the 1,200 ft. (366 m) apparent width on sections. Unfortunately, there are no marker beds that would enable this interpretation to be confirmed or changed. However, the geometry of the graphite schist body is the same in both interpretations, and it does affect the volume of graphite schist in the resource estimation to the depths drilled.

Figure 7-5: Cross-Sectional Cartoon of One Possible Interpretation of the Geology of the Coosa Deposit, with Tight to Isoclinal Folding Causing Structural Thickening of the Graphite Schist



The strike of the lithological boundaries is northeast, with an inflection to north-south in the southern part of the grid. This is interpreted to be a later F_3 fold rather than a fault offset.

The schistosity measured on surface and in drill core generally has a dip of 20 to 60° with an average of 45° (drill pad measurements) to the southeast and east and is lower than the modelled lithological contacts. In addition, the grade distribution in the block model has a 45° dip similar to the schistosity. No schistosity/bedding relationships were seen in core due to the lack of thinly bedded lithology's. Regionally the S_1 schistosity is reported to be generally parallel to bedding (S_0) (Tull, 1978) so the dip of the contacts may be lower than interpreted. A large number of structural measurements were taken in the trenching program and it is recommended that they be analyzed by stereonet and incorporated into the district scale geological mapping.

An attempt was made to wireframe model high grade graphite zones >2.5% Cg within the graphite schist, which defined six narrow bodies which may be multiple beds or a single folded bed. However, the zones do not correlate to any logged or recognizable geological feature and so have no geological support, vary widely in width, and are often constrained only by a single hole. In trial block models it was found that they strongly constrained the number of samples in the search ellipses and forced the creation of high grade blocks within the wireframes, so they were not used in the final geological model. A program of trenching is recommended to test the continuity of these high grade zones on surface.

Pegmatites (PEG) occur as narrow, centimeter-scale white bodies which are commonly parallel to the schistosity and in some cases show tight folding boudinage. Some pegmatites cross cut lithology and foliation. The pegmatites include metamorphic quartz veins and true, coarse-grained quartz-feldspar-biotite pegmatites. In core, there is often a concentration of coarse grained graphite in schist at the margins of the pegmatites. The pegmatites are rarely wide enough to be logged as a lithological unit [the only pegmatite actually logged is 8.1 ft. (2.5 m) wide]. Alabama Graphite logged the relative abundance of pegmatite in each sample in order to determine whether there is a relationship between pegmatite abundance and grade. The results are tabulated in Table 7-2 and show that the average graphite grade decrease with as increase in pegmatite. The pegmatites thus dilute the graphite grade.

Table 7-2: Relationship of Graphite Grade and Pegmatite Abundance

Pegmatite Abundance	Pegmatite %	Average Cg %	Number samples
0	0-1	2.48	1075
1	1-25	1.91	2775
2	25-75	1.61	595
3	75-100	1.46	121

Graphite flakes occur as part of the rock forming minerals in the schists. They are often associated with disseminated pyrrhotite and minor pyrite. In places the green vanadium-bearing muscovite, roscoelite, occurs. Minor late stage, straight-sided veinlets of cubic pyrite up to 10 mm wide with smectite clay cross cut the schistosity and pegmatites.

Graphite ores mined historically were almost entirely from the weathered zone, partly because weathering is deep here, and partly because the weathered rock could be gently crushed, liberating the graphite without significantly reducing the size of the larger flakes.

The oxide and transition zones were logged in core and modelled for resource estimation. The oxide zone is defined as the zone of total oxidation of sulfides to give an orange-red-brown color to core, and a much softer rock. The base of the oxide zone is often sharp and occurs over an interval of <1 ft. (0.3 m). The transition zone is defined as the zone of partial oxidation of sulfides and the rock is significantly crumblier than the underlying reduced or sulfide zone. The top of the transition zone is marked by the first appearance of sulfides downhole, and the base is marked by the disappearance of iron oxides, and a marked increase in hardness. Generally, there is more oxidation on fractures and veins in this zone.

Multi-element analysis' were made of all samples from one entire drill hole (AGC-010, n = 285 samples excluding QAQC samples). The graphitic schists are enriched in a number of metals and other elements including V (4 to 1695 ppm), Ag (0.04 to 14.90 ppm), Zn (4.2 to 3910 ppm), Ni (1.65 to 411 ppm), Mo (8.76 to 106.50 ppm) and Se (3.2 to 56.7 ppm).

Figure 7-6: Graphite Schist (QGS) with Sillimanite Porphyroblasts from the Oxide Zone. From 5 ft. Sample @ 2.27% Cg (AGC-15-F15, 48.6 ft.)

Figure 7-7: Graphite Schist with Green, Vanadium-Rich Muscovite, in the Oxide Zone. From a Sample of 5 ft. @ 2.87% Cg (AGC-15-F15, 24.0 ft.)

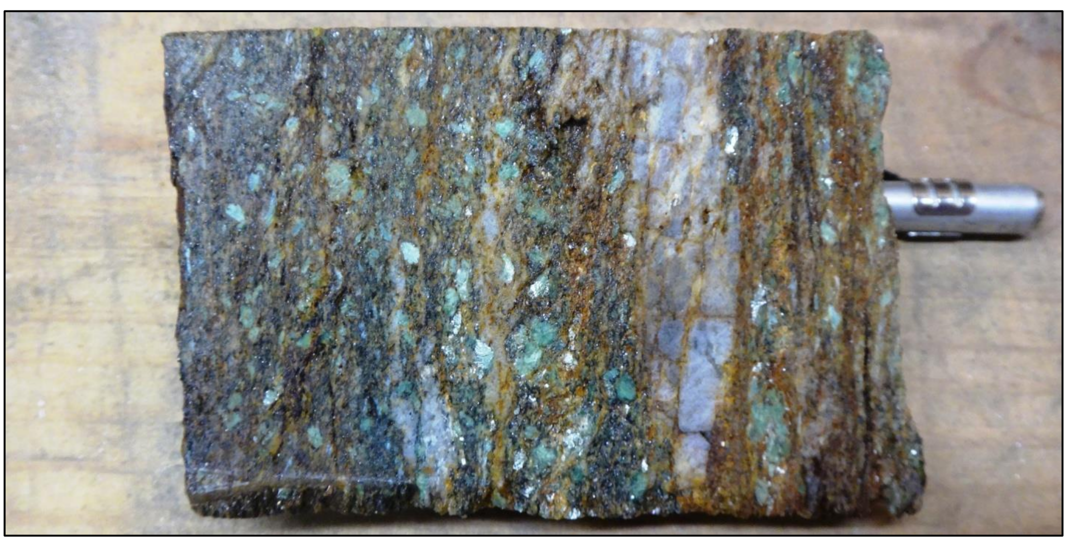


Figure 7-8: Biotite-Graphite Schist with Sillimanite(?) and Disseminated Pyrrhotite and Pyrite (Logged as QGS/QMBGS). From 5 ft. Sample @ 3.26% Cg (AGC-F05, 140.5 ft.)

Figure 7-9: Graphite Schist with Folded Pegmatites and Pyrrhotite (Logged as QMBGS). From 5 ft. Sample of 1.73% Cg (AGC-D0, 200 ft.)

Figure 7-10: High Grade Coarse Graphite in Sheared Pegmatite in Biotite-Graphite Schist (Logged as QGS/QMBGS). From 5 ft. Sample @ 4.56% Cg (ACG-F05, 134 ft.)

8 DEPOSIT TYPES

Graphite deposits occur in three forms: flake graphite, vein graphite, and amorphous graphite. These are described by Mitchell (1993):

> *"Graphite generally occurs as a result of metamorphism (regional or contact) of organic matter in sediments. Flake graphite is assumed to be derived from fine-grained sediments rich in organic matter. As metamorphic grade increases, carbonaceous material converts to "amorphous" graphite. Flake graphite forms from its amorphous precursor at or beyond amphibolite grade metamorphism (Landis, 1971). Vein graphite is assumed to form by partial volatilization of graphite and subsequent recrystallization during regional granulite and/or charnockite facies metamorphism. Amorphous graphite is generally considered to have originated by thermal or regional metamorphism of coal or carbonaceous sediments.*
>
> *"Positive vanadium and nickel anomalies and negative boron anomalies are possible signatures for graphite if geochemical survey data are available. The presence of sulphides and trace amounts of uranium may be an indicator."*

Landis (1971) tentatively concluded that graphite formation is primarily dependent on metamorphic temperature and forms above 400°C, with pressure and variation in starting material constituting secondary controls.

The Coosa graphite deposits are flake graphite deposits in high grade metamorphic rocks. They are associated with anomalous vanadium, including the vanadium-mica roscoellite, and nickel, as well as other elements.

9 EXPLORATION

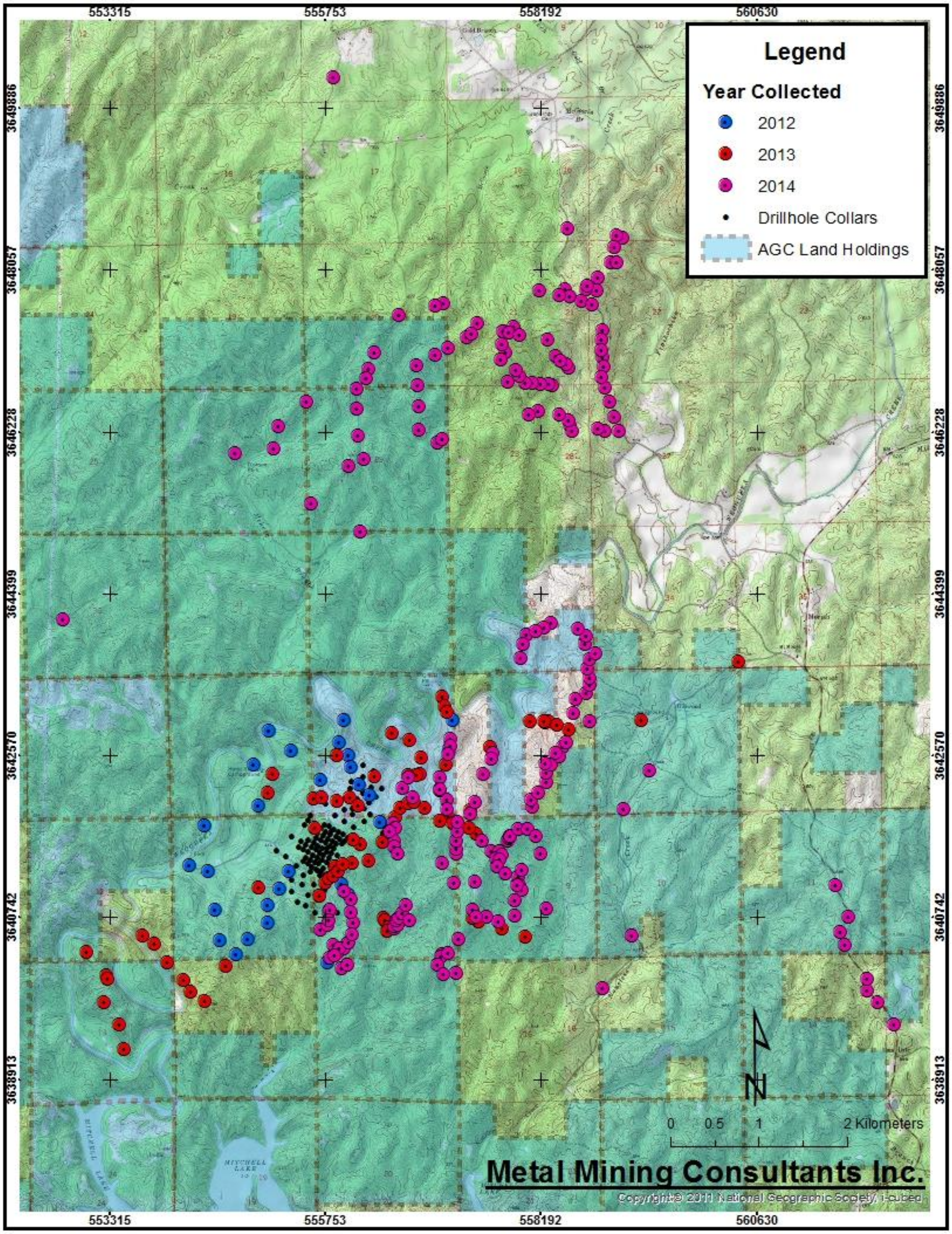
Alabama Graphite has conducted several surface exploration campaigns. Due to the lack of outcrop and dense forest, the exploration techniques used have been rock sampling in channels mainly along road cuttings, trenching, and geophysics.

9.1 Channel Sampling

Channel sampling was carried out initially close to the Coosa target and later over a large area around it. A total of 1,025 channel samples have been taken at 328 locations in 2012 to 2014, as shown in Figure 9-1. This comprised 115 samples in 2012 (a further 113 samples had unreliable analyses and the results were discarded, as described in Section 11.1), 268 samples in 2013, and 642 samples in 2014. One to sixteen samples would be taken at each sample locality, with an average of 3.125 samples per locality. The distribution of sample locations is not systematic due to the lack of rock outcrop and dense vegetation. Samples were typically collected along road cuts where bed rock is exposed. Small trenches were excavated by hand with a pick axe and shovel, and range from 2-3 in (5-8 cm) in depth and 5-35 ft. (1.5-10.7 m) in length. The trenches were typically cut perpendicular to the bed rock foliation to provide a representative sample of the outcrop and expose structural features such as foliation.

Before the sample is collected, a GPS is used to record the location of the start of the trench and all measurements are recorded in a field notebook which is later tabulated into an Excel database. Material collected from the trench is placed in a sample bag and labeled.

Figure 9-1: Map Showing the Location of Channel Samples at the Coosa Project (Metric UTM Grid)



9.2 Trenching

A program of mechanical trenching was carried out in the winter of 2014 using an excavator. A total of 30 trenches were dug with a total length of 10,790 ft. (3,288.8 m). Of these, nine trenches with a total length of 3,600 ft. (1,097 m) were dug in the Coosa resource grid and used in the database for the resource estimate, and another 21 trenches with a total length of 7,190 ft. (2,191.5 m) were dug on exploration targets. These are listed in Table 9-1**Error! Reference source not found.** and are shown on maps in Figure 9-3 and Figure 9-4.

The objective was to cut long trenches across the strike of the graphite mineralization in the grid area to infill between diamond drill holes and demonstrate continuity of mineralization. Trenches were also dug on geophysical exploration targets away from the drill grid. The average depth of the trenches was 4 to 5 ft. (1.2 to 1.5 m) and the maximum depth was 8 ft. (2.4 m). The trenches were backfilled and revegetated as soon as they had been sampled. Photographs of the trenching, sample collection and restoration are shown in Figure 9-2

The trenches were treated as low angle drill holes for surveying, sampling, logging, and the database. The starting point (collar) of the trench was surveyed by GPS, and measurements of azimuth and inclination were taken by compass and tape at 25 ft. (7.6 m) intervals. These were treated as down-hole directional surveys to plot the trench. Samples were surveyed by compass and tape to give length, azimuth and inclination. Samples were taken continuously in 5 ft. (1.5 m) lengths from the base of the wall of the trench using a geological hammer and a plastic core box as a receptacle. A 20-25 lb. (9-11 kg) bulk composite sample was also collected on 25 ft. (7.6 m) lengths for metallurgical test work.

Figure 9-2: Sample Collection in a Trench, and a Trench After Backfilling and Restoration

Photos R. Keevil, Alabama Graphite

Table 9-1: Trench Collar Table

Trench ID	Year	UTM Northing (m)	UTM Easting (m)	Altitude (m)	Azimuth (degrees)	Inclination (degrees)	Total Length (feet)
AGC-14-TR01A	2014	3642132	556181	225.0	210	4	550
AGC-14-TR01B	2014	3641968	556068	210.2	200	-7	425
AGC-14-TR02	2014	3642023	556146	226.6	130	3	475
AGC-14-TR03A	2014	3642022	555762	167.0	120	-13	1025
AGC-14-TR04	2014	3641756	556758	184.0	270	8	55
AGC-14-TR05	2014	3641717	555829	172.1	300	-3	275
AGC-14-TR07A	2014	3641509	555769	158.0	300	5	200
AGC-14-TR07B	2014	3641541	555674	155.1	295	-10	300
AGC-14-TR08	2014	3641261	555675	182.9	120	2	250
AGC-14-TR10A	2014	3640518	556102	160.0	305	13	100
AGC-14-TR10B	2014	3640552	555907	143.2	115	4	100
AGC-14-TR10C	2014	3640645	555787	158.5	135	-10	300
AGC-14-TR11	2014	3640831	556241	192.0	110	-2	300
AGC-14-TR12	2014	3640821	556530	165.6	300	2	950
AGC-14-TR13	2014	3642158	556805	203.3	300	10	75
AGC-14-TR14B	2014	3641921	556902	195.1	305	-10	600
AGC-14-TR14C	2014	3641829	557010	196.7	120	10	410
AGC-14-TR14D	2014	3641707	557093	204.2	110	0	350
AGC-14-TR16A	2014	3641519	557112	194.7	320	4	225
AGC-14-TR16B	2014	3641546	557031	200.5	320	-7	225
AGC-14-TR17	2014	3640390	557193	206.6	120	17	485
AGC-14-TR18A	2014	3641415	557700	228.6	135	-20	275
AGC-14-TR18B	2014	3641333	557803	198.1	110	-2	300
AGC-14-TR18C	2014	3641282	557957	181.4	130	18	350
AGC-14-TR19	2014	3641609	557887	221.5	120	-3	225
AGC-14-TR20	2014	3641760	558004	213.9	150	-10	1000
AGC-14-TR21A	2014	3642530	558515	207.2	280	4	325
AGC-14-TR21B	2014	3642556	558403	215.3	310	-2	125
AGC-14-TR22A	2014	3640743	556803	180.4	300	-8	175
AGC-14-TR22B	2014	3640707	556743	182.2	280	7	340

Note:

The coordinates, azimuth and inclination are for the trench starting point (collar).
The azimuth and inclination are more variable than for a drill hole.
Coosa Grid: TR01A, 01B, 02, 03, 05, 07A, 07B, 08, 10A, 10B, 10C (total 11).
Exploration: TR04, TR11 to TR22B (total 19).

Figure 9-3: Map Showing the Trench Locations in the Coosa Resource Grid (Metric UTM Grid)

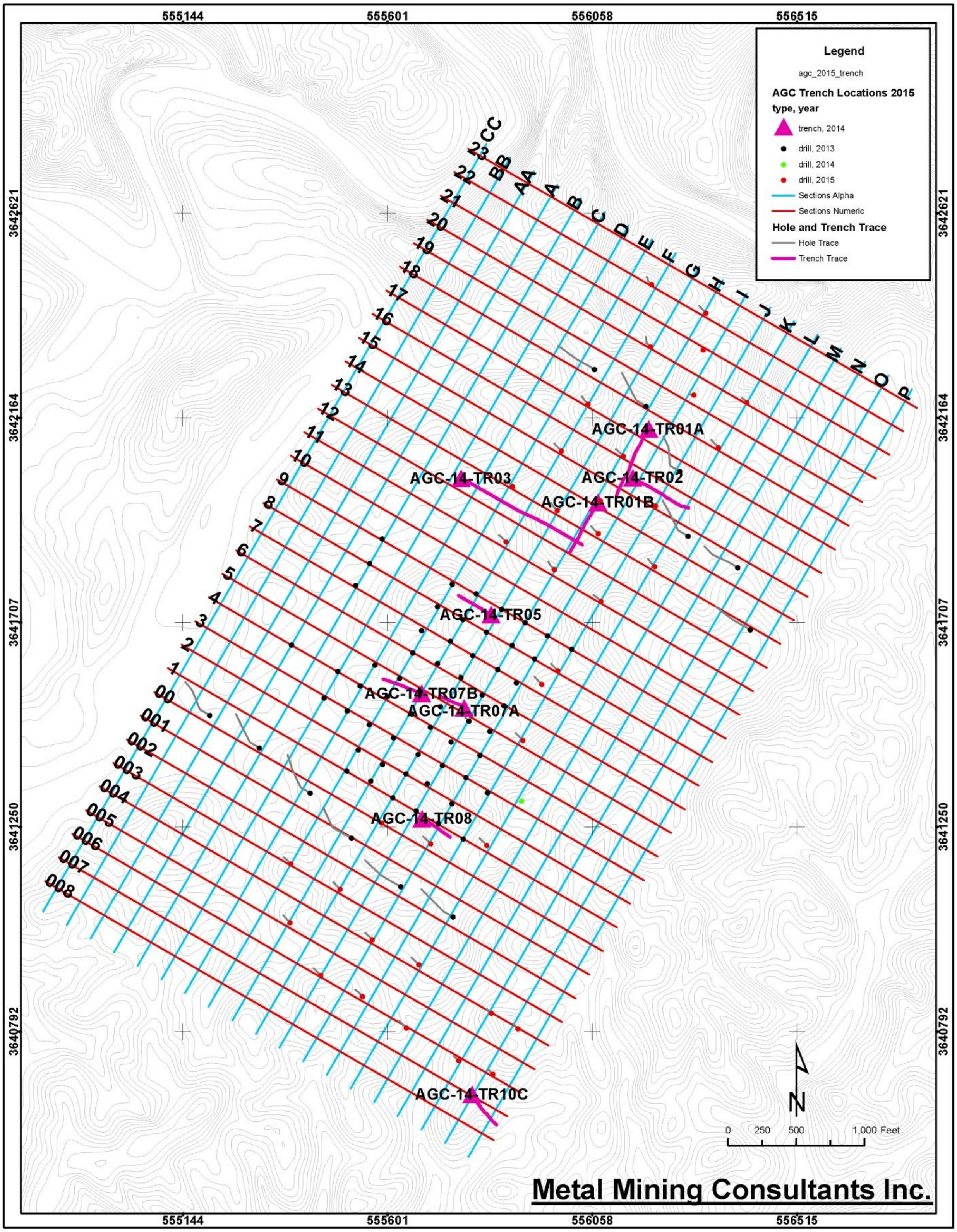
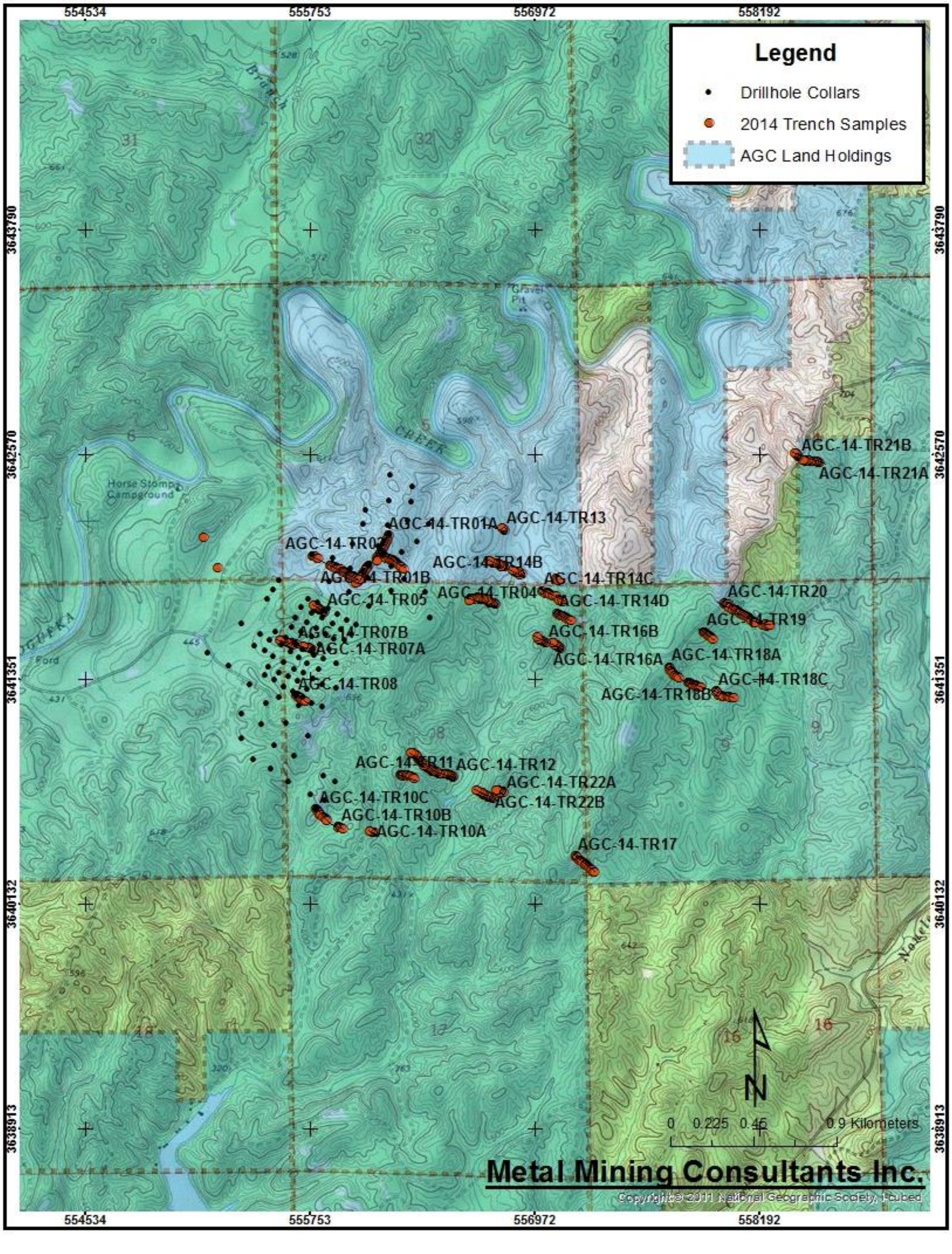


Figure 9-4: Plan of All Trenches



9.3 Geophysics

Alabama Graphite carried out two trial ground geophysical surveys. A GEM2 Ground frequency domain electromagnetic EM survey was carried out using a GEM2 instrument along roads. This had a depth of penetration of 50 ft. (15.2 m). Secondly, a ground time-domain electromagnetic (TDEM) survey was tested but was not found to be useful due to the high contrast in the EM response between the oxide and reduced (unweathered) zones.

A helicopter-borne magnetic, radiometric and time-domain electromagnetic (TDEM) survey was carried out in March 2014 for exploration surrounding the Coosa resource grid. The contractor was Prospectair (Quebec) and the data processing and interpretation were made by Dubé & Desaulniers Geoscience (Quebec) and is described in a report by Dubé (2014). The survey covered two areas named Coosa North, centered on the Coosa grid, and Coosa South over the Bama project in Chilton County, which is not described further in this report. A total of 891 line kms. was flown on Coosa North on lines oriented 126° with a 100 m line spacing, and perpendicular control lines spaced at 1,000 m, as shown in Figure 9-5. The average height above the ground of the helicopter was 89 m. Graphite targets were defined based on high conductivity from the TDEM survey (graphite and/or sulfides) combined with magnetic lows (no pyrrhotite), as shown in Figure 9-6. A number of these targets were followed up by channel sampling, trenching and, in some cases, drilling.

Figure 9-5: Area of the Coosa North Helicopter Geophysical Survey Showing Flight Lines and Digital Elevation Model Generated by Helicopter Altimetry

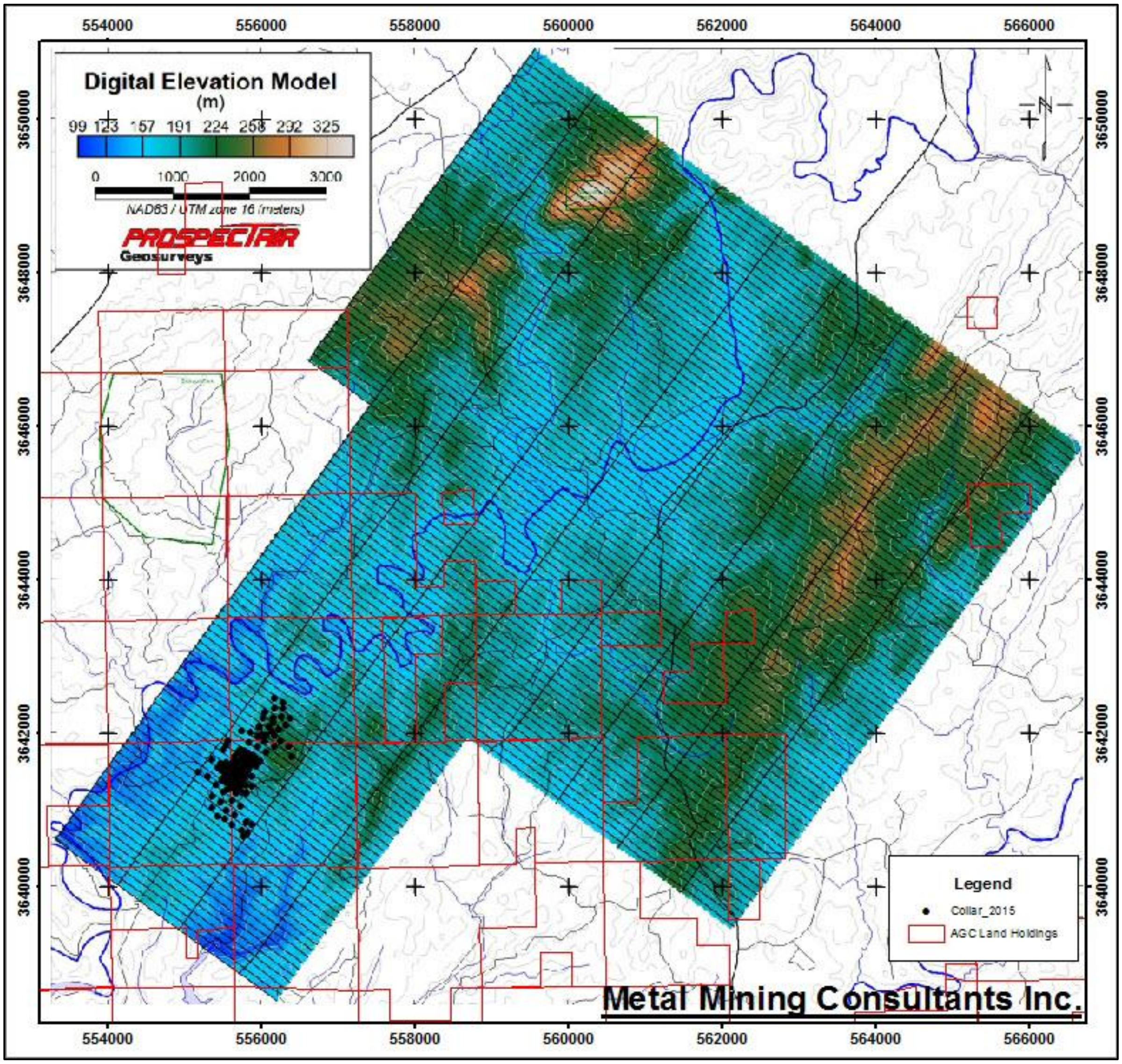
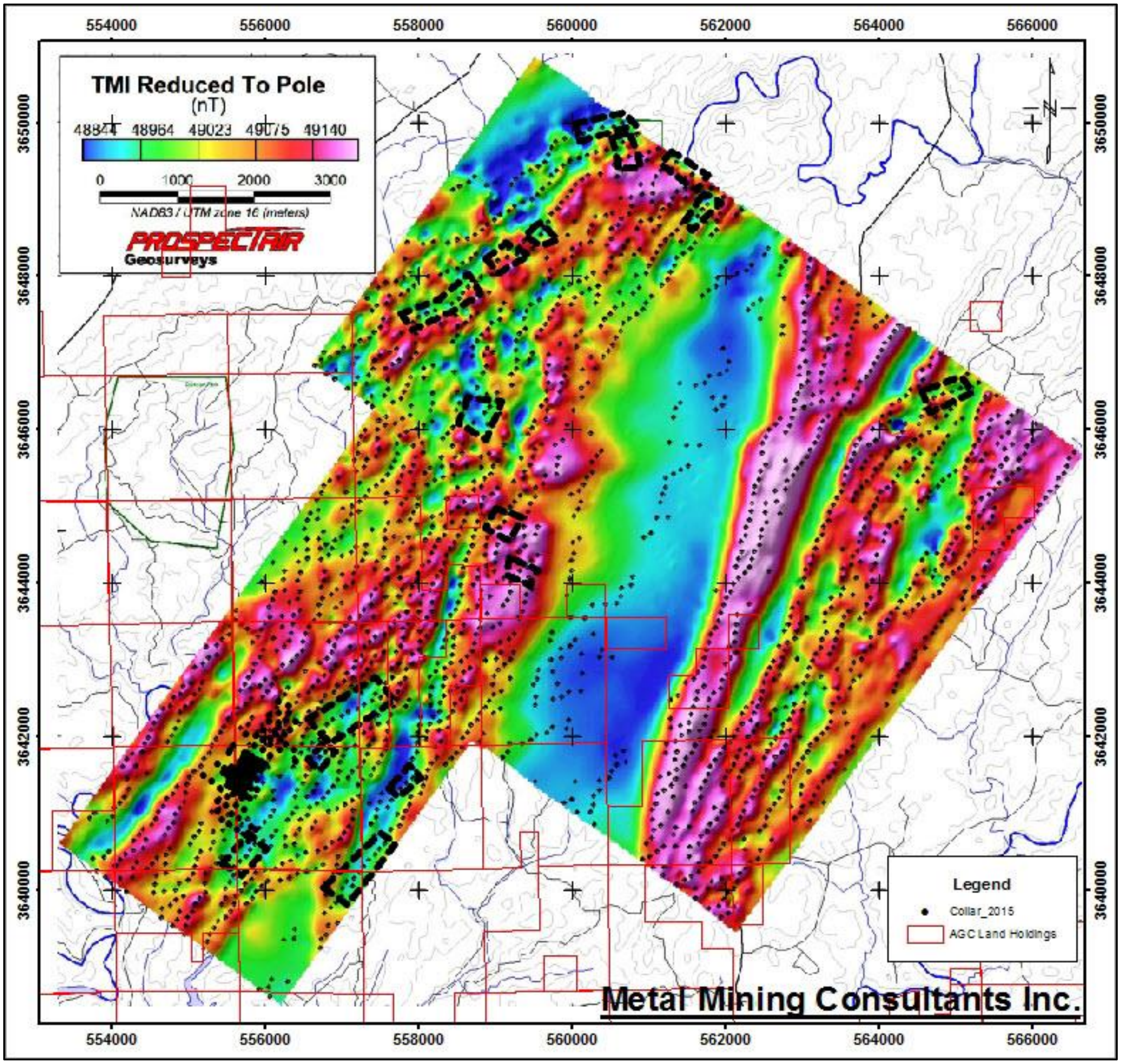


Figure 9-6: Aeromagnetic Map of Coosa North (Total Magnetic Intensity Reduced to the Pole) with TDEM Anomalies (Dotten Lines) and Targets (Boxes)



10 DRILLING

Alabama Graphite has conducted four drilling programs at the Coosa project between 2012 and 2015 comprising 135 drill holes totaling 28,604.9 ft. (8,718.8 m). Of these, 109 holes totaling 25,904.9 ft. (7,895.8 m) were drilled in the Coosa target and were used in the database for the resource estimation. The other 26 holes totaling 2,700.0 ft. (822.9 m) are exploration holes drilled out with the Coosa target and were not used in the database for resource estimation. The drill collars are listed in Table 10-1 and the Coosa target holes are shown in a map in Figure 10-4. The significant graphite intercepts are listed in Table 10-2.

10.1 2012 Diamond Drilling (Stratigraphic Lines)

The first program of diamond drilling was carried out between 27 September 2012 and 23 October 2012 and consisted of two northwesterly trending fences of six holes each for a total of 6,003 ft. (1,829.7 m, holes AGC-001 through AGC-012). All fence holes were drilled at an inclination of -50° perpendicular to the stratified graphite horizons to gain information regarding the stratigraphy and graphite distribution. The two fences were approximately 3,000 to 3,800 ft. (930 to 1,160 m) apart with hole spacing averaging 500 ft. (152.4 m). The fence holes were core holes of HQ diameter (2.5 in / 6.4 cm diameter) and were drilled to a nominal depth of 500 ft. (152.4 m).

10.2 2012 Diamond & Sonic Drilling (Resource Grid)

The second drilling program was carried out between 23 October 2012 and 21 December 2012 and consisted of 57 vertical holes drilled on a 200 ft. x 200 ft. (61 m x 61 m) grid for a total of 14,342.4 ft. (4,371.6 m). The holes were numbered by the grid location and are holes AGC-A04 to AGC-J09. The grid drilling was a combination of sonic core and HQ core. Diamond drilling gave low core recovery (<50%) in the upper weathered zone so a sonic drill was used to drill core through the weathered zone and set casing for the HQ diamond core drill. This method proved to be very effective due to the near 100% core recovery from the sonic drill. However, in some holes there is a gap of up to 10 ft. (3.05 m) in sampling between the end of the sonic core and the start of the diamond core. A photo of the sonic drill rig is shown in Figure 10-1. of the 57 vertical grid holes, 44 of them were drilled in the upper part with the sonic drill for a total of 1,523.5 ft. (464.4 m).

10.3 2014 Sonic Drilling for Exploration

Twenty-four sonic holes were drilled in summer 2014 for a total of 1,303 ft. (397.15 m) numbered AGC-10S, AGC-12S, AGC-14-K03S and AGC-14-01S to AGC-14-21S. The core diameter was HQ.

Six of these holes were located in the Coosa area and are included in the database for the resource estimate. Of these six, holes AGC-14-010S and AGC-14-012S were re-drilled in the upper parts of diamond drill holes AGC-010 and AGC-012, but are treated as separate drill holes as they were not pre-collars; three holes (AGC-14-13S, AGC-14-18S, AGC-14-20S) were deepened by diamond drilling in 2015 (respectively holes AGC-15-H14, AGC-15-I19 and ACG-15-I21; thus the sonic pre-collars are not listed separately in Table 10-1); and one hole was not deepened and is only a sonic hole (AGC-14-K03S).

The other 18 sonic holes drilled in 2014 are exploration holes to test geophysical anomalies away from the Coosa target and were not used in the database for the resource estimate.

Figure 10-1: Track-Mounted Sonic Drill Rig

Photo: R. Keevil, Alabama Graphite

10.4 2015 Diamond Drilling Program

The 2015 drilling program was designed to extend the drill grid and resource to the north and south, with the emphasis on the oxide and transition zones. A different drill contractor was used, Dycus Diamond Drilling LLC (3D Drilling) of Wytheville, Virginia, with two truck-mounted Longyear 38 drill rigs, one of which is shown in a photo in Figure 10-2. The contractor was chosen for their ability to get high core recoveries of almost 100% in the oxide and transition zones, which was achieved by drilling at a slower rate with less weight on the bit. The core diameter was HQ. The three deeper exploration holes were reduced to NQ diameter at 150 ft. (45.7 m) depth.

A total of 37 diamond drill holes for 5,333.5 ft. (1,625.65 m) were drilled on the Coosa resource grid in 2015, of which 3 holes totaling 142.0 ft. (43.28 m) were pre-collared by sonic drilling in 2014. The holes were numbered by the year and grid location and are AGC-15-E003 to AGC-15-O006.

In addition, nine holes for 1,755 ft. (534.92 m) were drilled on two exploration targets in 2015, Fixico (holes AGC-15-FIX01 to AGC-15-FIX03) and Holy Schist (holes AGC-15-HS01 to AGC-15-HS06). One of these was pre-collared by sonic drilling in 2014. These holes were not used in the database for the Coosa resource estimate.

Figure 10-2: Truck Mounted Longyear 38 Drill of 3D Drilling on Hole AGC-15-I13, June 2015

Photo S. Redwood

Downhole surveys of the diamond drill holes were taken by the drilling company with a Reflex Instruments EZ Shot Survey tool (all 2012 holes, 22 of the 2015 holes) or a Multishot tool (24 of the 2015 holes). The instruments were adjusted for the local magnetic declination of 3.5° west and recorded the azimuths relative to grid north. Measurements were made at the end of the hole and, in the deeper holes, at a midway depth. Downhole surveys were not carried out on the sonic drill holes as they were short and vertical, and most were pre-collars for diamond drill holes.

Drill collars are capped by plastic pipe with a cap and cemented in place, as shown in Figure 10-3. Collar locations were surveyed with a high precision 2005 Trimble GeoXM GPS unit with sub-meter accuracy. The collars of the 2014 sonic holes were surveyed with a hand-held Garmin GPS unit with lower accuracy (3-5 m). The datum used was NAD83 UTM Zone 16N in meters.

Figure 10-3: Drill Collar for Hole AGC-14-I05

Photo S. Redwood

Table 10-1: Coosa Project Diamond and Sonic Drill Hole Collar Data

Hole ID	Year	UTM Northing (m)	UTM Easting (m)	Altitude (m)	Azimuth (degrees)	Inclination (degrees)	Sonic Collar Depth (feet)	Total Depth (feet)
AGC-001	2012	3641504	555202	137	325	-55	0	500
AGC-002	2012	3641431	555314	141	325	-55	0	503
AGC-003	2012	3641330	555427	152	325	-55	0	500
AGC-004	2012	3641229	555519	168	325	-55	0	500
AGC-005	2012	3641120	555629	183	325	-55	0	500
AGC-006	2012	3641052	555746	180	325	-55	0	500
AGC-007	2012	3641694	556410	234	325	-55	0	500
AGC-008	2012	3641834	556382	218	325	-55	0	500
AGC-009	2012	3641903	556271	210	325	-55	0	500
AGC-010	2012	3642048	556252	222	325	-55	0	500
AGC-011	2012	3642194	556177	210	325	-55	0	500
AGC-012	2012	3642276	556062	188	325	-55	0	500
AGC-A04	2012	3641661	555386	142	0	-90	25	279.5
AGC-B07	2012	3641794	555528	145	0	-90	39	39
AGC-B08	2012	3641842	555560	148	0	-90	32.5	250
AGC-B09	2012	3641898	555588	155	0	-90	39	46
AGC-C03	2012	3641542	555458	144	0	-90	0	256
AGC-C04	2012	3641601	555489	147	0	-90	25	250
AGC-D03	2012	3641513	555510	146	0	-90	0	256
AGC-D04	2012	3641569	555539	152	0	-90	30	250
AGC-D05	2012	3641615	555572	151	0	-90	39	254
AGC-E01	2012	3641379	555509	149	0	-90	34	256

Hole ID	Year	UTM Northing (m)	UTM Easting (m)	Altitude (m)	Azimuth (degrees)	Inclination (degrees)	Sonic Collar Depth (feet)	Total Depth (feet)
AGC-E02	2012	3641426	555534	151	0	-90	34	237
AGC-E03	2012	3641483	555559	145	0	-90	0	287
AGC-E04	2012	3641545	555600	153	0	-90	35	256.5
AGC-E05	2012	3641586	555626	157	0	-90	39	256
AGC-E06	2012	3641643	555656	164	0	-90	0	252
AGC-E07	2012	3641692	555675	159	0	-90	0	305
AGC-E08	2012	3641746	555712	169	0	-90	25	276
AGC-E09	2012	3641796	555744	176	0	-90	25	264
AGC-F01	2012	3641356	555563	158	0	-90	49	256
AGC-F02	2012	3641394	555589	164	0	-90	29	303
AGC-F03	2012	3641453	555613	155	0	-90	29	255
AGC-F04	2012	3641507	555654	149	0	-90	0	256.4
AGC-F05	2012	3641556	555673	156	0	-90	45	250
AGC-F06	2012	3641619	555711	165	0	-90	30	249
AGC-F07	2012	3641669	555740	166	0	-90	40	265
AGC-F08	2012	3641717	555768	173	0	-90	35	256
AGC-F09	2012	3641775	555798	169	0	-90	0	256
AGC-G01	2012	3641319	555611	166	0	-90	59	302
AGC-G02	2012	3641373	555641	170	0	-90	49	266
AGC-G03	2012	3641416	555668	160	0	-90	39	276
AGC-G04	2012	3641477	555695	154	0	-90	34	272.5
AGC-G05	2012	3641523	555718	154	0	-90	25	276
AGC-G06	2012	3641588	555764	165	0	-90	0	286
AGC-G07	2012	3641637	555790	165	0	-90	0	257
AGC-G08	2012	3641689	555821	171	0	-90	0	256
AGC-G09	2012	3641741	555855	171	0	-90	25	250
AGC-H01	2012	3641290	555664	181	0	-90	59	260
AGC-H02	2012	3641351	555689	168	0	-90	49	252
AGC-H03	2012	3641391	555722	165	0	-90	39	274
AGC-H04	2012	3641444	555741	164	0	-90	59	273
AGC-H05	2012	3641490	555782	157	0	-90	40	256
AGC-H06	2012	3641549	555811	173	0	-90	25	264.5
AGC-H07	2012	3641606	555846	180	0	-90	30	256
AGC-H08	2012	3641660	555878	180	0	-90	25	254
AGC-H09	2012	3641711	555907	175	0	-90	20	305
AGC-I01	2012	3641262	555713	180	0	-90	40	291
AGC-I02	2012	3641306	555743	173	0	-90	30	253
AGC-I03	2012	3641364	555775	182	0	-90	60	256
AGC-I04	2012	3641414	555806	170	0	-90	20	296
AGC-I05	2012	3641468	555828	162	0	-90	15	256
AGC-I06	2012	3641524	555861	165	0	-90	10	261
AGC-I07	2012	3641576	555887	170	0	-90	23	285
AGC-I08	2012	3641629	555928	177	0	-90	0	265
AGC-I09	2012	3641681	555957	175	0	-90	0	256
AGC-J01	2012	3641227	555769	182	0	-90	40	253
AGC-J03	2012	3641331	555823	191	0	-90	20	30

Hole ID	Year	UTM Northing (m)	UTM Easting (m)	Altitude (m)	Azimuth (degrees)	Inclination (degrees)	Sonic Collar Depth (feet)	Total Depth (feet)
AGC-J09	2012	3641652	556011	191	0	-90	0	257
AGC-10S	2014	3642048	556247	222	0	-90	55	55
AGC-12S	2014	3642274	556058	188	0	-90	55	55
AGC-14-K03S	2014	3641301	555897	177	0	-90	47	47
AGC-14-01S	2014	3640884	556047	171	0	-90	65	65
AGC-14-02S	2014	3641798	556561	216	0	-90	55	55
AGC-14-03S	2014	3642023	556772	203	0	-90	60	60
AGC-14-04S	2014	3642289	557063	188	0	-90	60	60
AGC-14-05S	2014	3641580	557247	180	0	-90	60	60
AGC-14-07S	2014	3641680	557476	218	0	-90	60	60
AGC-14-08S	2014	3641349	557726	223	0	-90	50	50
AGC-14-09S	2014	3641637	557864	230	0	-90	60	60
AGC-14-10S	2014	3642604	558467	227	0	-90	55	55
AGC-14-11S	2014	3641680	558141	188	0	-90	60	60
AGC-14-12S	2014	3640370	557212	215	0	-90	53	53
AGC-14-14S	2014	3641495	556567	208	0	-90	55	55
AGC-14-15S	2014	3642240	556572	210	0	-90	57	57
AGC-14-16S	2014	3640660	556077	177	0	-90	55	55
AGC-14-17S	2014	3641431	557211	201	0	-90	50	50
AGC-14-19S	2014	3641971	556616	216	0	-90	25	25
AGC-14-21S	2014	3641605	556536	210	0	-90	65	65
AGC-15-E003	2015	3641160	555383	162	315	-60	0	150
AGC-15-E13	2015	3642003	555878	165	315	-60	0	150
AGC-15-F005	2015	3641029	555381	156	315	-60	0	150
AGC-15-F11	2015	3641880	555863	167	315	-60	0	140
AGC-15-F15	2015	3642083	555986	186	315	-60	0	150
AGC-15-F17	2015	3642188	556046	190	315	-60	0	150
AGC-15-F22	2015	3642455	556188	175	315	-60	0	136.5
AGC-15-G00	2015	3641252	555585	157	315	-60	0	150
AGC-15-G003	2015	3641103	555493	176	315	-60	0	150
AGC-15-G13	2015	3641950	555977	194	315	-60	0	150
AGC-15-G20	2015	3642316	556186	172	315	-60	0	150
AGC-15-H006	2015	3640911	555449	162	315	-60	0	150
AGC-15-H11	2015	3641818	555971	178	315	-60	0	150
AGC-15-H14	2014-15	3641971	556074	211	0	-90	55	132
AGC-15-H16	2015	3642071	556125	221	315	-60	0	150
AGC-15-H22	2015	3642392	556309	184	315	-60	0	150
AGC-15-I00	2015	3641205	555694	175	315	-60	0	150
AGC-15-I004	2015	3640991	555564	162	315	-60	0	150
AGC-15-I13	2015	3641898	556069	199	315	-60	0	150
AGC-15-I19	2014-15	3642209	556283	207	0	-90	42	190
AGC-15-I21	2014-15	3642309	556304	198	0	-90	45	195
AGC-15-J006	2015	3640864	555542	168	315	-60	0	150
AGC-15-J05	2015	3641436	555901	170	315	-60	0	150
AGC-15-J07	2015	3641562	555942	185	315	-60	0	150
AGC-15-J08	2015	3641592	555978	185	315	-60	0	145

Hole ID	Year	UTM Northing (m)	UTM Easting (m)	Altitude (m)	Azimuth (degrees)	Inclination (degrees)	Sonic Collar Depth (feet)	Total Depth (feet)
AGC-15-J11	2015	3641747	556074	205	315	-60	0	150
AGC-15-J15	2015	3641960	556197	218	315	-60	0	150
AGC-15-K004	2015	3640935	555668	171	315	-60	0	150
AGC-15-K01	2015	3641202	555820	192	315	-60	0	150
AGC-15-K13	2015	3641825	556195	207	315	-60	0	150
AGC-15-K18	2015	3642091	556337	201	315	-60	0	145
AGC-15-K20	2015	3642192	556401	192	315	-60	0	100
AGC-15-L006	2015	3640793	555640	150	315	-60	0	150
AGC-15-N004	2015	3640827	555830	163	315	-60	0	100
AGC-15-N006	2015	3640720	555757	150	315	-60	0	100
AGC-15-O004	2015	3640792	555889	171	315	-60	0	100
AGC-15-O006	2015	3640690	555834	162	315	-60	0	100
AGC-15-FIX01	2015	3640969	556290	164	315	-60	0	300
AGC-15-FIX02	2015	3640871	556373	162	315	-60	0	300
AGC-15-FIX03	2015	3640838	556507	140	315	-60	0	300
AGC-15-HS01	2015	3641535	557205	167	315	-60	0	175
AGC-15-HS02	2015	3641668	557170	176	315	-60	0	140
AGC-15-HS03	2014-15	3641727	557258	179	0	-90	57	145
AGC-15-HS04	2015	3641585	557079	182	315	-60	0	195
AGC-15-HS05	2015	3641489	557107		315	-60	0	100
AGC-15-HS06	2015	3641462	557017		315	-60	0	100

Notes:

1. The coordinates are in NAD83 UTM zone 16N in meters. The altitudes were registered to the digital elevation model.
2. Sonic collar depth = 0 = no sonic collar.
3. AGC-010 and AGC-012 were re-drilled in the upper part by sonic holes AGC-10S and AGC-12S. They are treated as separate drill holes and not pre-collars as they were drilled later, and are vertical rather than inclined.
4. AGC-15-H14 (AGC-14-13S), AGC-15-I19 (AGC-14-18S), AGC-15-I21 (AGC-14-20S) and AGC-15-HS03 (AGC-14-06S) were drilled by sonic drilling in 2014 (original sonic hole number in brackets) and were deepened by diamond drilling in 2015.
5. AGC-14-01S to AGC-14-12S, AGC-14-14S to AGC-14-17S, AGC-14-19S, AGC-14-21S, AGC-15-FIX01 to AGC-15-FIX03, and AGC-15-HS01 to AGC-15-FIX06 (total 26 holes) are exploration holes with the Coosa grid and were not used in the database for the mineral resource estimate.

Table 10-2: Significant Intercepts from the 2012-15 Drilling

Hole ID	From (ft.)	To (ft.)	Width (ft.)	Cg%
AGC-001	90	100	10	1.18
and	175	185	10	1.00
and	250	400	150	2.50
incl	265	275	10	3.50
and	465	495	30	1.80
AGC-002	65	75	10	1.75
and	375	390	15	2.04

Hole ID	From (ft.)	To (ft.)	Width (ft.)	Cg%
AGC-003	0	160	160	2.00
incl	85	95	10	3.15
incl	130	145	15	3.22
AGC-004	65	150	85	1.32
and	170	180	10	2.27
and	305	315	10	1.25
and	425	455	30	1.52
and	475	500	25	2.01
AGC-005	70	255	185	2.47
incl	70	85	15	3.95
incl	90	100	10	3.40
incl	140	165	25	3.72
incl	175	185	10	3.18
and	310	500	190	1.99
incl	330	340	10	3.24
incl	380	400	20	3.79
incl	465	480	15	3.29
AGC-006	85	495	410	2.12
incl	85	130	45	3.71
AGC-007	145	280	135	2.98
incl	150	170	20	4.58
incl	190	235	45	3.62
incl	265	275	10	3.00
AGC-008	NSV			
AGC-009	53	65	12	1.29
and	90	500	410	2.89
incl	100	125	25	3.68
and	200	215	15	3.95
and	255	265	10	3.37
and	305	370	65	3.77
and	395	470	75	3.71
AGC-010	55	500	445	2.82
incl	70	155	85	3.59
incl	175	190	15	3.72
incl	230	240	10	3.43
incl	280	305	25	3.79
incl	330	340	10	3.83
incl	405	425	20	3.86
incl	440	475	35	3.69
AGC-011	100	500	400	2.75
incl	150	265	115	3.45
incl	275	300	25	3.38
incl	315	345	30	3.27
incl	415	425	10	3.28
incl	470	485	15	3.55
AGC-012	65	85	20	4.56
and	95	135	40	2.95

Hole ID	From (ft.)	To (ft.)	Width (ft.)	Cg%
incl	95	110	15	3.58
and	150	165	15	1.92
and	205	220	15	1.16
and	390	405	15	1.52
and	490	500	10	2.22
AGC_14-010S	15	55	40	3.26
incl	25	55	30	3.67
AGC_14-012S	0	55	55	2.84
incl	35	50	15	3.74
AGC-14-K03	0	47	47	2.66
AGC-14-TR01A	0	550	550	2.92
incl	15	45	30	3.62
incl	55	105	50	3.28
incl	120	130	10	3.66
incl	140	170	30	3.37
incl	180	200	20	3.22
incl	230	270	40	3.31
incl	320	340	20	4.00
incl	365	435	70	3.74
incl	510	550	40	3.06
AGC-14-TR01B	10	425	415	2.62
incl	65	95	30	3.21
incl	115	135	20	3.05
incl	175	185	10	3.79
incl	200	225	25	3.08
incl	240	250	10	3.16
incl	300	340	40	3.05
incl	390	425	35	3.90
AGC-14-TR02	0	230	230	2.58
incl	130	195	65	3.35
incl	220	230	10	3.39
and	235	475	240	2.06
incl	365	375	10	3.31
AGC-14-TR03	0	25	25	1.60
and	375	455	80	2.51
and	545	735	190	2.19
and	705	735	30	3.92
and	740	1025	285	2.81
incl	765	855	90	3.70
incl	945	955	10	3.48
incl	980	1020	40	3.31
AGC-14-TR05	0	275	275	3.65
incl	0	30	30	2.04
incl	215	240	25	3.65
AGC-14-TR07A	0	200	200	2.14
incl	85	145	60	3.22
AGC-14-TR07B	0	250	250	2.97

Hole ID	From (ft.)	To (ft.)	Width (ft.)	Cg%
incl	5	30	25	3.84
incl	55	95	0	3.44
incl	225	250	25	3.48
AGC-14-TR08	0	250	250	3.12
incl	0	15	15	4.62
incl	30	105	75	4.26
incl	120	195	75	3.31
AGC-14-TR10C	0	300	300	2.53
incl	0	20	20	3.60
incl	65	120	55	3.68
incl	175	185	10	4.58
incl	200	215	15	3.55
incl	225	235	10	4.02
AGC-15-E003	5	150	145	1.78
AGC-15-E13	0	150	150	2.53
incl	25	40	15	3.93
incl	50	60	10	3.55
incl	130	145	15	3.75
AGC-15-F005	45	150	105	1.48
AGC-15-F11	NSV			
AGC-15-F15	10	150	140	2.60
incl	25	55	30	3.78
incl	130	150	20	3.62
AGC-15-F17	0	150	150	3.29
incl	35	75	40	4.19
incl	90	120	30	4.43
AGC-15-F22	5	136.5	131.5	1.09
AGC-15-G00	25	150	125	2.31
incl	25	50	25	3.54
AGC-15-G003	55	150	95	2.08
incl	60	70	10	3.26
incl	85	95	10	3.15
AGC-15-G13	10	145	135	3.67
incl	10	75	65	3.87
incl	85	95	10	3.05
incl	110	135	25	6.06
AGC-15-G20	0	150	150	3.37
incl	0	55	55	3.80
incl	65	140	75	3.30
AGC-15-H006	35	105	70	1.60
and	130	150	20	1.23
AGC-15-H11	0	150	150	2.54
incl	0	25	25	3.90
incl	40	80	40	3.67
AGC-15-H14	0	53	53	3.13
incl	10	35	25	4.02
and	55	132	77	3.16

Hole ID	From (ft.)	To (ft.)	Width (ft.)	Cg%
incl	70	105	35	3.83
AGC-15-H16	5	150	145	3.11
incl	30	90	60	4.23
AGC-15-H22	10	50	40	2.58
and	55	150	95	3.30
incl	55	120	65	4.00
AGC-15-I00	0	120	120	2.57
and	135	150	15	4.06
AGC-15-I004	20	150	130	2.94
incl	35	80	45	3.29
incl	90	105	15	3.11
incl	140	150	10	4.14
AGC-15-I13	10	35	25	1.61
and	40	150	110	3.20
incl	60	105	45	3.50
and	125	140	15	3.60
AGC-15-I19	0	70	70	2.31
AGC-15-I21	0	195	195	3.77
incl	35	190	155	4.01
AGC-15-J006	0	10	10	2.38
and	15	150	135	1.89
AGC-15-J05	10	100	90	1.71
incl	10	20	10	3.92
incl	120	150	30	1.27
AGC-15-J07	0	35	35	1.14
and	45	150	105	2.68
and	65	75	10	4.00
and	115	130	15	3.61
AGC-15-J08	5	145	140	3.05
incl	55	60	5	3.73
incl	130	145	15	3.27
AGC-15-J11	0	150	150	3.60
incl	5	45	40	6.12
incl	80	125	45	3.46
incl	140	150	10	4.04
AGC-15-J15	0	20	20	2.47
and	25	150	125	4.07
AGC-15-K004	10	150	140	2.57
incl	15	25	10	3.57
incl	115	150	35	3.43
AGC-15-K01	0	150	150	3.64
incl	10	60	50	4.02
incl	70	125	55	3.84
incl	135	150	15	4.16
AGC-15-K13	0	30	30	2.68
and	35	150	115	3.00
incl	5	15	10	3.22

Hole ID	From (ft.)	To (ft.)	Width (ft.)	Cg%
incl	35	50	15	4.36
incl	60	70	10	4.50
incl	110	120	10	3.23
incl	135	145	10	3.24
AGC-15-K18	0	125	125	1.62
AGC-15-K20	NSV			
AGC-15-L006	0	85	85	1.39
and	90	150	60	2.82
incl	90	120	30	3.75
AGC-15-N004	0	25	25	2.19
and	30	50	20	1.66
and	65	100	35	2.63
incl	65	85	20	3.51
AGC-15-N006	0	100	100	2.46
incl	5	20	15	4.87
AGC-15-O004	30	100	70	1.37
AGC-15-O006	0	25	25	3.71
incl	5	25	20	4.19
AGC-A04	105	115	10	1.60
and	230	240	10	1.03
AGC-B07	NSV			
AGC-B08	95	105	10	1.08
AGC-B09	NSV			
AGC-C03	190	210	20	1.29
AGC-C04	195	230	35	1.15
AGC-D03	50	80	30	1.78
and	105	120	15	1.26
and	135	145	10	1.65
AGC-D04	10	85	75	1.52
and	125	160	35	1.77
AGC-D05	0	45	45	1.25
and	145	165	20	1.61
AGC-E01	9	29	20	
and	34.5	60	25.5	1.80
and	90	110	20	1.20
and	125	135	10	1.55
and	165	185	20	1.74
and	230	256	26	3.08
incl	235	245	10	4.28
AGC-E02	45	55	10	1.22
and	110	135	25	1.61
and	185	215	30	1.93
AGC-E03	110	125	15	2.90
incl	115	125	10	3.09
and	145	205	60	1.96
and	235	245	10	1.08
AGC-E04	5	35	30	2.40

Hole ID	From (ft.)	To (ft.)	Width (ft.)	Cg%
and	46	55	9	2.67
and	65	95	30	2.68
incl	65	80	15	3.71
and	145	256.5	111.5	1.53
AGC-E05	0	24	24	1.88
and	55	256	201	2.26
incl	120	140	20	3.12
incl	175	185	10	3.57
incl	200	215	15	3.55
AGC-E06	50	252	202	2.95
incl	65	75	10	3.78
incl	105	145	40	3.40
incl	160	185	25	3.51
AGC-E07	23	305	282	2.68
incl	23	35	12	3.83
incl	135	160	25	3.37
incl	180	205	25	3.72
incl	235	265	30	3.86
AGC-E08	0	276	276	2.45
incl	45	60	15	3.19
incl	105	140	35	3.41
incl	160	180	20	3.61
incl	195	205	10	3.47
incl	220	240	20	4.01
AGC-E09	10	25	15	1.90
and	75	264	189	2.98
incl	75	105	30	3.74
and	145	155	10	3.73
and	170	240	70	3.35
AGC-F01	0	80	80	1.60
and	170	185	15	1.44
and	225	240	15	1.13
AGC-F02	0	60	60	1.04
and	155	303	148	1.39
AGC-F03	0	35	35	1.40
and	65	255	190	2.10
incl	110	120	10	3.25
AGC-F04	30	256.4	226.4	1.97
incl	240	256.4	16.4	3.77
AGC-F05	0	45	45	2.80
incl	5	20	15	3.81
a	50	245	195	2.55
incl	130	155	25	3.53
and	220	235	15	3.71
AGC-F06	0	30	30	2.60
and	60	249	189	2.27
incl	65	90	25	3.47

Hole ID	From (ft.)	To (ft.)	Width (ft.)	Cg%
incl	205	210	5	3.39
AGC-F07	0	265	265	2.68
incl	10	75	65	3.40
incl	135	180	45	3.34
incl	225	240	15	3.45
AGC-F08	0	25	25	1.91
and	60	256	196	2.54
incl	60	85	25	3.37
incl	115	125	10	3.33
incl	135	145	10	3.61
incl	165	175	10	3.66
incl	215	225	10	3.30
AGC-F09	0	10	10	2.45
and	55	90	35	3.00
incl	65	75	10	3.64
and	105	125	20	2.27
and	190	200	10	1.07
AGC-G01	0	90	90	3.13
incl	5	70	65	3.26
and	110	302	192	2.59
incl	145	160	15	3.37
incl	220	230	10	3.42
AGC-G02	0	49	49	2.96
incl	19	49	30	3.28
and	96	266	170	2.75
incl	155	165	10	3.44
incl	175	200	25	3.26
incl	210	220	10	3.47
incl	230	240	10	3.15
incl	255	266	11	3.56
AGC-G03	0	45	45	2.99
incl	14	24	10	3.23
and	55	276	221	2.64
incl	55	75	20	3.09
incl	85	110	25	3.18
incl	140	150	10	3.37
incl	170	190	20	3.29
incl	255	276	21	3.05
AGC-G04	0	272.5	272.5	2.49
incl	45	55	10	3.17
incl	145	175	30	3.62
incl	225	272.5	47.5	3.10
AGC-G05	0	25	25	1.42
and	40	276	236	2.98
incl	50	75	25	4.17
incl	180	276	96	3.33
AGC-G06	65	286	221	2.41

Hole ID	From (ft.)	To (ft.)	Width (ft.)	Cg%
incl	100	135	35	3.60
AGC-G07	36	245	209	2.02
AGC-G08	45	210	165	1.81
AGC-G09	0	25	25	3.31
and	40	120	80	2.52
incl	40	80	40	3.28
and	160	210	50	2.28
incl	165	180	15	4.13
AGC-H01	0	69	69	3.36
and	76	255	179	2.66
incl	76	85	9	3.63
incl	120	135	15	3.52
incl	225	255	30	5.00
AGC-H02	0	49	49	2.29
and	65	252	187	2.68
incl	80	175	95	3.67
AGC-H03	0	274	274	2.69
incl	0	50	50	3.79
incl	70	160	90	3.87
AGC-H04	0	273	273	2.71
incl	0	29	29	4.06
incl	49	75	26	4.66
incl	125	140	15	3.47
incl	205	230	25	3.16
AGC-H05	0	210	210	1.86
incl	45	60	15	3.28
incl	150	160	10	3.46
AGC-H06	0	25	25	2.26
and	45	255	210	2.35
incl	50	65	15	3.42
incl	80	90	10	3.14
incl	190	225	35	3.39
incl	235	245	10	3.19
AGC-H07	0	30	30	2.50
incl	15	25	10	3.38
and	55	256	201	2.25
incl	65	80	15	3.14
incl	195	210	15	3.44
incl	235	245	10	3.05
AGC-H08	10	25	15	3.11
and	45	245	200	2.18
and	175	185	10	3.62
and	195	205	10	3.64
AGC-H09	0	20	20	1.92
and	30	210	180	2.12
AGC-I01	0	46	46	2.15
and	55	291	236	2.04

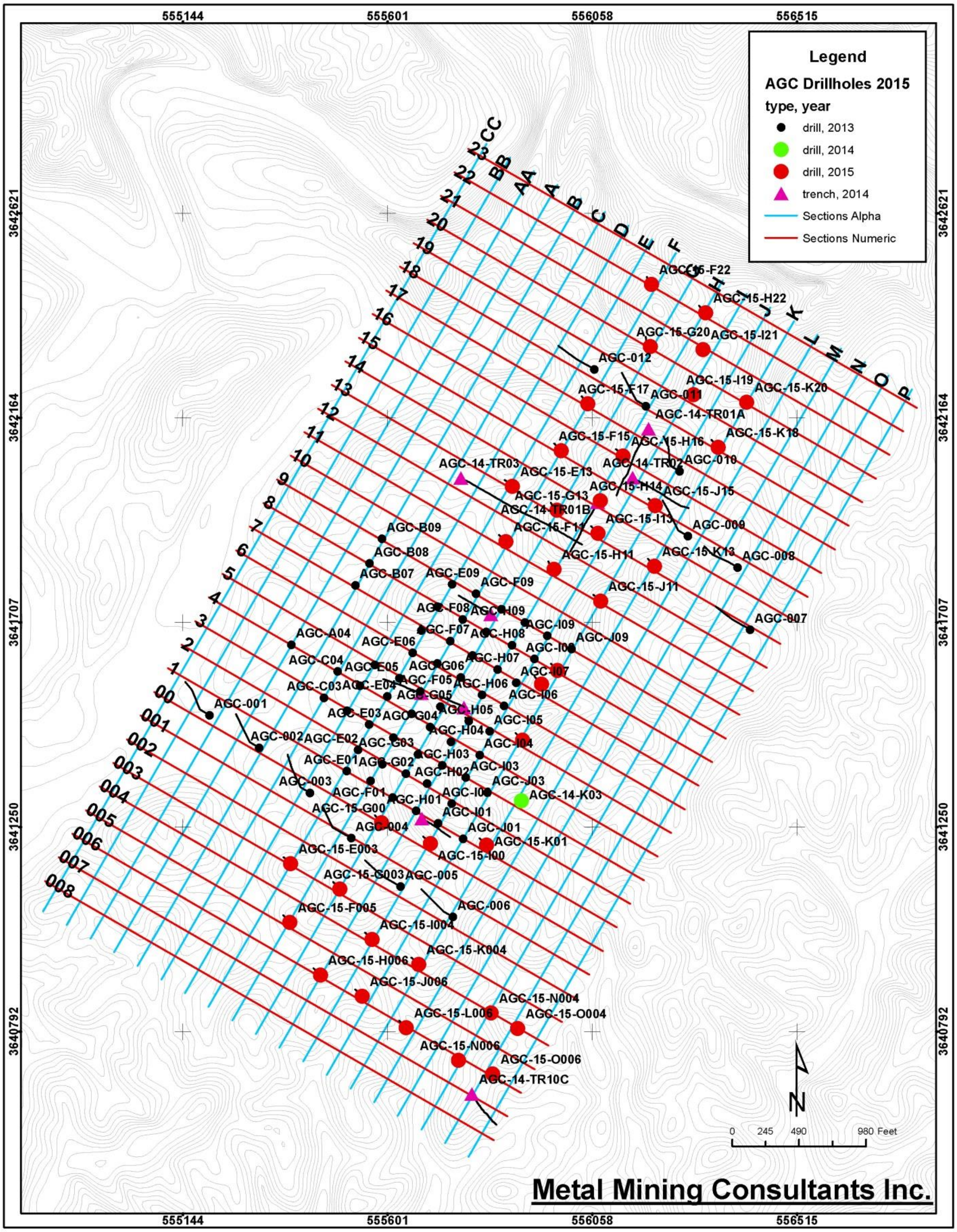
Hole ID	From (ft.)	To (ft.)	Width (ft.)	Cg%
AGC-I02	0	30	30	2.24
and	35	253	218	2.06
incl	35	60	25	3.42
incl	100	110	10	4.05
AGC-I03	0	60	60	3.56
incl	10	25	15	3.44
and	76	250	174	1.75
and	130	140	10	3.88
and	180	190	10	3.40
AGC-I04	0	296	296	1.63
incl	4	12	8	3.56
incl	245	255	10	3.49
AGC-I05	5	240	235	1.73
incl	135	145	10	3.40
AGC-I06	0	215	215	2.14
incl	70	105	35	3.39
and	250	261	11	1.47
AGC-I07	0	275	275	1.93
incl	75	195	120	3.62
AGC-I08	45	260	215	2.36
incl	45	70	25	3.12
incl	150	175	25	3.19
incl	185	200	15	3.47
AGC-I09	0	256	256	2.62
incl	65	140	75	3.48
incl	155	165	10	3.36
incl	205	220	15	3.53
AGC-J01	0	46	46	1.50
and	56	253	197	1.94
incl	60	75	15	3.20
AGC-J03	0	30	30	3.10
incl	9	30	21	3.23
AGC-J09	36	70	34	2.46
and	90	110	20	1.85
incl	130	250	120	1.58

Notes:

1. The cutoffs used are 1% Cg (main intervals) and 3% Cg (sub-intervals) with a minimum of 2 samples and no more than 2 samples of internal dilution.
2. The significant intercepts are downhole widths and are not necessarily true widths or representative of true thickness.

NSV = no significant values.

Figure 10-4: Plan Map of Coosa Project Drill Hole Collars (Metric UTM Grid)



11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Channel Samples 2012

AGC initially used Mineral Labs, Inc. of Salyersville, Kentucky (not accredited) and SGS at Lakefield, Ontario (ISO/IEC 17025:2005 certified) for sample preparation and carbon analyses of channel samples taken in 2012 and for the first drill hole, AGC-001C, as described in a NI 43-101 report by Durgin (2013). Both of these laboratories are independent of AGC.

The first batch of 113 channel samples was prepared by Mineral Labs Inc. at their laboratory in Birmingham and analyzed for carbon by the loss on ignition method at their laboratory in Salyersville. These results were subsequently found to be erroneous and too high, and were discarded. Unfortunately, the laboratory had discarded the sample pulps and rejects and it was not possible to reanalyze them at another laboratory.

The second batch of 115 channel samples was prepared and analyzed for carbon by the LECO method by SGS at Lakefield, and were shown to be reliable by check analyses at ALS Minerals. The sample pulps were also analyzed at Mineral Labs, and comparison with the other laboratories showed that the Mineral Labs assays were unreliable.

Core from the first drill hole, AGC-001C, was submitted to SGS at their laboratory in Birmingham, Alabama for preparation. The pulps were sent to SGS Lakefield for carbon analysis. The marble blank samples were reported to contain greater than 11% Cg as a result of inadequate acid removal of carbonate carbon prior to the LECO analysis. In addition, it was found that the sample preparation laboratory in Birmingham had only prepared a few pieces of core from each bag instead of the entire sample. The SGS analyses of hole AGC-001C were not used.

The unprepared samples, rejects and pulps were retrieved from the two laboratories, recombined by AGC, and submitted to ALS Minerals (ISO/IEC 17025:2005 certified/accredited) for preparation and analysis.

Since then all samples have been prepared and analyzed either by ALS Minerals in Elko, Nevada and Vancouver, British Columbia (ISO 9000:2008 registered and ISO 17025 accredited in North America) or by Activation Laboratories Ltd. (Actlabs) in Ancaster, Ontario (ISO 9000:2008 registered and ISO 17025 accredited, as well as accredited to CAN-P-1579, which is specific to mineral analysis laboratories). Both laboratories are independent of AGC. ALS Minerals was used as the primary laboratory in 2012-13 drill program and Actlabs as the secondary laboratory, while this was reversed for the 2014-15 drilling, trenching and sampling programs.

11.2 Sample Security

Core and samples are stored in a secure warehouse rented by AGC in Sylacauga, Alabama 20 mi (32.5 km) NNE of the project that also has facilities for the field office, core logging, core cutting and core sampling. Core is stored in waxed card boxes on wooden pallets by hole number. Sonic core is stored in long plastic boxes. Pallets are moved by fork lift truck.

Core is collected from the drill rig by an AGC field helper and brought to the warehouse by an AGC truck, where it is logged, photographed and sampled. Diamond core is cut lengthwise by diamond saw in the warehouse, and one-half core is put in a plastic or cloth bag with a consecutive sample number tag, sample number in indelible marker, and sealed with a cable tie. The remaining half-core is returned to the core box and stored in the warehouse for reference. The sample number is marked in the core box by a sample number tag. Sonic core is cut lengthwise with a knife and sampled, numbered and stored in the same way as diamond core.

Samples are put in wooden crates on a pallet for shipping, each holding approximately 1,000 lbs (454 kg). The sample crates are covered but not sealed. They are transported by a UPS courier truck to the ALS Minerals laboratory in Elko, Nevada, or to the Activation Laboratories Ltd. (Actlabs) laboratory in Ancaster, Ontario, for sample preparation and analysis. The coarse sample rejects and pulps are back-shipped in the same crates by UPS to the warehouse in Sylacauga for storage.

Figure 11-1: The Alabama Graphite Core Warehouse and Field Office in Sylacauga

Photo S. Redwood

11.3 Sample Preparation

11.3.1 ALS Minerals

Samples were prepared by ALS Minerals at their laboratory in Elko, Nevada. The sample preparation procedure was to log the samples into the tracking system and add a bar code label (code LOG-22), weight the samples (code WEI-21), dry them at high temperature of up to 120°C (code DRY-21), fine crush by jaw crusher to better than 70% passing -2 mm (code CRU-31), split off 1000 g using a riffle splitter (code SPL-21), and pulverize the 1000 g split to better than 85% passing 75 microns (200 mesh) in a ring and puck style grinding mill (code PUL-32). The entire sample preparation method is referred to by ALS Minerals as code PREP-31B.

11.3.2 Actlabs

Samples were prepared by Actlabs at their laboratory in Ancaster, Ontario. The sample preparation procedure was to log the samples into the tracking system and add a bar code label, weight the samples, dry them at 60°C, crush the entire sample by jaw crusher to better than 90% passing 10 mesh (1.7 mm), split off 250 g using a riffle splitter, and pulverize the split to better than 95% passing 150 mesh (105 microns) in a mild steel ring and puck style grinding mill (code RX1).

11.4 Analyses

11.4.1 ALS Minerals

ALS Minerals shipped the sample pulps from Elko, Nevada to their laboratory in Vancouver, British Columbia for analysis. A 1 g subsample of the sample pulp was analyzed for total carbon by sample combustion in a LECO induction furnace at high temperature which generates carbon dioxide. This is quantitatively detected by infrared spectroscopy and reported as percent carbon, with a range of detection of 0.01%-50% (code C-IR07). Inorganic carbon (carbonate) was analyzed by carbon dioxide (CO_2) coulometery, with a range of detection of 0.2%-15% CO_2 (code C-GAS05). The procedure is to acidify the sample with perchloric acid ($HClO_4$) in a heated reaction vessel to evolve free carbon dioxide. This is subsequently transferred to a CO_2 coulometer using a carbon-free gas, where the CO_2 is quantitatively absorbed and reacts with monoethanolamine in the presence of an indicator that fades in color with increasing CO_2 concentration. The color change is detected by a photo-cell and is used to determine the amount of CO_2 in the sample. Carbon is calculated from carbon dioxide and the results are rounded to two decimal points and one decimal point respectively. Check calculations often show variation of ±0.01% carbon due to rounding.

The sample weight and results for C (C-IR07), C (C-GAS05) and CO2 (C-GAS05) are reported to AGC on Excel spreadsheets and in Certificates of Analysis in secure Adobe Acrobat file format, transmitted by email and available on a secure internet sample tracking site called Webtrieve™.

Graphite carbon was calculated by AGC by subtracting inorganic carbon from total carbon. This assumes that the only other form of carbon present in the samples, other than graphite carbon, is contained within carbonates.

Multi-element analysis was carried out for one entire drill hole (AGC-010) by ALS Minerals at their Reno, Nevada laboratory. The samples were analyzed for 53 elements by aqua regia digestion and ICPMS analysis (method code ME-MS41L).

11.4.2 Actlabs

Actlabs analyzed graphitic carbon on a 0.5 g sample in an Eltra resistance or induction furnace with measurement by infrared spectrometer (method code 5D-C Graphitic Infrared). The sample is subjected to a multistage furnace treatment to remove all forms of carbon with the exception of graphitic carbon, with no acid digestion required. The sample is heated to 1000°C in nitrogen in a resistance or induction furnace to burn off CO_2 in carbonate and any organic carbon, leaving graphite carbon behind. The residue is then combusted in an oxygen environment to oxidize the graphite to determine the graphite content. The laboratory report graphite carbon (C-Graph) with a lower limit of detection of 0.05%. The sample weight and results for Cg are reported to AGC on Excel spreadsheets and in Certificates of Analysis in Adobe Acrobat file format.

11.5 Specific Gravity

A total of 263 specific gravity measurements were carried out by ALS Minerals at their laboratory in Elko, Nevada in 2013 on 3 to 5 inch (7.6 to 12.7 cm) samples of split core sampled by AGC from several holes from the 2012 program. The holes were selected from around the grid for representability of different rock types. A further 12 determinations were made by ALS Minerals in 2015 on samples from the oxide and transition zones in the 2015 drill program. Seven check specific gravity determinations were made by Actlabs in September 2015.

The specific gravity was determined after wax coating the sample (ALS Minerals method OA-GRA08n). At Actlabs the specific gravity was determined without wax coating (code Specific Gravity Core). The results are described in Section 14.1.3.

11.6 Quality Assurance and Quality Control (QA-QC)

AGC have a comprehensive QA-QC program for trench and drill core samples that meets the best practices guidelines currently used within the industry. QA-QC samples were not inserted in the channel samples as these were not used for resource estimation.

The company inserted 1 coarse blank in every 20 samples, and took 1 field (core) duplicate in every 20 samples, giving approximately 10% QC samples. No certified standard reference materials (CSRM) were available commercially for graphite carbon for the 2012-14 programs. For the 2012 drill programs, AGC monitored the laboratory's internal CSRM data. AGC made two CSRM in 2015 and inserted 1 CSRM in every 20 samples in the 2015 drill program. Replicate analyses at a second certified laboratory were carried out on a complete hole at the start of the 2012 drill program, and after the end of the drill programs replicate analyses were made of approximately 10% of all samples at the second laboratory.

The following sections describe the results of the QA-QC for the samples that were used to make the resource estimate.

11.6.1 Certified Standard Reference Materials (CSRM)

2012 Drill Programs

No CSRM were available commercially for graphite in order to maintain an independent control. Instead, the internal standards used by the laboratory, ALS Minerals, were plotted. Four CSRM with a range of carbon grades were used routinely, and the recommended values and 95% certainty limits, as supplied by ALS Minerals, are shown in Table 11-1. These samples were manufactured by CANMET Mining and Mineral Services Laboratories, Ottawa, Canada as part of the Canadian Certified Reference Materials Project (CCRMP). These samples are certified for other elements and carbon is uncertified and is given as an "informational value", "provisional value" or "approximate value". Plots of the results of total carbon analyses by sample (time) are shown in Figure 11-2 to Figure 11-5. The results are within the upper and lower confidence limits and show no trends or drift with time, and indicate that the analyses have good precision and accuracy.

Table 11-1: List of CANMET CSRM used for Carbon by ALS Minerals

CSRM	Ct% Lower Limit	Ct% Rec Value	Ct% Upper Limit
NBM-1	0.75	0.79	0.83
MA-1b	2.34	2.44	2.54
DS-1	3.01	3.13	3.25
STSD-3	8.09	8.40	8.71

Figure 11-2: Plot of Results of CSRM NBM-1 (2012 Drilling Programs)

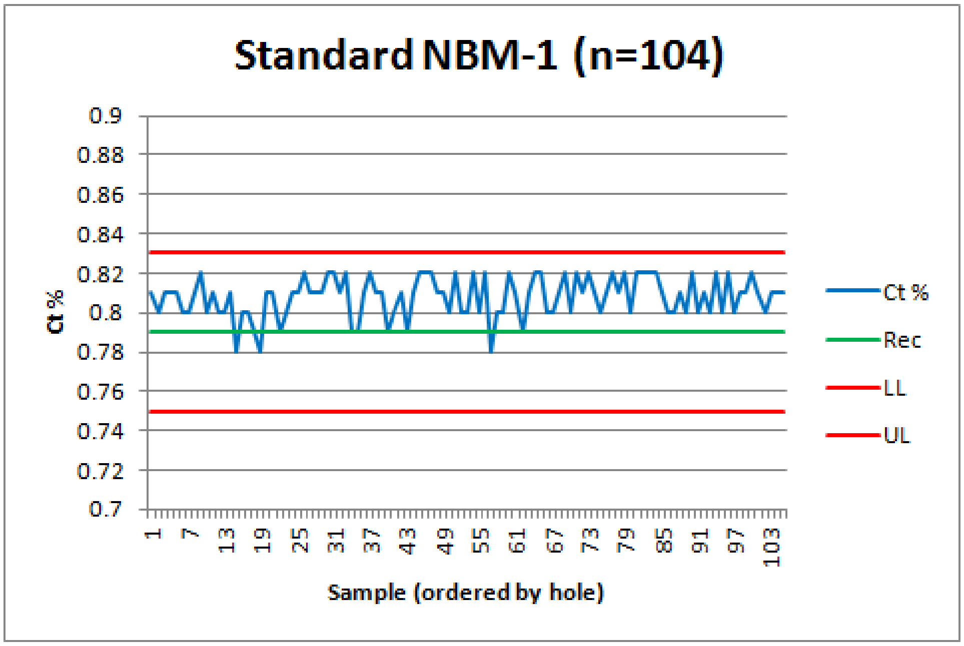


Figure 11.3: Plot of Results of CSRM MA-1b (2012 Drilling Programs)

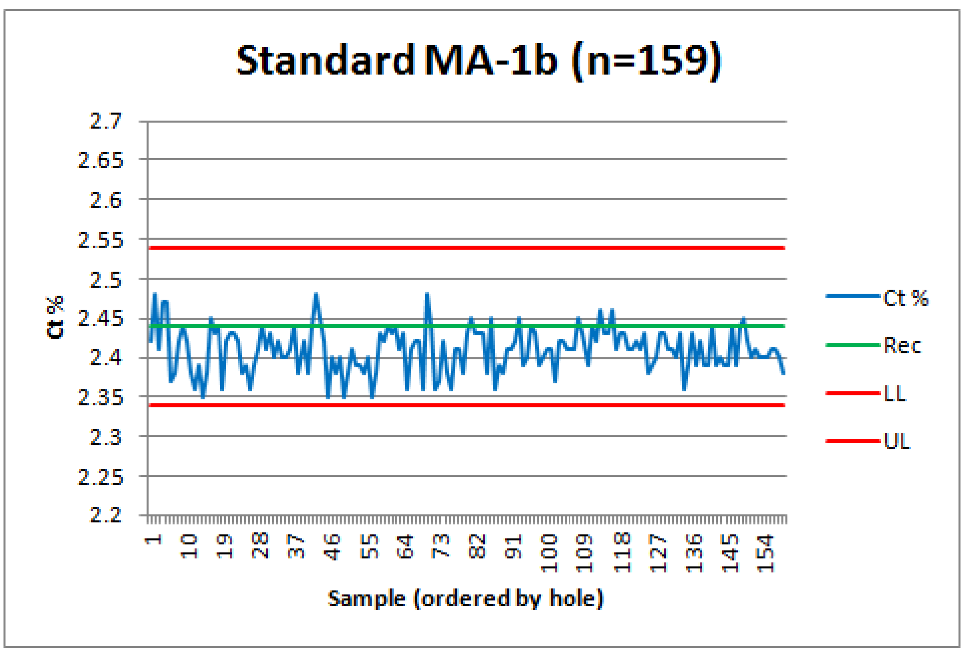


Figure 11-4: Plot of Results of CSRM DS-1 (2012 Drilling Programs)

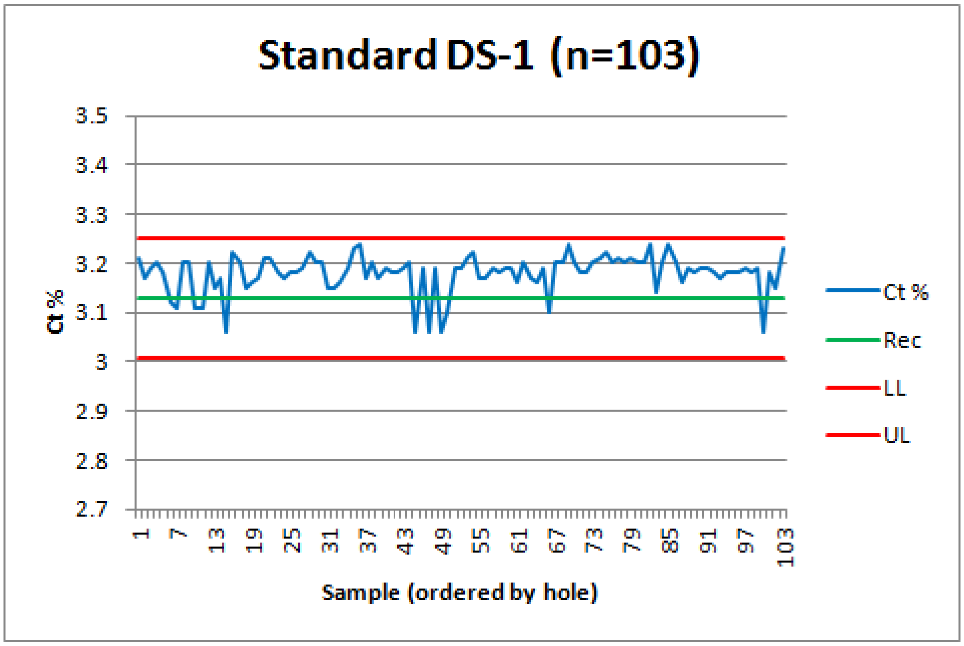
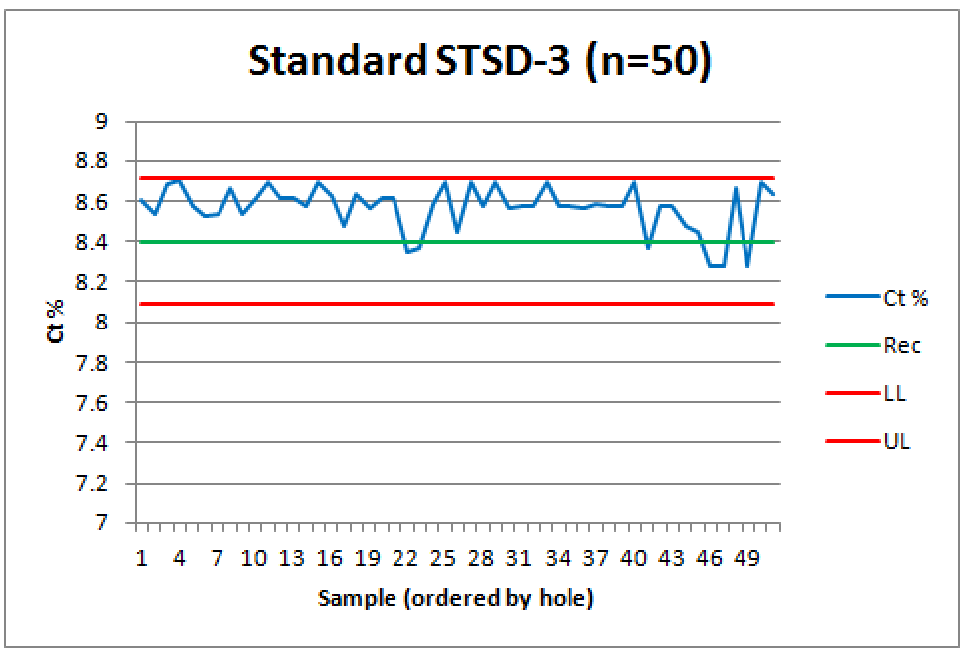


Figure 11-5:Plot of Results of CRSM STSD-3 (2012 Drilling Programs)



2015 Drill Program

AGC had two CSRM made by Actlabs in 2015 using coarse rejects of oxidized graphite schist from Coosa trench samples. The samples were pulverized, mixed and homogenized, and 10 samples of each were analyzed at four laboratories (Actlabs, SGS, ALS Minerals and Acme), from which the mean and standard deviation were calculated (the Acme data was not used as they use a different ashing and digestion which gave a different mean and standard deviation). The recommended values are shown in Table 11-2.

Table 11-2: AGC Coosa CSRM Recommended Values

CSRM	Mean Cg%	Standard Deviation Cg%
Standard A	2.47	0.0395
Standard B	4.07	0.1403

The CSRM check analytical precision and accuracy, analytical drift and sample switches. The CSRM are monitored by performance gates which are graphs with sample number or time on the x-axis and values on the y-axis. There are horizontal lines for the recommended mean value (green), and ±2 (yellow) and ±3 (red) standard deviations (SD). CSRM values within ± 2 SD are accepted; an isolated sample above ± 2 SD but below ± 3 SD is acceptable but is a warning; two consecutive samples above ± 2 SD are rejected; and any sample above ± 3 SD is rejected.

Scatter plots of the results of the Cg analyses by sample (time) are shown in Figure 11-6 and Figure 11-7. The results are acceptable and show no trends or drift with time, and indicate that the analyses have good precision and accuracy.

Figure 11-6: Plot of Results of CSRM Standard A (2015 Drilling Programs)

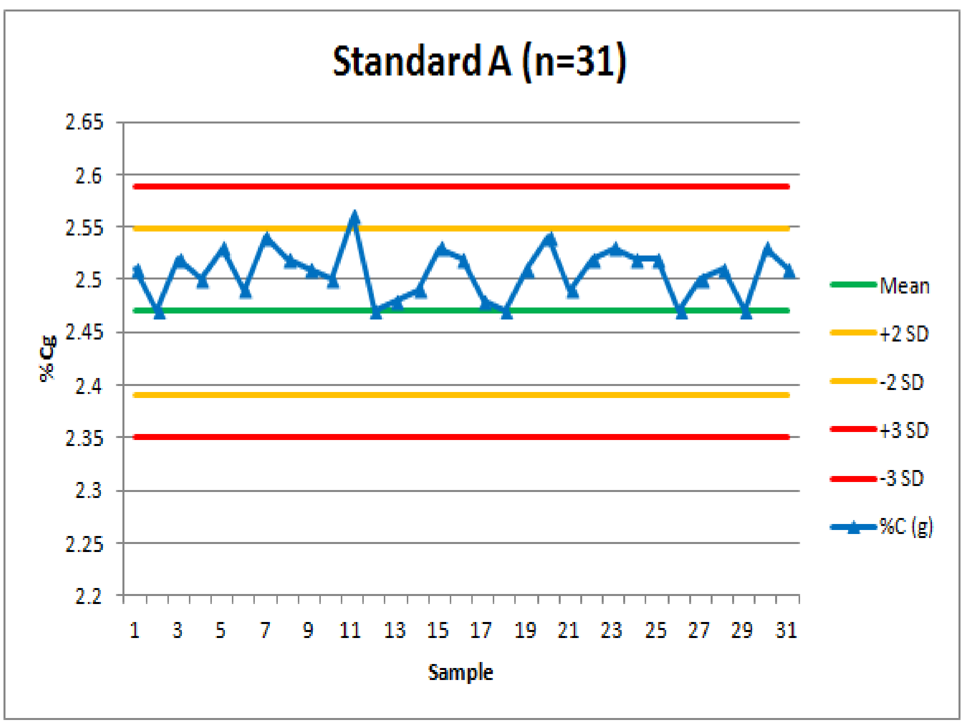
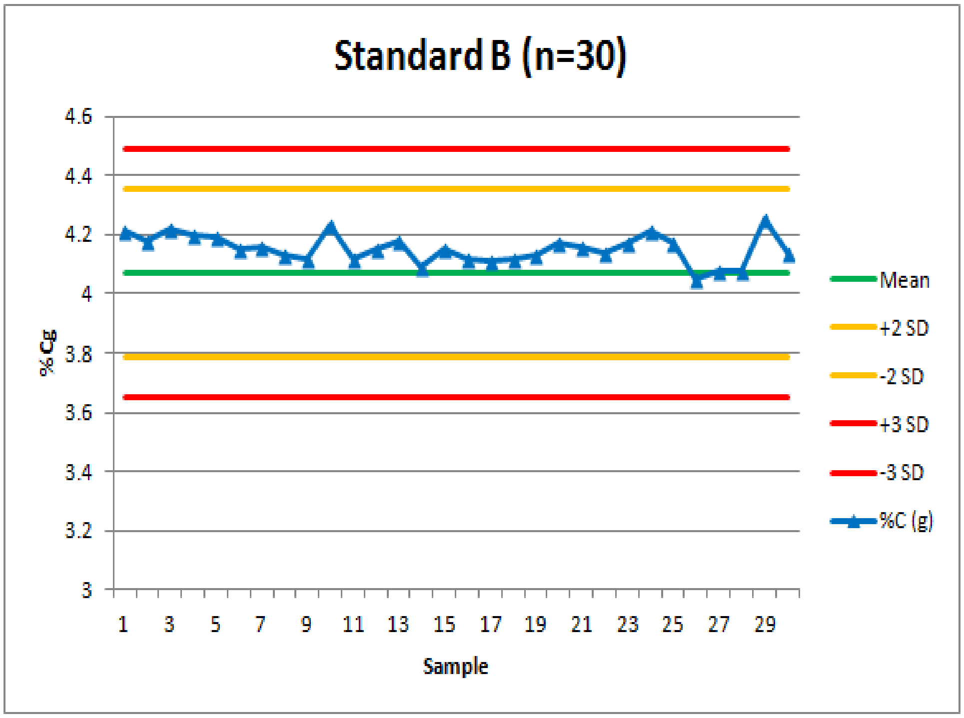


Figure 11-7: Plot of Results of CSRM Standard B (2015 Drilling Programs)



11.6.2 *Blanks*

Coarse blanks were inserted in the drill and trench samples at approximately one every 100 ft (30 m), which equals one in every twentieth sample (approximately 5%).

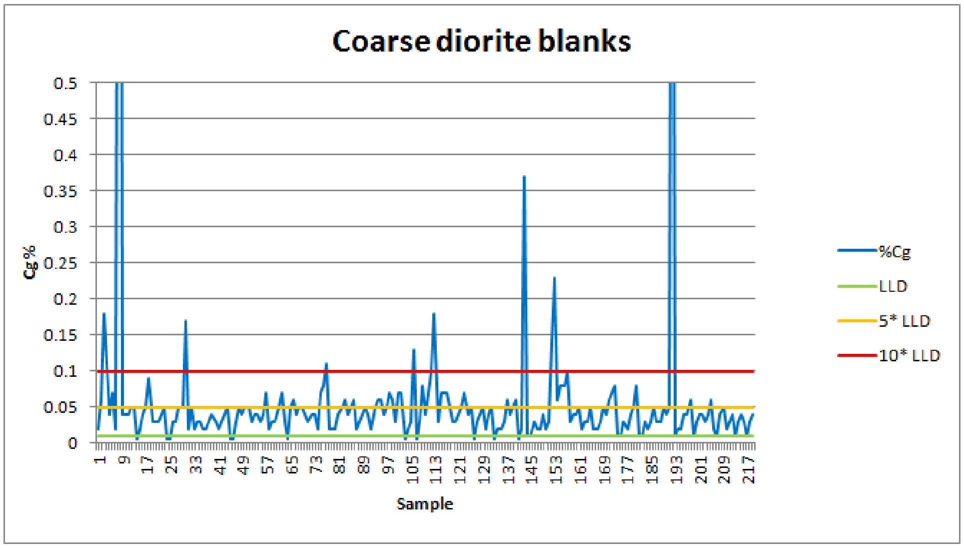
Local marble was initially used as a blank but reported high C in Leco analyses because of problems with incomplete dissolution of carbonate. The blank was then changed to diorite in the form of coarse rock chips of 1 to 2 inch (2.5-5.1 cm) size bought as landscape rock from a local hardware store. Neither of these samples were analyzed previously for carbon or other elements. The diorite blank was introduced together with the marble blank from hole AGC-002C, and diorite was used exclusively from hole AGC-007C onwards. A total of 18 marble blanks and 221 diorite blanks were analyzed out of a total of 3856 unknown drill samples in the 2012 drill programs (6.2% total blanks including 5.7% diorite blanks; note that the total unknown samples analyzed was higher and some have been eliminated from the database due to overlap between core samples and sonic samples). The diorite blanks are plotted in Figure 11-8 with reference lines at the lower limit of detection (0.01%), and five and ten times the lower limit of detection. Eleven samples (5%) are greater than 0.10%, including 4 samples (1.8%) above 0.20 % Cg, with values of 0.23%, 0.37%, 1.35% and 1.91% Cg, which may be contamination, sample switches or mislabeling.

The diorite blanks from the 2014 trenching program are shown in a scatter plot in Figure 11-9 which shows that all but two samples were below the detection limit.

The diorite blanks from the 2015 drilling program are shown in a scatter plot in Figure 11-10 which shows that all samples were below the detection limit.

Overall the blank results are acceptable and show no systematic cross contamination of carbon between samples during sample preparation, or contamination during analysis.

Figure 11-8: Scatter Plot of Coarse Diorite Blanks (n=221) (2012 Drilling Programs)



Note: Samples below detection limit were replaced by half of the detection limit.

Figure 11-9: Scatter Plot of Coarse Blanks (2014 Trenching Program)

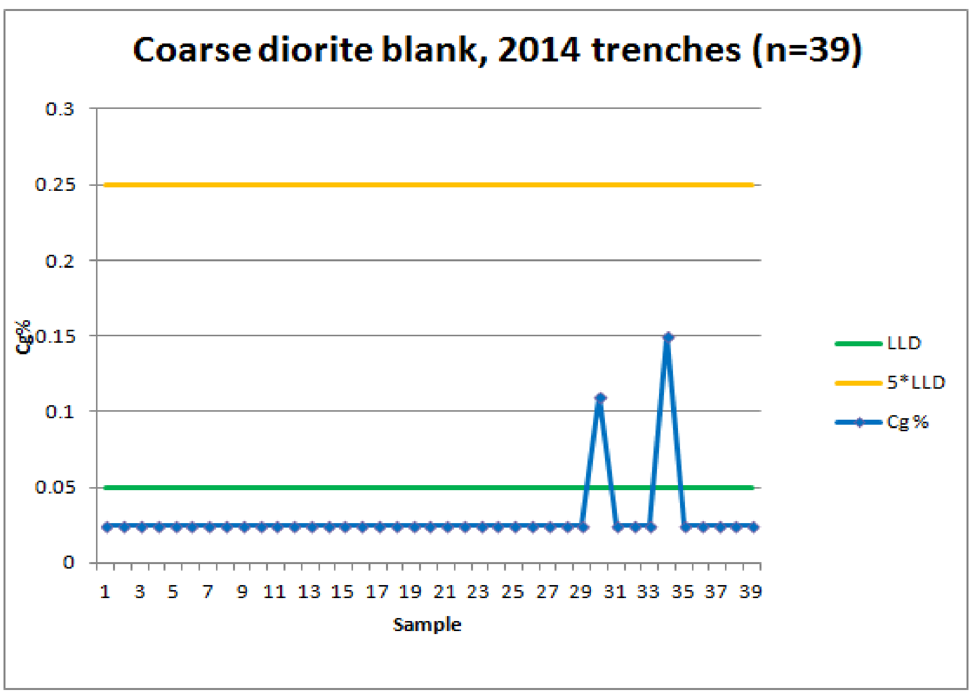
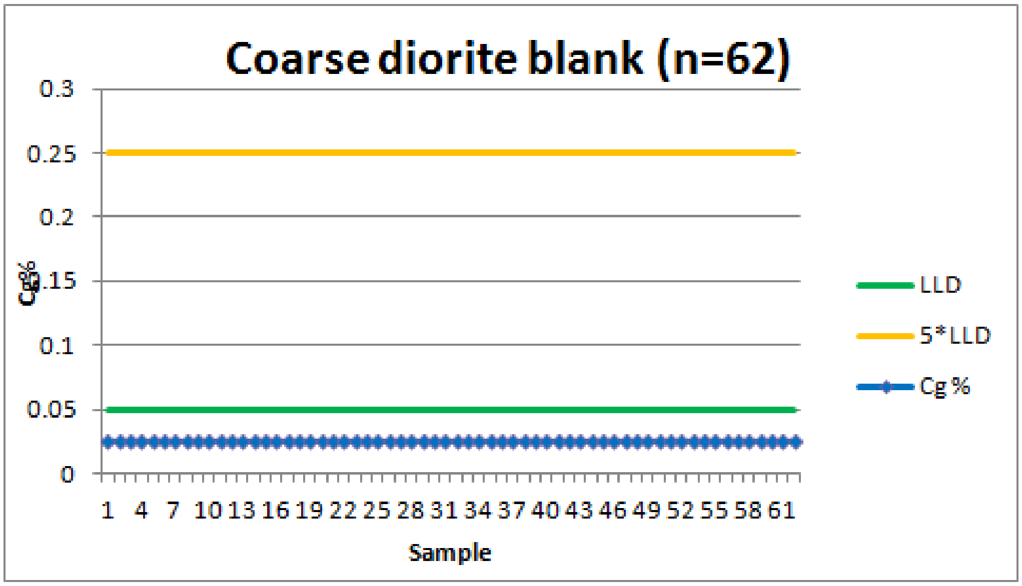


Figure 11-10: Scatter Plot of Coarse Blanks (2015 Drilling Program)



Note: Samples below detection limit were replaced by half of the detection limit.

11.6.3 Field Duplicates

One field (core or trench) duplicate was collected every 100 ft. (30 m) approximately, which equals one in every twentieth sample (approximately 5%).

Core duplicates were taken by cutting the normal half core sample lengthwise to give two one-quarter core samples which were submitted for analysis as original and subsequent duplicate. These samples are half of the weight of normal half-core samples which may thus introduce greater inhomogeneity than average. Trench duplicates were made by taking another sample along the same channel as the original.

A total of 160 pairs of field duplicates were analyzed out of a total of 3,856 unknown drill samples (4.1%) from the 2012 drill programs. They are shown in a scatter plot in Figure 11-11 and in a plot of relative difference versus original value in Figure 11-12. This shows that 91% of the duplicates are within ±20% of the average. The overall correlation is close to unity with scatter distributed evenly on either side, interpreted as geological heterogeneity, and there is no systematic bias. The average of the original samples is 1.86% Cg, and the duplicates is 1.86% Cg, with a relative difference of -0.23%.

The 39 pairs of field (trench channel) duplicates from the 2014 trenching program that was used in the resource estimate are shown in Figure 11-13. This shows that 87% of the duplicates are within ±20% of the average. This is slightly lower than the core samples and is a function of greater sample variability in trench channels. The overall correlation is close to unity with scatter distributed evenly on either side, interpreted as geological heterogeneity, and there is no systematic bias. The average of the original samples is 2.26% Cg, and the duplicates is 2.31% Cg, with a relative difference of 2.26%.

A total of 61 pairs of field duplicates were analyzed out of a total of 1,064 unknown drill samples (5.7%) from the 2015 drill program. They are shown in a scatter plot in Figure 11-14 and in a plot of relative difference versus original analysis in Figure 11-15**Error! Reference source not found.**. This shows that 93% of the duplicates are within ±20% of the original, with four outliers. The overall correlation is close to unity with scatter distributed evenly on either side, interpreted as geological heterogeneity, and again there is no systematic bias. The average of the original samples is 2.57% Cg, and the duplicates is 2.61% Cg, with a relative difference of 1.50%.

It is concluded that the duplicate core samples show geological variability, which is expected to be higher than in normal samples due to the duplicates being half the normal sample weight, but there is no systematic bias, and the relative difference of the average grade of all originals and duplicates very low, i.e. the average values are almost identical.

Figure 11-11: Scatter Plot of Field (core) Duplicates (2012 Drilling Programs)

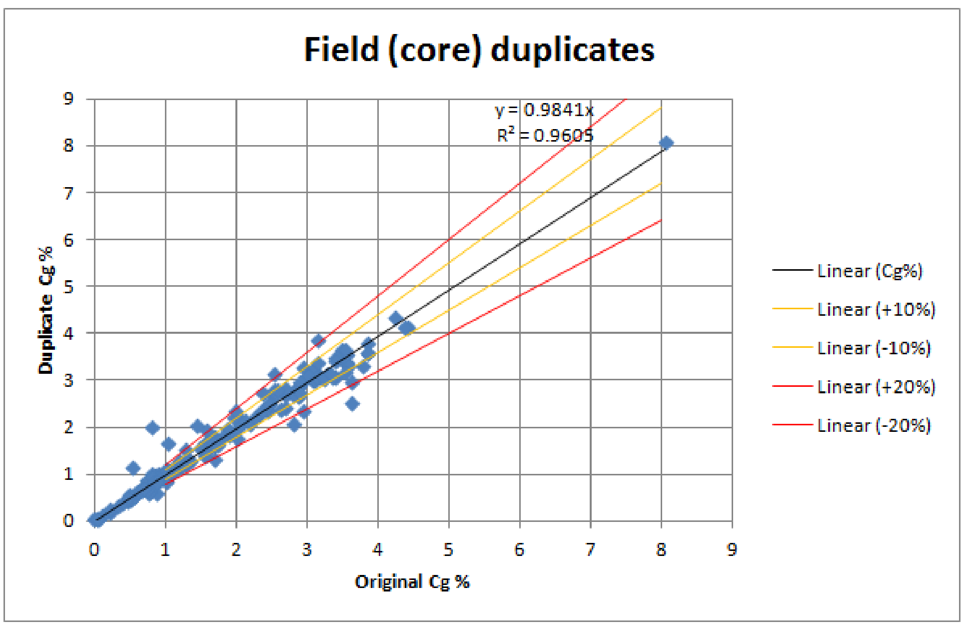


Figure 11-12: Plot of Mean Versus Relative Difference for Field (core) Duplicate Samples (2012 Drilling Programs)

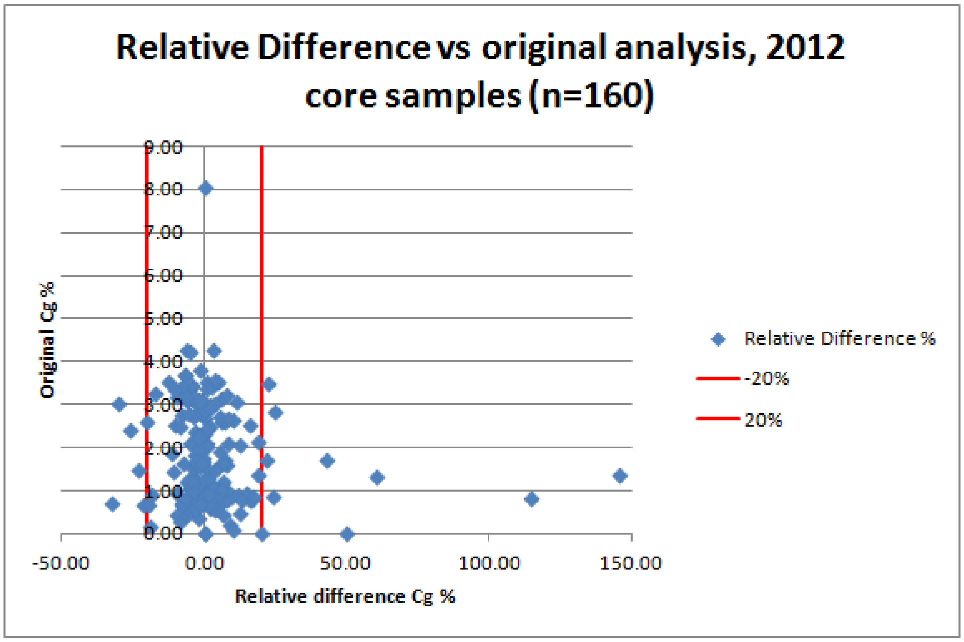


Figure 11-13: Scatter Plot of Field (trench) Duplicates (2014 Trenching Program)

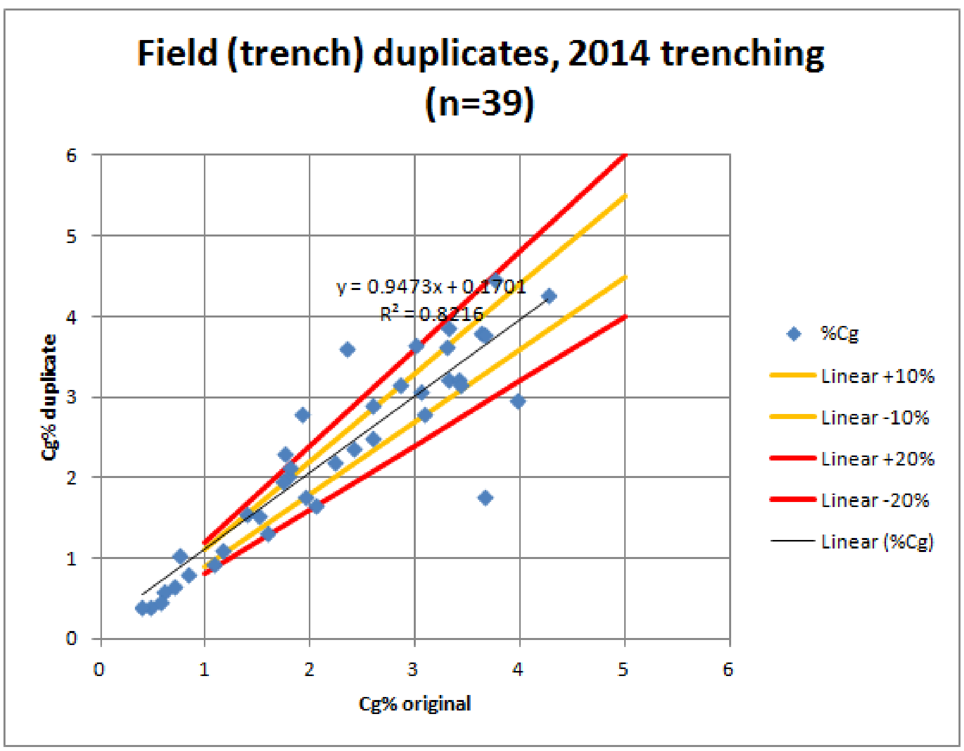


Figure 11-14: Scatter Plot of Field (core) Duplicates (2015 Drilling Program)

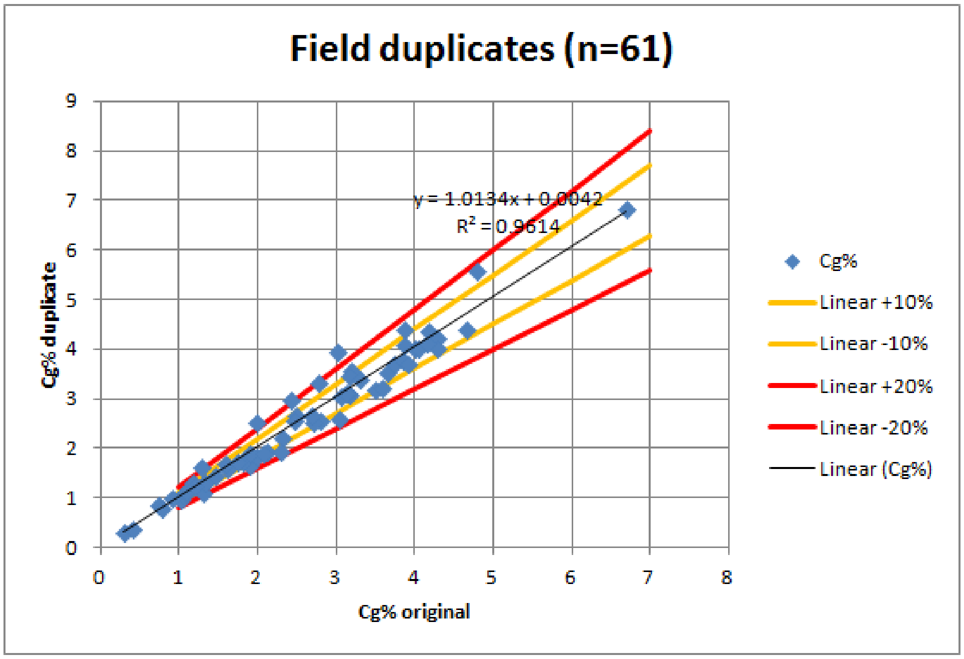
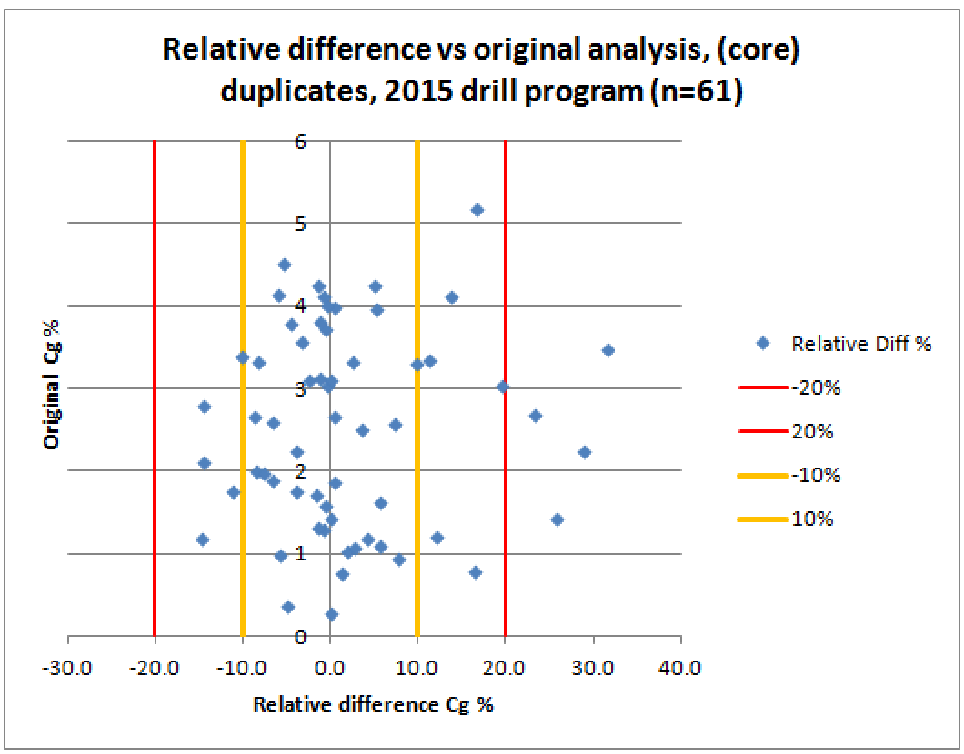


Figure 11-15: Plot of Mean Versus Relative Difference for Field (core) Duplicate Samples (2015 Drilling Program)



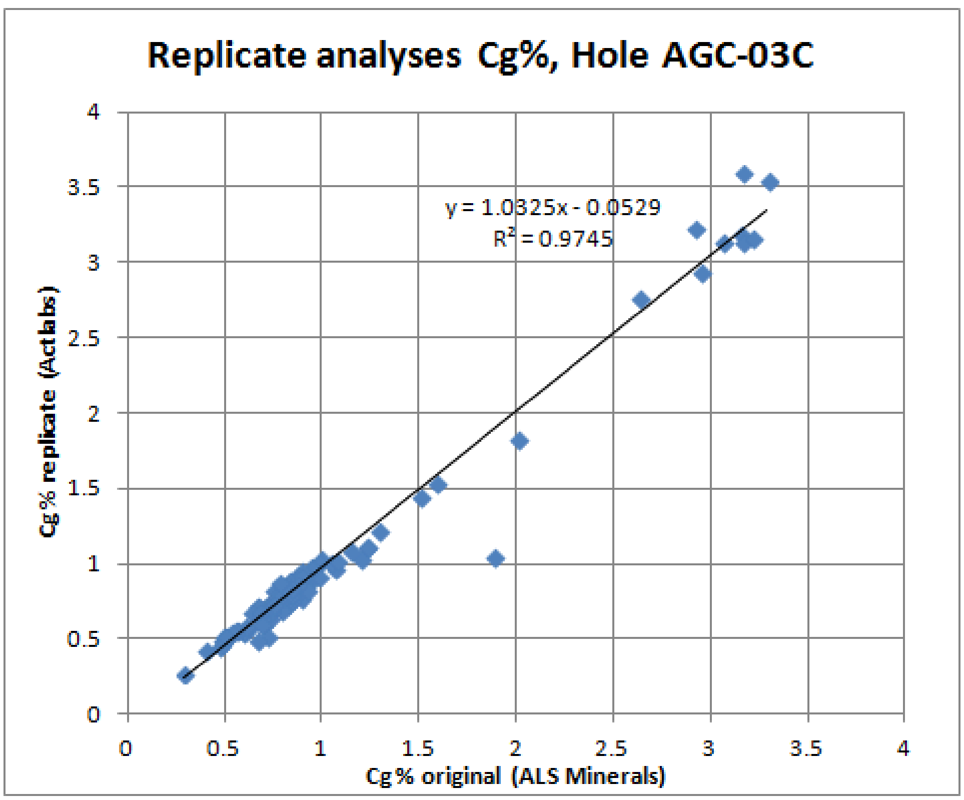
11.6.4 Replicate Analyses

2012 drill programs

AGC carried out replicate analyses of a complete drill hole early in the drill program, and of approximately 11.4% of samples from all drill holes after the end of the drill program.

AGC reanalyzed all samples from hole AGC-03C at a second laboratory in the early stages of the drill program. The preparation was carried out by a local laboratory which split the sample pulp into two, hence these are replicate analyses (same pulp). One pulp was sent to the prime laboratory, ALS Minerals, and the second sent to the second laboratory, Actlabs. A scatter plot of the two sets of analyses is shown in Figure 11-16 and shows a good correlation, with some scatter about the mean above 1% Cg. The average grade of the original analyses is 1.03% Cg versus the 1.01% for the replicate analyses, a difference of 0.02% Cg, and a relative difference of 1.9%.

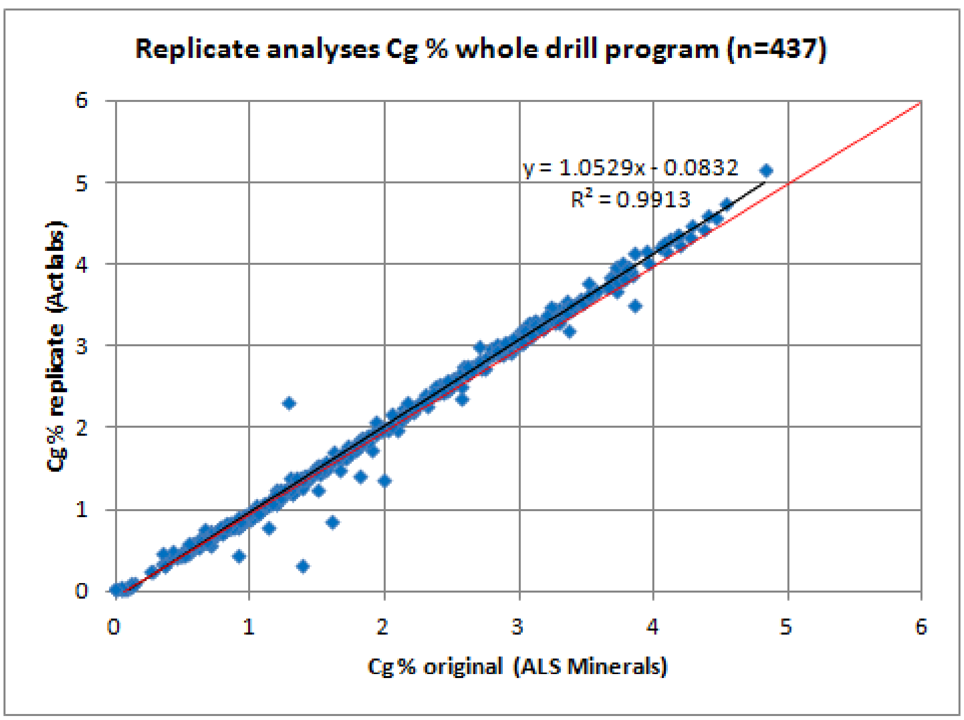
Figure 11-16: Scatter Plot of Replicate Analyses of Hole AGC-03C (n=94)



AGC sent 438 sample pulps from the 2012 drill programs to Actlabs for replicate analyses. Samples were selected by taking every tenth sample. This represents 11.3% of the total of 3856 unknown drill samples. One replicate sample was a marble blank which had originally given an unreliable analysis and so it was eliminated, leaving 437 samples to plot. A scatter plot of the original and replicate analyses is shown in Figure 11-17 and shows a close correlation between the two laboratories, with some scatter in the 1-2% range, and a possible slight high bias at Actlabs above 2.0% Cg. The average grade of the original analyses is 1.85% Cg and the average grade of the replicate analyses is also 1.86%, with a relative difference of -0.8%.

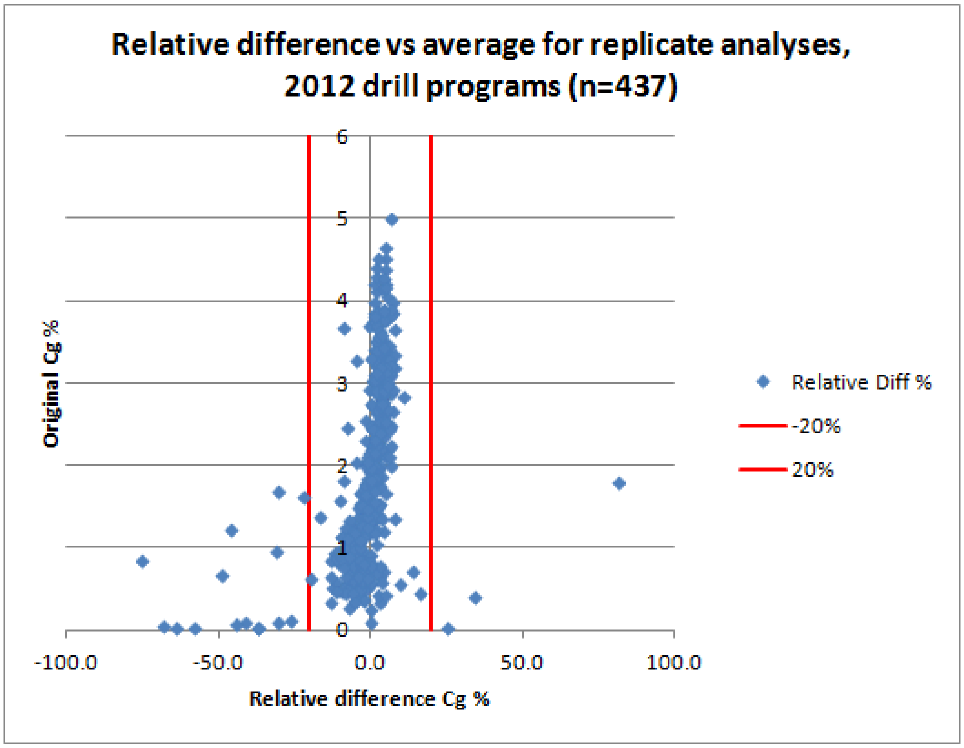
A plot of the mean versus relative difference is shown in Figure 11-18. This shows that the majority of the samples (95.0%) have less than ±20% relative difference, in other words the reproducibility is very good. Samples with a high relative difference along the x-axis are a data artifact close to or below the lower limit of detection (LLD): Actlabs has a LLD of 0.05% Cg, with 8 samples below this which were converted to half of the LLD to plot, whereas ALS has a lower LLD of 0.01% with 2 samples below the LLD. The plot also shows a slight trend for the original ALS Minerals analyses to be higher than the replicates below about 1.5% Cg, with more scatter, whereas the Actlabs analyses are slightly higher than the originals above 3% Cg. These small systematic differences are interpreted to be inter-laboratory differences which may be due to differences such as analytical procedure, reagents used and instrument calibration. The differences are not considered to be significant. Overall the data shows very good reproducibility.

Figure 11-17: Scatter Plot of Replicate Analyses of Cg for the 2012 Drill Programs



Note: 8 values <0.05% Cg in Actlabs replaced by 0.5*LLD (0.025% Cg) and 2 values <0.01% Cg in ALS Minerals replaced by 0.5*LLD (0.005% Cg).

Figure 11-18: Plot of Mean vs. Relative Difference of Cg for Replicate Samples from the 2012 Drill Programs

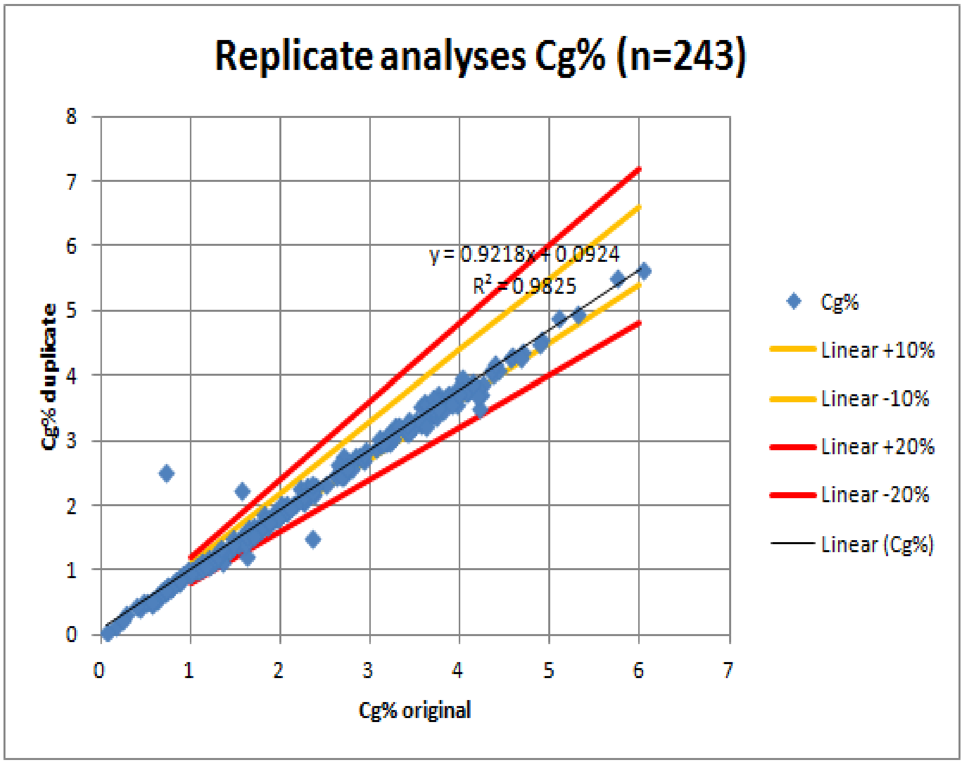


Note: 8 values <0.05% Cg in Actlabs replaced by 0.5*LLD (0.025% Cg) and 2 values <0.01% Cg in ALS Minerals replaced by 0.5*LLD (0.005% Cg). One sample with 400% relative difference plots off chart on the x-axis and is an artifact of both samples below different detection limits.

2014 Trench Program & 2015 drill program

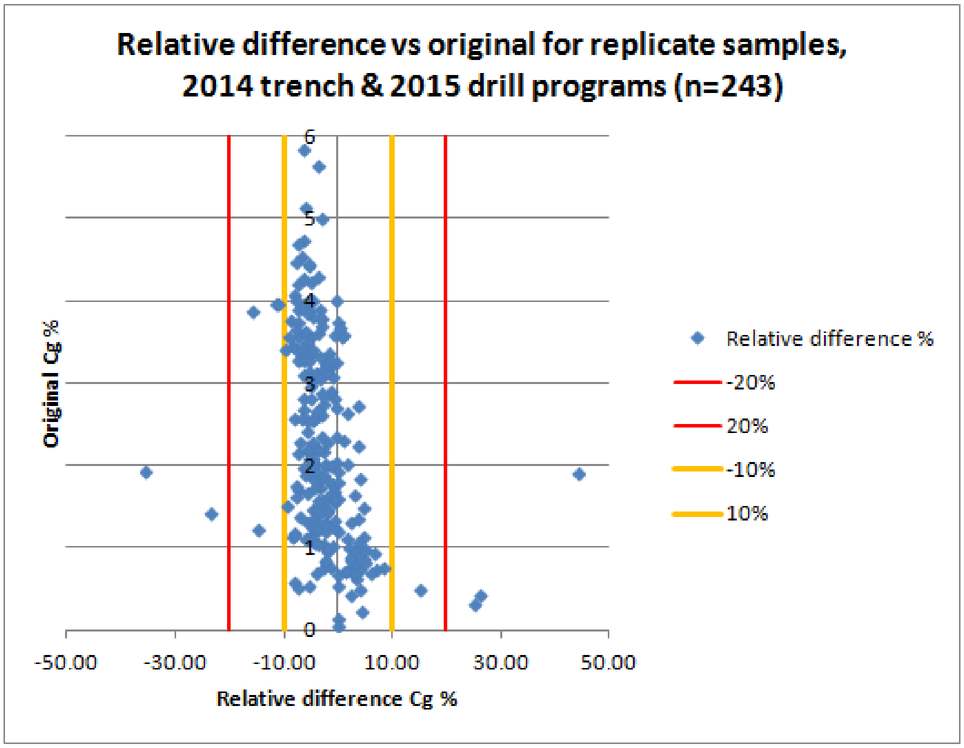
AGC sent 244 sample pulps from the 2015 drill program and 2014 trenches within the drill grid to ALS Minerals for replicate analyses. There was insufficient sample for one leaving 243 to plot. Samples were selected by taking every tenth sample. A scatter plot of the original and replicate analyses is shown in Figure 11-19 and shows a close correlation between the two laboratories, with a few outliers. The average grade of the original analyses is 2.30% Cg and the average grade of the replicate analyses is 2.22%, with a relative difference of -3.8%.

Figure 11-19: Scatter Plot of Replicate Analyses of Cg for the 2014 Trench Program and the 2015 Drill Program



A plot of the relative difference versus the original is shown in Figure 11-20. This shows that the majority of the samples (95.0%) have less than ±10% relative difference, in other words the reproducibility is very good. The small differences are not considered to be significant. There are also 6 outliers with a relative difference of >20% which may be due to sample heterogeneity.

Figure 11-20: Plot of Mean vs. Relative Difference of Cg for Replicate Samples from the 2014 Trench Program and the 2015 Drill Program



Note: one sample plots off the chart with 256% relative difference (0.71% Cg original vs 2.53% Cg check).

11.7 Author's Opinion

The authors conclude that sampling, sample security, sample preparation and analyses have been carried out in accordance with best current industry standard practices and are suitable for estimation of mineral resources and to plan further exploration. Sampling and analyses include quality assurance and quality control procedures. The QA-QC results show that the sampling and analytical data are reliable. The analyses for Cg show good reproducibility within and between laboratories.

12 DATA VERIFICATION

The authors verified the data used upon in this report by visiting the property and confirming the geology and mineralization, and by review of the QA-QC, the laboratory certificates and the database. In addition, a number of check samples were sent for analysis at an independent laboratory.

MMC requested splits of ten random coarse reject drill core samples for analysis at a second laboratory. AGC took a split from the coarse reject by rolling the sample bag, pouring the sample into a 5-gallon bucket, stirring the sample, then taking a scoop from the top.

The samples were shipped by AGC to MMC, and by MMC to Acme Analytical Laboratories (Vancouver) Ltd., Vancouver, part of the Acme Labs Group. Acme is independent of AGC and MMC. The samples were crushed to 80% passing 2 mm (10 mesh), homogenized, riffle split and 250 g pulverized to 85% passing minus 200 mesh (75 microns) (method code R200-250). A 0.2 g split was analyzed for graphite by ignition to 600°C for one hour, hydrochloric acid leach (to remove carbonates), and Leco analysis of the residue (method code 2A09). Another 15 g split was analyzed for multielements by aqua regia digestion and inductively coupled plasma mass spectrometer (ICP-MS) analysis (method code Ultratrace 1F02). One blank sample was inserted for QC and returned a satisfactory value below the detection limit.

The check samples consistently gave lower grades than the original samples. The check samples are between 10.3% and 22.0% lower than the original samples for individual samples, with an average of 14.0% lower (originals average 1.69% Cg, checks average 1.46% Cg, difference -0.23%).

In order to check this, a second batch of 20 samples was sent directly from AGC to Acme. The samples comprise both the coarse rejects and pulps of 10 samples. A plot of the Acme results for sample pulps against new pulps prepared from coarse rejects shows a good correlation indicating good precision. However, the Acme pulps are consistently lower than ALS analyses with average of 0.83% Cg vs 1.20% Cg, with an average difference of 30% lower, and a variation of individual samples from 17% to 58% lower. For new pulps prepared from coarse rejects, the average is 0.88% Cg vs 1.20% Cg in the originals, with an average difference of 26% lower, and a variation of individual samples from 2% to 60% lower.

The trend lines and averages are significantly different between batches 1 and 2. This indicates different accuracy between the two batches and further indicates the issue is an analytical problem at Acme. Actlabs found a similar problem with the Acme results when preparing CSRM for AGC in 2015 and concluded it was due to a different ashing/digestion method.

For all samples the average of check analyses is 22% lower (average originals 1.36% Cg, checks 1.6% Cg, range of individual samples 2% to 60% lower, median 25%).

It is concluded from the check samples that:

- The check samples confirm the presence of graphite carbon in the samples in amounts proportional to the original analyses.
- The amount of graphite carbon in the check samples is consistently lower than in the originals. This is interpreted to be due to an analytical issue at Acme.
- The extensive QA/QC data collected during the drilling program and described in Section 11.3 of this report shows that the original ALS analyses are reliable. In particular, the large number of replicate samples analyzed at different stages of the drill program at another independent laboratory, Actlabs, show a very good correlation.
- The check sampling was not carried out for QA/QC. The issue needs to be investigated with Acme before carrying out any further analyses at that lab.

The authors conclude that:

- Sampling, sample security, sample preparation, and analyses have been carried out in accordance with best current industry standard practices and are suitable for estimation of mineral resources and to plan further exploration.
- The exploration programs are well planned and executed and supply sufficient information to plan further exploration.
- Sampling and analyses include QA/QC procedures.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Bench-scale metallurgical testwork programs on samples from the Coosa County deposit commenced at the SGS Minerals laboratories in Lakefield in April 2013 and culminated in a full flowsheet development program in August 2015. This work focused on the grindability and flotation response of the various samples

In addition, a program of preliminary test work on flotation concentrates was initiated at a Canadian laboratory in 2015 to examine the response to thermal purification processing.

A summary of results from the various test programs is provided chronologically in the following sections.

13.2 2013 Metallurgical Testwork

Three different samples weighing approximately 20 kg each were received at the SGS Lakefield site in April 2013. The three samples were obtained from surface and drill core covering a depth from 0 feet to 270 feet.

13.2.1 Sample Preparation and Head Characterisation

Each sample was stage crushed to -6 mesh, homogenized, and sub-sampled for chemical analysis. The balance of the material was rotary split into 2 kg test charges. A summary of the pertinent head assay results are presented in Table 13-1:. The three samples graded between 2.92% graphitic carbon and 3.71% graphitic carbon.

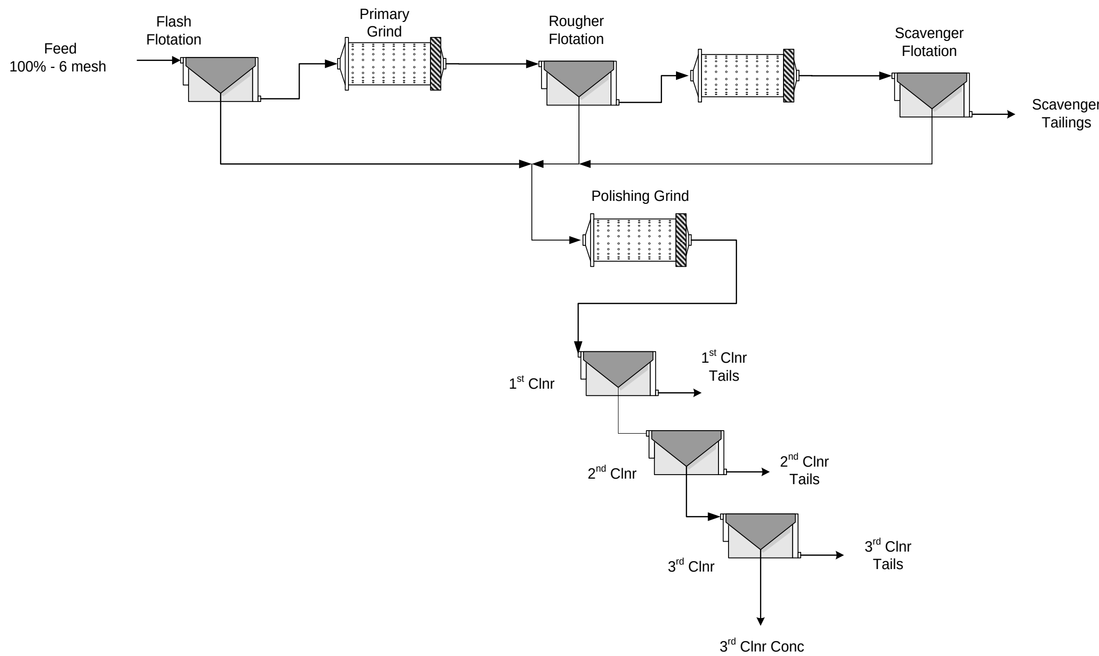
Table 13-1: Head Assays for 2013 Coosa County Metallurgical Samples

Sample ID	C(t) %	C(g) %	TOC LECO %	CO_3 %	S %
AGC-011C-190-220	3.43	3.53	<0.05	1.55	2.09
AGC-H045 0-34	3.74	3.71	0.17	0.65	0.04
AGC-G04C 240-270	2.95	2.92	<0.05	2.90	1.75

13.2.2 Cleaner Flotation Testing

A single batch cleaner flotation test was carried out on each sample. The test conditions were determined by the metallurgist based on observations made during the tests. The overall flowsheet was kept almost the same in all tests and is depicted in Figure 13-1. In only one test the primary grind was eliminated as the graphite flakes in the flash flotation tailings occurred as fine particles locked with quartz. Consequently, a short primary grind would not have achieved the necessary mineral liberation.

Figure 13-1: Open Circuit Flowsheet for April 2013 Coosa County Samples



The open circuit metallurgical performance of the three tests is presented in Figure 13-2 in the form of total carbon grade versus total carbon recovery. The two samples from greater depth (AGC-011C 190-220 and AGC-G-04c 240-270) outperformed the shallow sample (AGC-H045 0-34) in terms of carbon recovery. The two deeper samples achieved open circuit carbon recoveries of 93.5% and 97.7% compared to only 80.2% for the shallow sample. However, the shallow sample produced the highest concentrate grade of 91.1% total carbon compared to 80.8% and 88.3% total carbon for the deeper samples. It is postulated that the liberation properties were improved for the shallow sample due to weathering.

It should be noted that these tests were carried out as scoping level tests only and AGP does not consider the results to be representative of the metallurgical response that might be achieved with a properly designed flowsheet and conditions. The primary objective of these initial scoping level tests was to evaluate the amenability of the Coosa County graphite mineralization to standard graphite processing technology.

A secondary objective of this work was to develop an understanding of the flake size distribution in a flotation concentrate. The mass recovery into different size fractions of the flotation concentrate is depicted in Figure 13-3. The results suggest that the flake size distribution of the shallow sample was significantly finer compared to the other two samples.

Figure 13-2: Total Carbon Grade vs. Total Carbon Recovery - 2013 Coosa County Samples

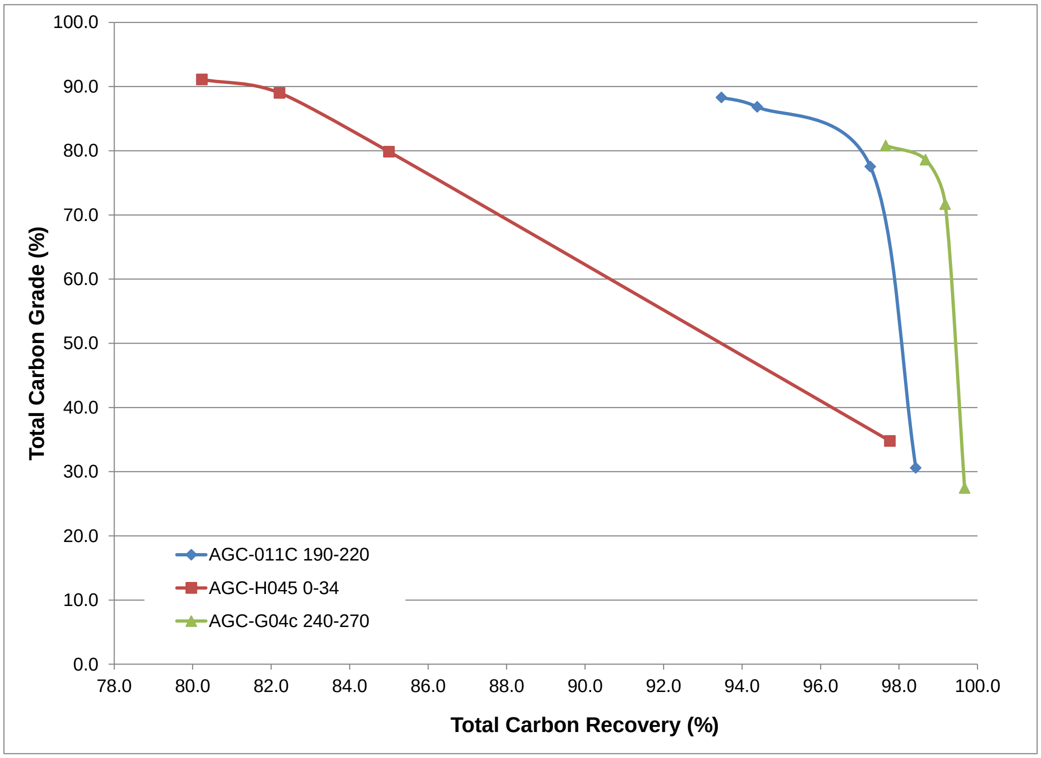
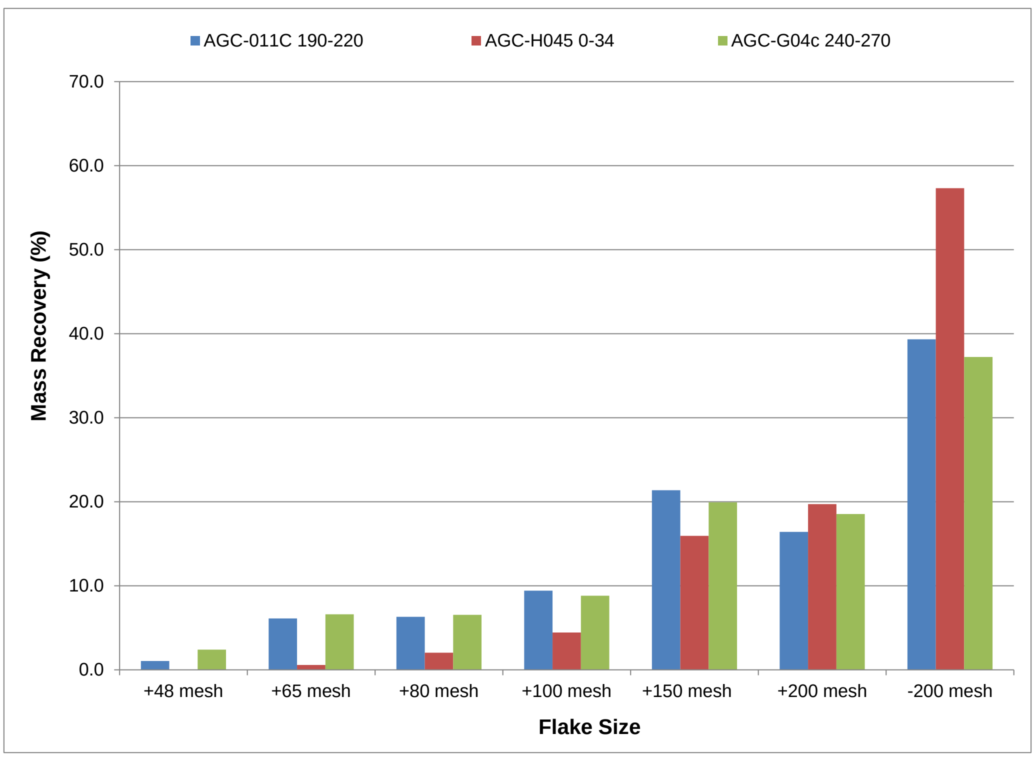


Figure 13-3: Mass Recovery into Different Flake Sizes - 2013 Coosa County Samples



13.3 2014 Metallurgical Testwork

A second scoping level metallurgical test program was completed at SGS Lakefield on four samples originating from the Coosa County graphite mineralization in Alabama, USA. The primary objective of the test program was to evaluate the metallurgical response of the four graphite samples to support the identification of the most promising mineral zone for flowsheet development purposes.

13.3.1 Sample Preparation and Head Characterisation

Four Coosa county samples, weighing between 1.2 kg and 3.7 kg each, were received at the SGS Lakefield site in April 2014. Each sample was stage-crushed to -6 mesh and split into 2 kg and 1 kg test charges. A representative head sample of the four samples was submitted for chemical analysis and the pertinent results are presented in Table 13-2:. The head grades of the four samples ranged between 1.79% and 4.42% graphitic carbon.

Table 13-2: Carbon Speciation and Sulphur Analysis of Coosa County Samples

Sample ID	Assays, %				
	C(t)	C(g)	TOC	CO_2	S
C5-155 A-k	1.86	1.79	0.12	< 0.05	0.02
AGC-10c 70-80	4.67	4.42	0.46	1.28	2.10
BAMA BLK #1 Upper & Lower	2.54	2.41	0.59	0.29	0.02
Hole E-09 85-90, Hole 10 330-335, Sample 132962	3.30	3.21	0.34	0.26	2.25
J03 0-30	3.05	2.87	0.47	2.75	0.02

13.3.2 Batch Cleaner Flotation

In order to preserve the graphite flakes, a two-stage grinding approach was chosen for the flotation. One batch cleaner flotation test was carried out on each of the four samples. The conditions of the tests such as flotation times, grinding and polishing times, and reagent dosages were established during the tests by evaluating the intermediate concentrate and tailings products in a binocular microscope. Due to the significant amount of liberated entrained gangue minerals for the C5-155-A-K sample, this test included a pre-cleaner prior to polishing and three stages of cleaning of the polishing mill discharge. The remaining three tests subjected the combined flash and rougher concentrate to polishing grinding followed by four stages of cleaning.

The carbon grade versus carbon recovery curves of the four tests are depicted in Figure 13-4. While three samples produced somewhat comparable grade-recovery curves, sample C5-155-A-K yielded the lowest carbon recovery of 51.8% into the graphite concentrate. While this number is expected to increase in closed circuit operation, the difference in metallurgical response compared to the other samples is notable. These samples produced concentrates grading between 83.7% total carbon and 90.0% total carbon. While the AGC-10c 70-80 sample yielded the highest head grade of 4.42% graphitic carbon, it produced the lowest 4th cleaner concentrate grade of 83.7% total carbon.

The carbon grade and mass recoveries into the different size fractions in the final concentrates are presented in Figure 13-5 and Figure 13-6, respectively.

The samples consisting of Hole E-09 85-90, Hole 10 330-335, and 132962 and the J-03 0-30 sample were the best performing samples when considering both the flake size distribution and concentrate grades in the various size fractions. The flake size distribution of the C5-155-A-K sample was fine with a P80 of only 106 microns. Only 1.6% of the concentrate mass reported to the plus 80 mesh products compared to 12.3% to 28.4% for the other three samples.

The C5-155 A-K and J03 0-30 samples were characterized by a very low sulphur grade of 0.02% S, which almost certainly will render the tailings streams of these three samples non-acid generating. However, this would have to be verified with environmental tests such as acid base accounting (ABA) or net acid generation (NAG) tests. The J03 0-20 sample also contained carbonates that would neutralize any small quantities of acid that may develop.

Figure 13-4: Carbon Grade versus Carbon Recovery Curves

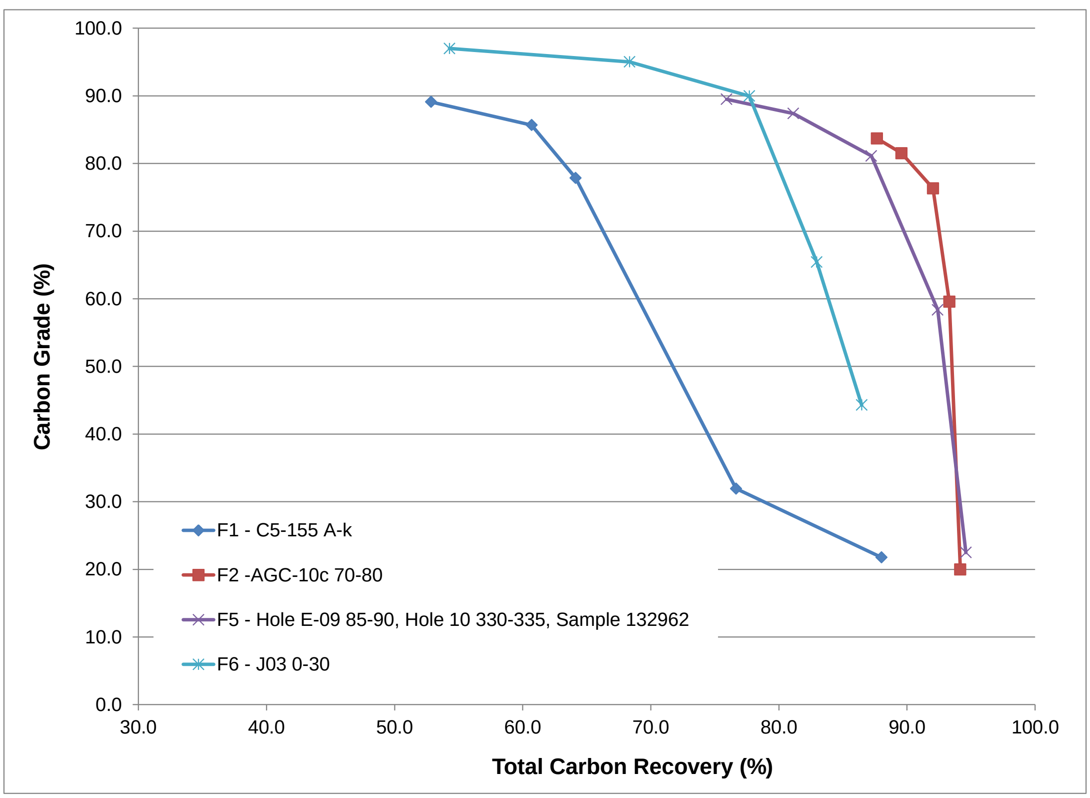


Figure 13-5: Carbon Grades of Concentrate Size Fractions

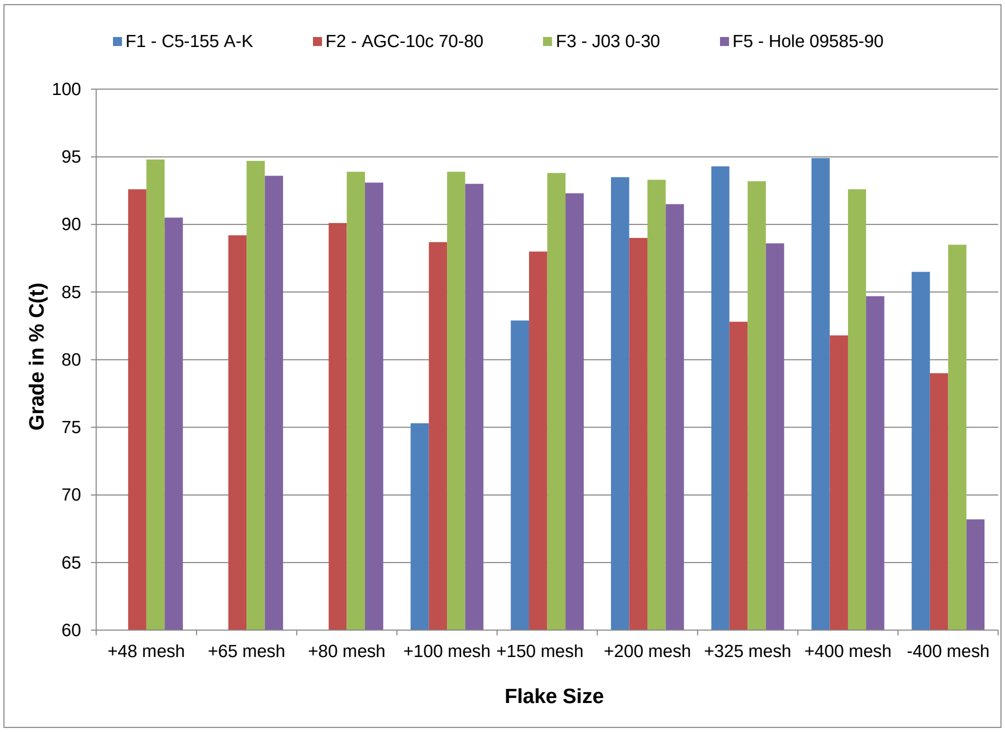
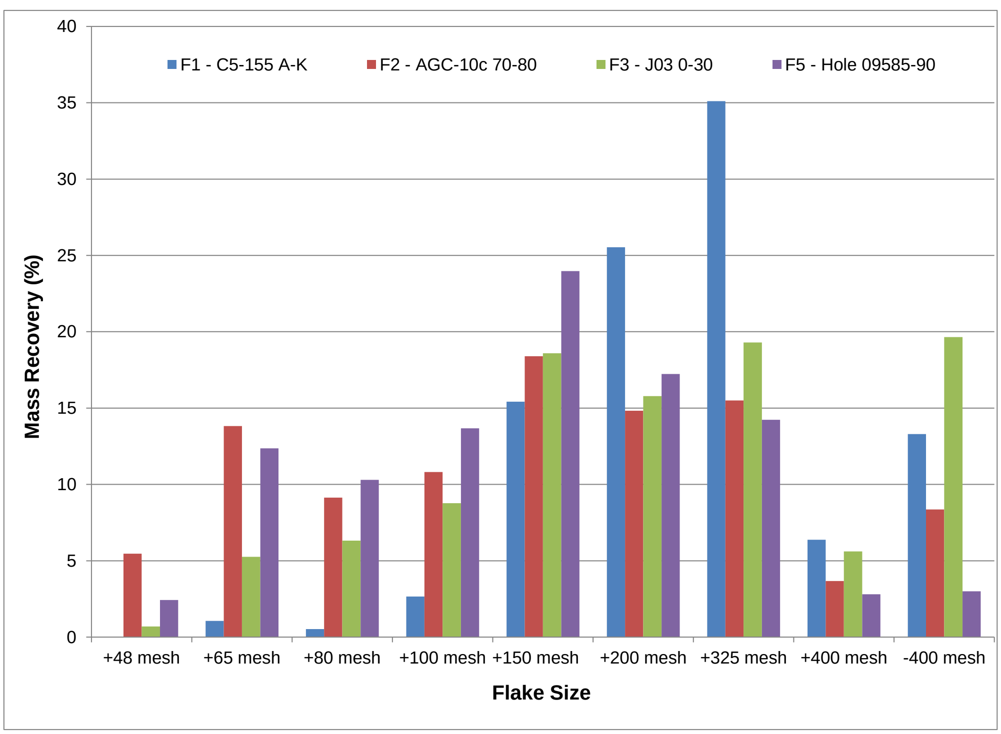


Figure 13-6: Mass Recovery into Concentrate Size Fractions



The following recommendations were made for the next phase of metallurgical testing focusing on the most encouraging zone(s) within the samples tested to-date.

- Basic comminution testing;
- Flowsheet development program to derive at a flowsheet that maximizes concentrate grade and carbon recovery, while minimizing the degradation of flakes, and
- Preliminary environmental testing to identify any major deleterious elements that may be present in the feed material and to quantify the acid generating potential of the Coosa County mineralization.

13.4 2014/15 Metallurgical Testwork

The subsequent metallurgical program was run at SGS Lakefield April 2014 to July 2015 and comprised of scoping level testing and basic comminution testing on two samples.

13.4.1 Sample Preparation and Head Characterisation

All samples were stage-crushed to -6 mesh and rotary split into 2 kg test charges. Each sample was processed separately and no blending of different samples was performed. The results of the head analysis of the eleven samples are presented in Table 13-3: and Table 13-4. Only total carbon and graphitic carbon analysis was carried out for the eight trench samples. The grades of the eleven samples ranged between 2.55% graphitic carbon for the TR05 sample and 4.42% graphitic carbon for the AGC-12A (0-8') sample.

Table 13-3: Carbon Speciation and Sulphur Analysis of Two Coosa County Samples

Sample ID	C(t)	C(g)	TOC	CO_3	S
AGC-12A (0-8')	4.48	4.42	0.18	< 0.05	0.04
AGC-09C (300-325)	3.35	3.35	0.05	< 0.05	1.92

Table 13-4: Total Carbon and Graphitic Carbon Grades for Eight Trench Samples

Sample ID	C(t)	C(g)
	%	%
TR07B	3.11	3.12
TR08	3.50	3.34
TR01A	3.17	3.17
TR19	3.03	2.95
TR05	2.55	2.55
TR03A	3.16	3.05
TR14D	3.85	3.71
TR12	3.21	3.09

13.4.2 Comminution Testing

One sample originating from between 0 and 8 feet below surface and another sample from 100 to 150 feet below surface were subjected to a Bond ball mill grindability test to develop a preliminary understanding of the hardness of the Coosa County mineralization. The results of the two tests are presented in Table 13-5.

Table 13-5: Bond Ball Mill Work Index Results - AGC-12B (0-8) and AGC-10C (100-150) Samples

Sample	Bond Ball Mill Work Index	
	BWI Metric (kWh/t)	BWI Imperial (kWh/t)
AGC-12B (0-8)	5.3	4.8
AGC-10C (100-150)	15.0	13.6

The near surface sample AGC-12B (0-8) yielded a very low Bond ball mill work index (BWI) of 5.3 kWh/t. The sample originating from 100-150 feet below surface produced a substantially higher BWI of 15.0 kWh/t, which is comparable with many base metal ores.

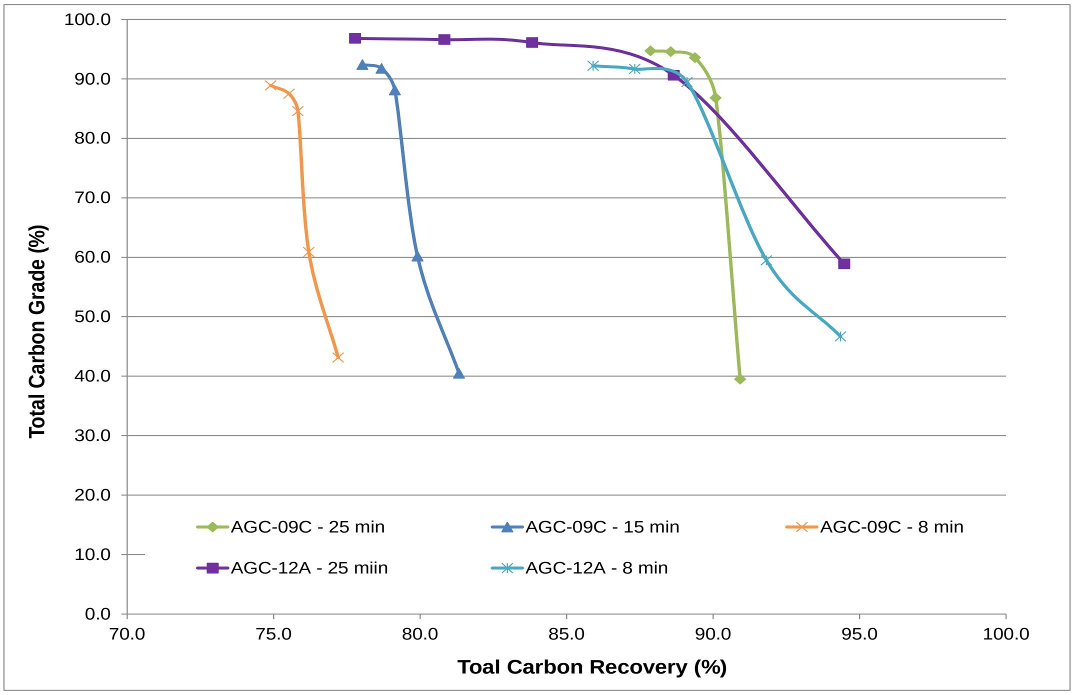
13.4.3 Cleaner Flotation Tests on AGC-12A and AGC-09C Samples

The AGC-09C sample from a depth between 300 and 350 feet and the near surface sample AGC-12A from a depth between 0 and 8 feet were the first samples to be subjected to more than a single scoping level cleaner flotation test. Five tests were completed with the two samples to evaluate the impact of polishing time on the cleaner performance and combined concentrate product quality.

The grinding time in the polishing mill treating the pre-cleaner concentrate was varied between 8 minutes and 25 minutes. All other flotation conditions were kept the same. The total carbon grade versus recovery curves for the five tests are depicted in Figure 13-7.

The low rougher carbon recovery of the AGC-09C sample is believed to be linked to the fact that this material was much harder compared to all other samples tested to-date and the grind time of 3 minutes proved insufficient to achieve a satisfactory liberation prior to rougher flotation. The cleaner stage-recovery for the AGC-09C sample was very high in all three tests, suggesting that the graphite originating from deeper within the deposit displays superior flotation kinetics.

Figure 13-7: Total Carbon Grade versus Recovery Curves - AGC-09A and AGC-12A Samples



The carbon grade and mass recoveries into the different size fractions in the final concentrates are presented in Figure 13-8 and Figure 13-9 respectively.

The impact of polishing time on the flakes size distribution is evidenced in Figure 13-8. As the polishing time is increased, the mass recovery into the coarser flake size category is reduced. The increase from 15 to 25 minutes was most pronounced. Both samples displayed a similar response to the changes in polishing times.

As the polishing time is increased the concentrate grades improved for all size fractions with the exception of the -48/+65 mesh product for the AGC-12A sample. With longer polishing times the probability of removing impurities attached to the graphite flakes increases and, therefore, these results were expected and are consistent with other graphite deposits.

Figure 13-8: Mass Recovery into Size Fractions - AGC-09C and AGC-12A Samples

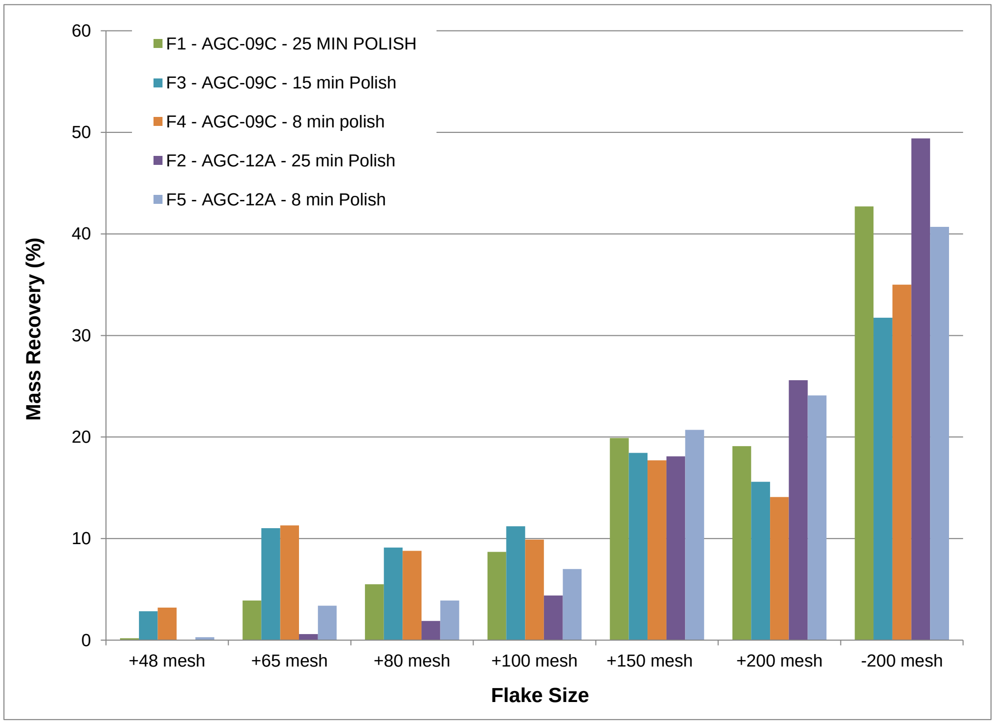
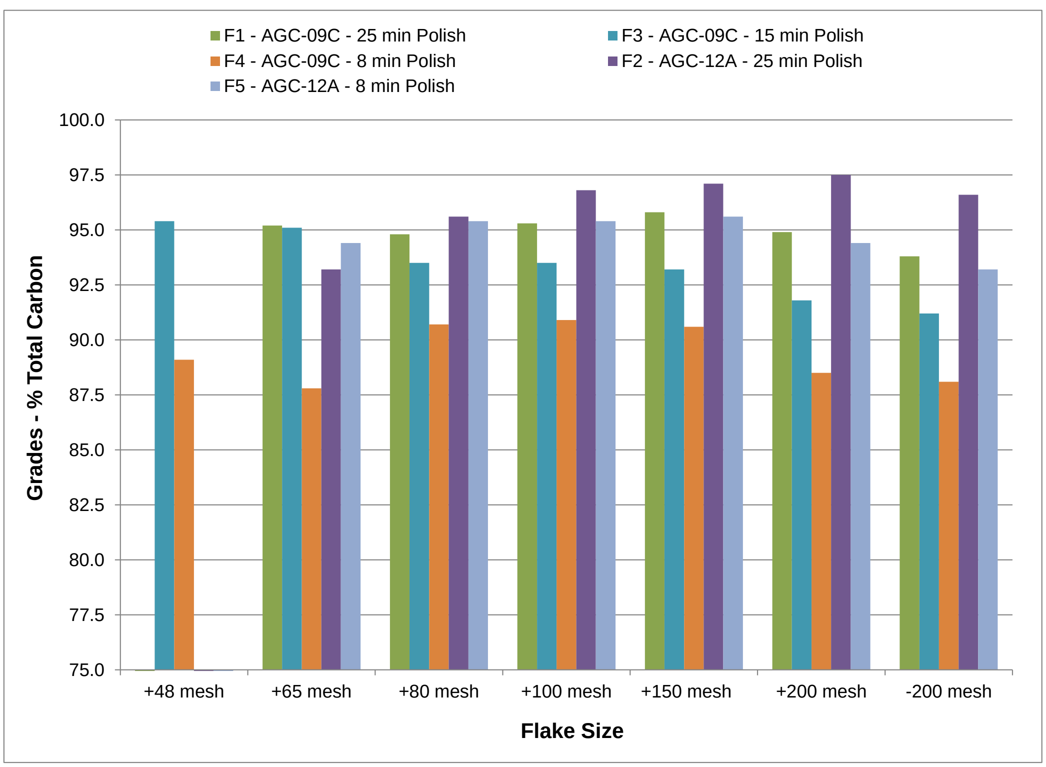


Figure 13-9: Carbon Grades of Concentrate Size Fractions - AGC-09C and AGC-12A Samples



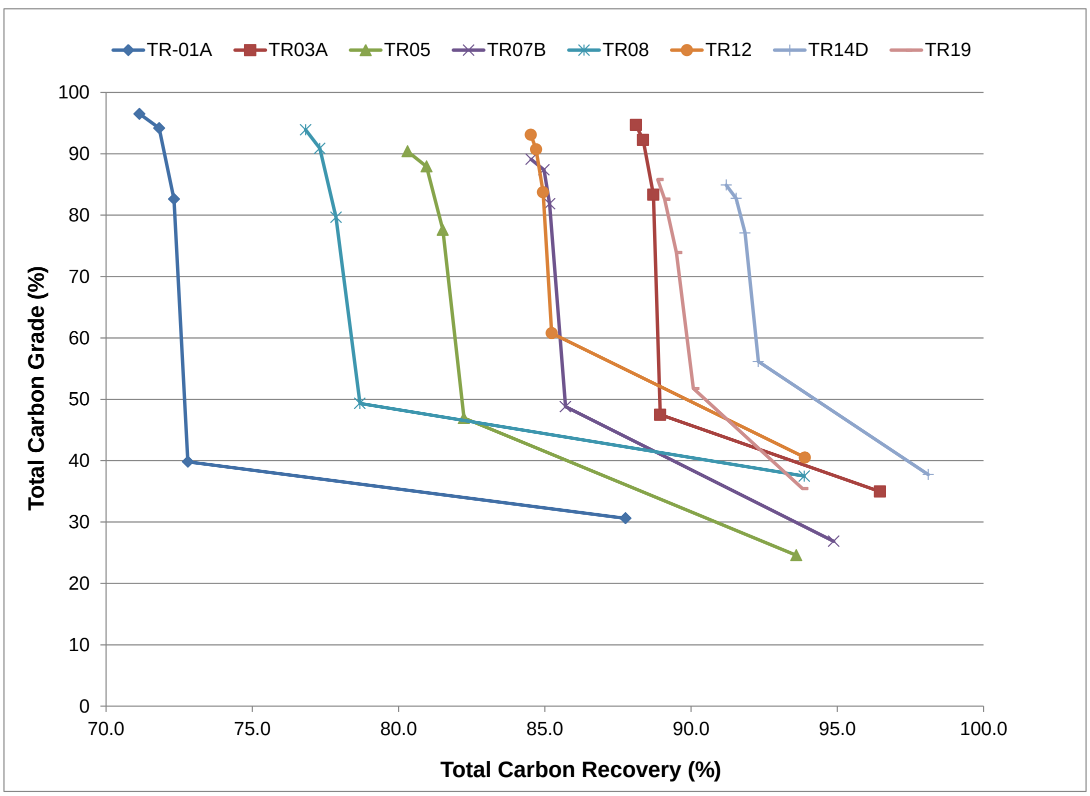
13.4.4 Cleaner Flotation Tests on Nine Trench Samples

Eight trench samples were subjected to batch cleaner tests using the same conditions to facilitate a direct comparison of the metallurgical response of the eight samples. A grind time for the flash tailings of 3 minutes, a pre-cleaner, and a polishing time of 15 minutes were chosen for these tests based on the experience from the previous tests conducted on Coosa County samples.

The total carbon grade recovery curves for the eight tests are depicted in Figure 13-10. The combined flash and rougher carbon recovery ranged between 87.8% for the TR-01A sample and 98.1% for the TR14D sample. The rougher P80 ranged between 195 microns for the TR05 sample and 313 microns for the TR03A sample and did not appear to have an immediate impact in the flash and rougher carbon recovery.

The pre-cleaner losses were much higher in all eight tests compared to previous cleaner tests and ranged between 5.8% for the TR14D sample and 15.2% for the TR08 sample. It was concluded that this was not a function of the samples, but rather the pre-cleaner flotation conditions. It is postulated that these elevated pre-cleaner tailings losses can be reduced substantially with higher reagent dosages and longer flotation times.

Figure 13-10: Total Carbon Grade versus Recovery Curves - Trench Samples



The carbon grade and mass recoveries into the different size fractions in the final concentrates are presented in Figure 13-11 and Figure 13-12 respectively. The flake size distribution varied noticeably for of the eight samples. The mass recovery into the plus 80 mesh size fractions ranged between 9.2% for the TR08 sample and 33.9% for the TR19 sample.

With regards to concentrate grades the TR03A sample produced the best results with a combined concentrate grade of 94.1% total carbon using the reconciled size fraction analysis results. The TR01, TR14D and TR19 samples yielded the lowest combined concentrate grades between 83.2% and 86.8% total carbon in the size fraction analysis.

Figure 13-11: Mass Recovery into Size Fractions - Trench Samples

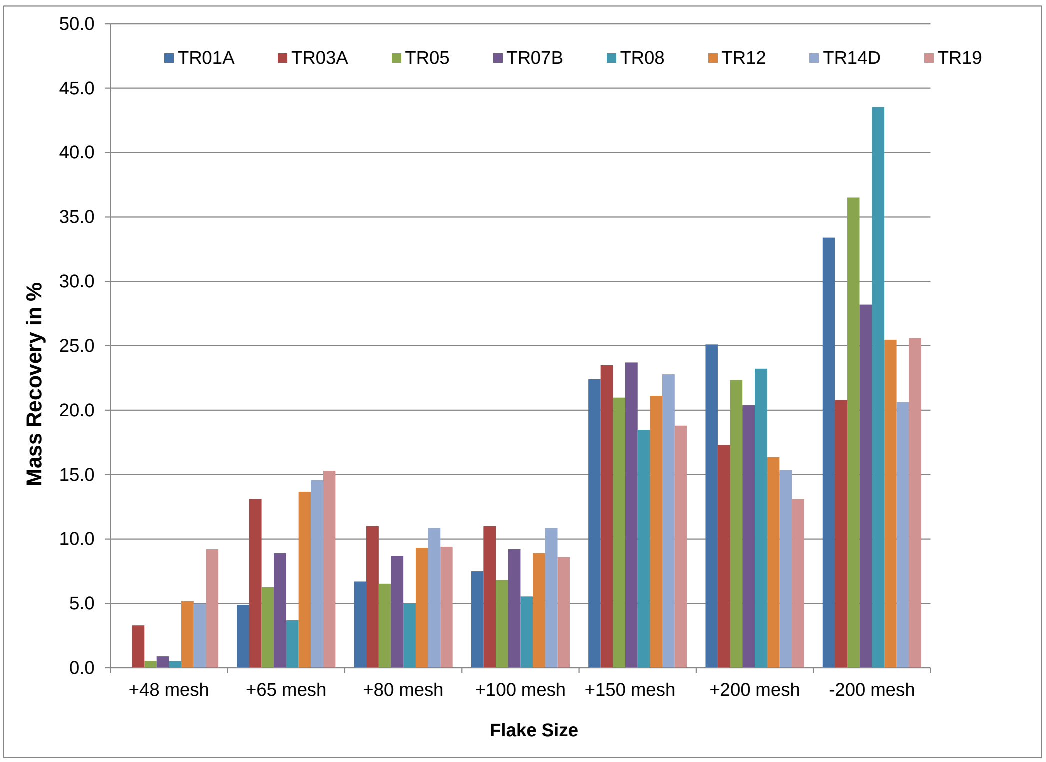
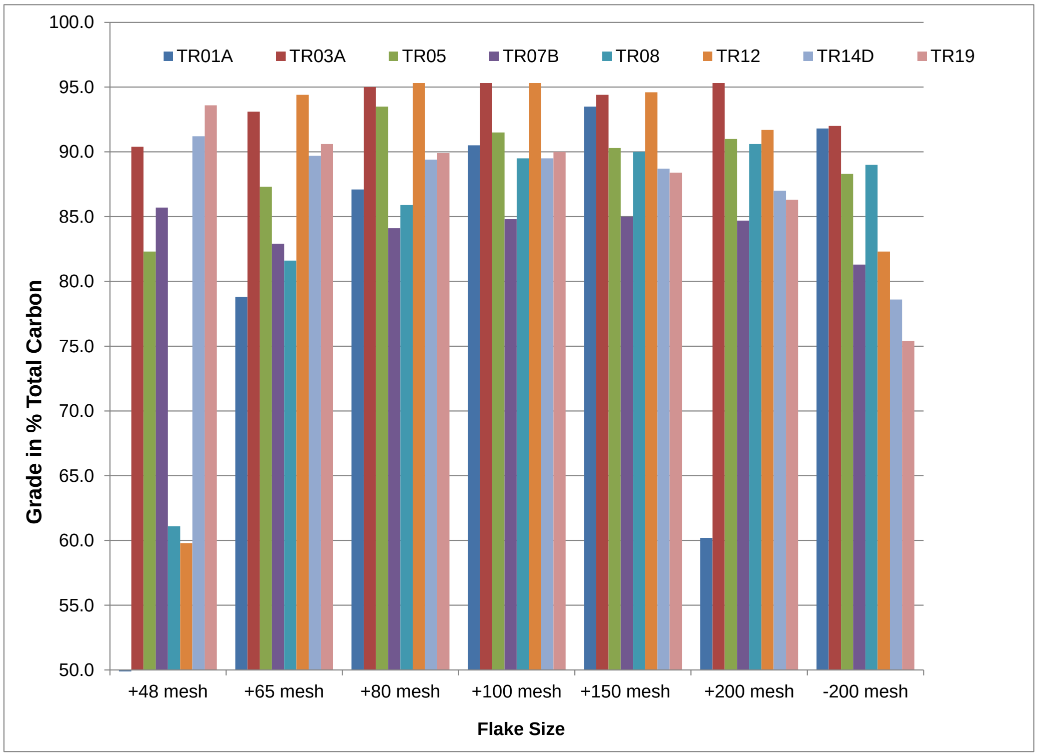


Figure 13-12: Carbon Grades of Concentrate Size Fractions - Trench Samples



13.5 2105 Metallurgical Testwork

A full flowsheet development program was initiated by Alabama Graphite at the SGS Lakefield site in August 2015 and the program was completed within one month.

13.5.1 Sample Preparation and Head Characterisation

Six samples from the Coosa County mineralization were shipped to SGS Lakefield in August 2015. The six samples comprised of two samples each from the Coosa, Roscoe Ridge, and Holy Schist zones within the Coosa County mineralization.

Each sample was stage crushed to -6 mesh and homogenized before sub-samples were extracted for chemical analysis and a Master composite. The Master composite was generated by extracting 10 kg from each of the individual samples. The six samples and one composite were split into test charges for flotation testing.

A summary of pertinent chemical analysis results is presented in Table 13-6. The head grades ranged between 2.73% graphitic carbon in the Coosa Transition sample and 3.85% graphitic carbon in the Roscoe Ridge sample. The concentration of sulphur was generally low at 0.04% sulphur or less with the exception of the Coosa Transition sample at 1.46% sulphur, which caused the elevated value in the Master composite.

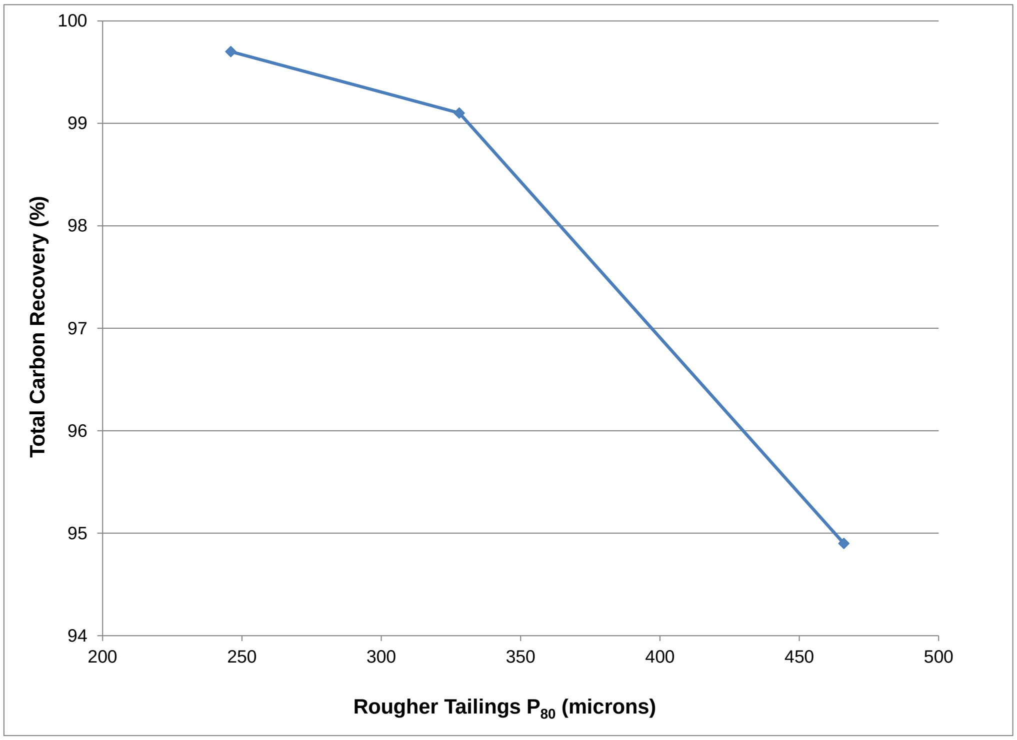
Table 13-6: Carbon Speciation and Sulphur Analysis of Flowsheet Development Samples and Composite

Comp ID	Assays, %				
	C(t)	C(g)	TOC	CO_3	S
Master Comp Aug 15	3.32	3.24	< 0.05	1.00	0.23
Roscoe Ridge #1	3.22	3.09	< 0.05	0.20	0.04
Roscoe Ridge #2	3.85	3.85	< 0.05	1.35	0.02
Coosa Oxide	3.27	3.12	< 0.05	1.05	0.01
Coosa Transition	2.81	2.73	< 0.05	2.65	1.46
Holy Schist #1	3.35	3.22	< 0.05	0.10	0.03
Holy Schist #2	3.32	3.27	< 0.05	0.30	0.02

13.5.2 Rougher Flotation Testing

A total of three rougher kinetics tests were carried out to establish the grind size requirements to achieve a satisfactory graphite recovery. Each test consisted of a flash flotation stage treating the -6 mesh test charges followed by grinding of the flash flotation tailings and rougher flotation on the mill discharge. The grind times were varied between 1.5 minutes in test F1 and 5 minutes in test F3. The results of the three tests are summarized in Figure 13-13 in form of the total carbon recovery into the combined flash and rougher concentrate versus rougher tailings grind size plot. Test F1 with rougher tailings P80 of 328 yielded a very high total carbon recovery of 99.1%, thus suggesting that a grind size of P80 = 325 microns should be targeted.

Figure 13-13: Total Carbon Recovery as a Function of Grind Size



13.5.3 Cleaner Flotation Testing

A total of ten open circuit cleaner tests were carried out on the Master composite with the objective to develop the full cleaning circuit.

The first four cleaner tests evaluated the impact of primary polishing of the combined flash and rougher concentrate using ceramic media. Since some samples in the scoping level test programs produced a significant amount of entrained gangue minerals in the combined flash and rougher concentrate, a pre-cleaner flotation stage was incorporated prior to the polishing mill. The polishing times were varied between 10 minutes in test F5 and 28 minutes in test F7.

The total carbon recovery did not appear to be affected by the polishing times and ranged between 94.7% for a polishing time of 22 minutes and 95.9% for a polishing time of 28 minutes. Although test F4 with polishing time of 15 minutes yielded a recovery into the final concentrate of only 89.4%, the source of the elevated losses was the flash and rougher stage rather than the cleaning circuit. The graphitic carbon losses into the rougher tailings of this test were 7.1% compared to only 2.1 to 2.7% in the other three tests. The reason for the elevated carbon losses is not understood as the flash and rougher conditions were identical in all four tests.

In order to evaluate the impact of the different polishing times on the cleaner performance, the 3rd cleaner concentrates of the four tests were subjected to a size fraction analysis. The mass recovery into the various size fractions and corresponding total carbon grades are depicted in Figure 13-14 and Figure 13-15, respectively. As expected, the flake size distribution gradually got finer as the polishing time was increased. The test with the shortest polishing time was a slight outlier with regards to mass recovery into the size fractions, but fell in line with regards to total carbon grades of the various size fractions.

The total carbon grades of the size fractions improved with longer polishing times. However, all size fractions with the exception of the +48 mesh size fraction of the test using the longest polishing time of 28 minutes failed to reach the minimum grade size target of 95.0% total carbon. These results reveal the necessity of a secondary cleaning circuit in order to achieve acceptable concentrate grades. These secondary cleaning circuits are generally required for graphite ores with the exception of very few deposits. Since the impact of the polishing time on the flake size distribution was relatively small and the improvement of concentrate grades in the finer size fractions was pronounced, a decision was made to proceed with a primary polishing time of 30 minutes in the remaining cleaner tests.

Figure 13-14: Mass Recovery into Size Fractions - Cleaner Tests F4 to F7

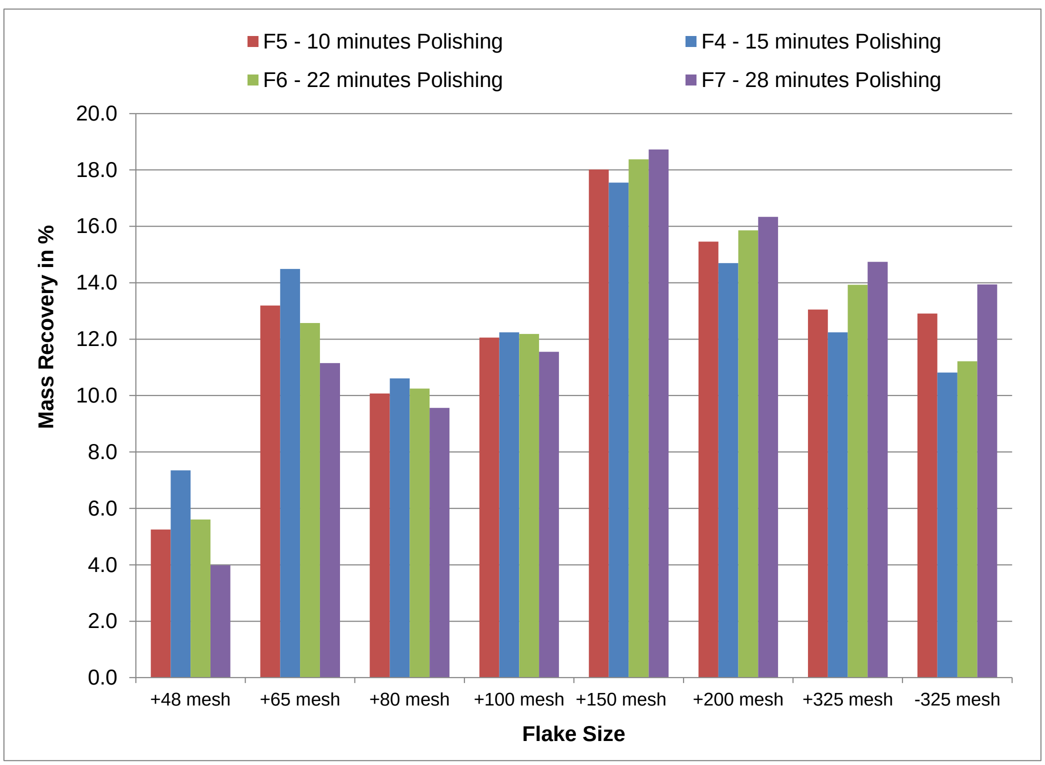
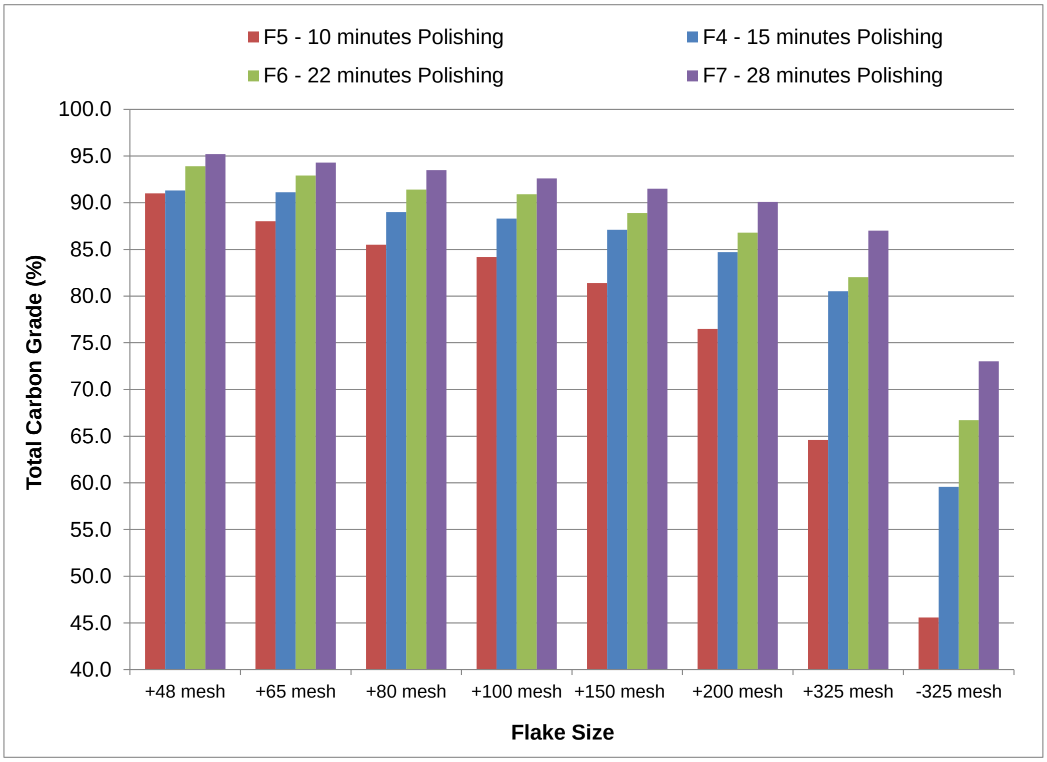


Figure 13-15: Total Carbon Grades of Size Fractions - Cleaner Tests F4 to F7



While the flake sizes larger than 100 mesh (150 microns) yielded grades of 92.6% total carbon in the test with the longest polishing time of 28 minutes, the finer size fractions fell significantly short of the minimum grade target of 95% total carbon. Hence, a decision was made to classify the intermediate concentrate at 100 mesh and process the screen oversize and undersize in separate cleaning circuits. This approach is commonly taken for graphite projects if liberation properties are a function of the flake size. The screen oversize was subjected to conventional polishing grinding with ceramic media, while the screen undersize was also treated with alternative grinding technologies with different liberation mechanisms.

Four batch cleaner flotation tests were conducted to evaluate the best configuration of the secondary cleaning circuit. In addition to the conventional polishing with ceramic media, the minus 100 mesh material was processed in a pebble mill with 6 mm steel media and an attrition scrubber with 2 mm ceramic media.

The two concentrates of each test were submitted for a size fraction analysis. The mass recovery into the various size fractions and corresponding total carbon grades are depicted in Figure 13-16 and Figure 13-17, respectively. The mass recovery into the plus 100 mesh size fractions only changed marginally as the polishing time was increased from 10 minutes in test F8 to 25 minutes in test F11. Since the differences in mass recovery into the four size fractions were small and did not reveal a trend, it is concluded that they are mainly test-to-test variances and that the polishing time did not have an impact on the mass recovery into the coarser size fractions. The average grade improvement as a result of the secondary cleaning circuit for the plus 100 mesh material was 1.5% from 93.9% total carbon to 95.4% total carbon.

The concentrate grades of the plus 100 mesh size fractions trended to lower grades as the polishing time was increased. While this is in contrast to expectations, it is postulated that some of the impurities that were initially liberated from the flakes were activated by graphite smearing during the longer polishing times. Hence, a short polishing grind of 10 minutes was established for the secondary cleaning circuit processing the plus 100 mesh intermediate concentrate.

With regards to the cleaning circuit for the minus 100 mesh product, only the test using the attrition scrubber achieved the minimum acceptable concentrate grade. In this test the minus 100 mesh product was upgraded by 10.5% from 86.2% total carbon to 96.7% total carbon. The other three tests employing ceramic media in a polishing mill and steel media in a pin mill produced concentrate grades ranging between 92.5% and 93.4% total carbon. Despite the significantly higher concentrate grade obtained in the test with the attrition scrubber, the flake size distribution was the coarsest for this test with a P80 of the combined minus 100 mesh concentrate of 109 microns compared to 103-105 microns for the other three tests. Although differences in size distribution were relatively small, they are evidenced in Figure 13-16. Consequently, the attrition scrubber was chosen as the grinding technology for the secondary cleaning circuit.

Figure 13-16: Mass Recovery into Size Fractions - Cleaner Tests F8 to F11

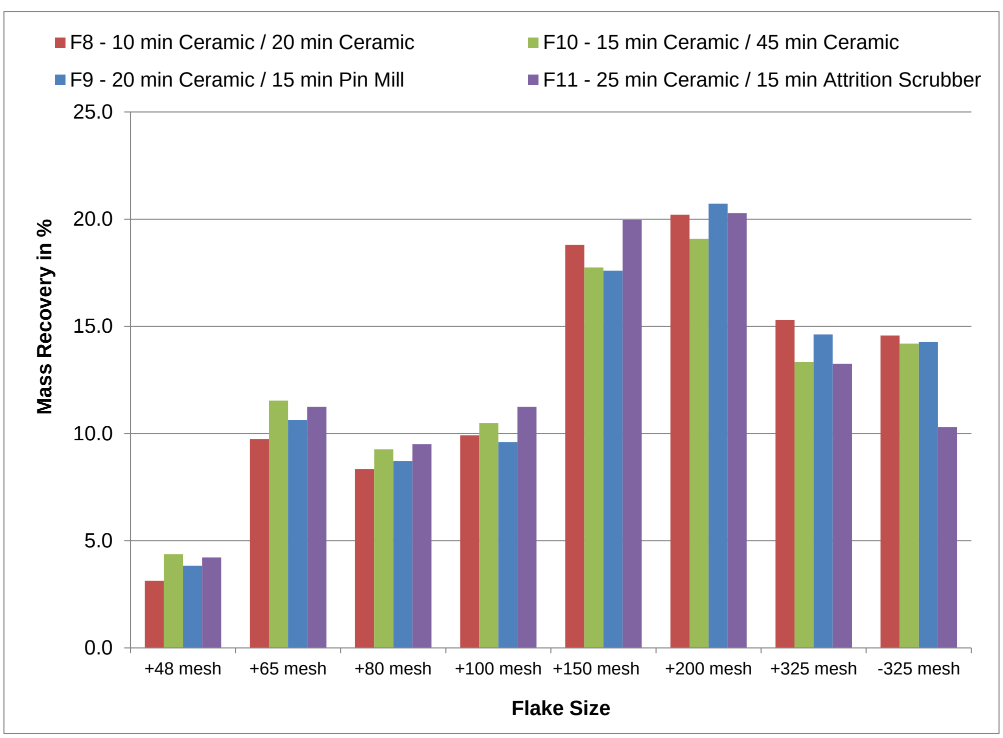
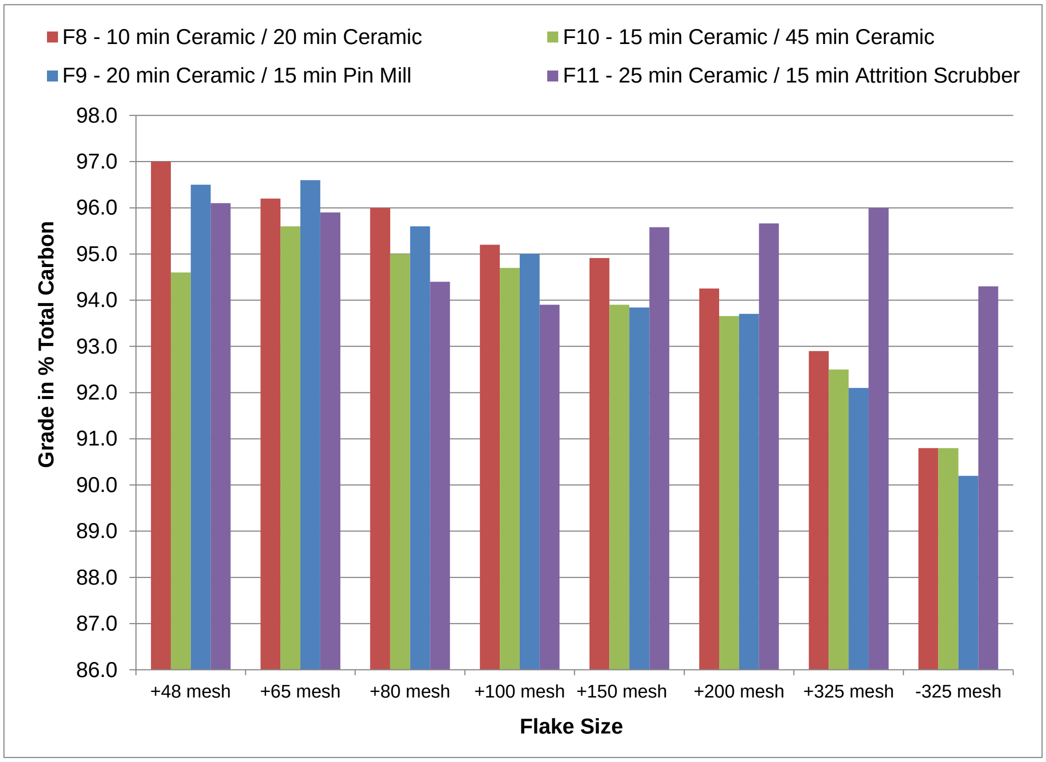


Figure 13-17: Total Carbon Grades of Size Fractions - Cleaner Tests F8 to F11



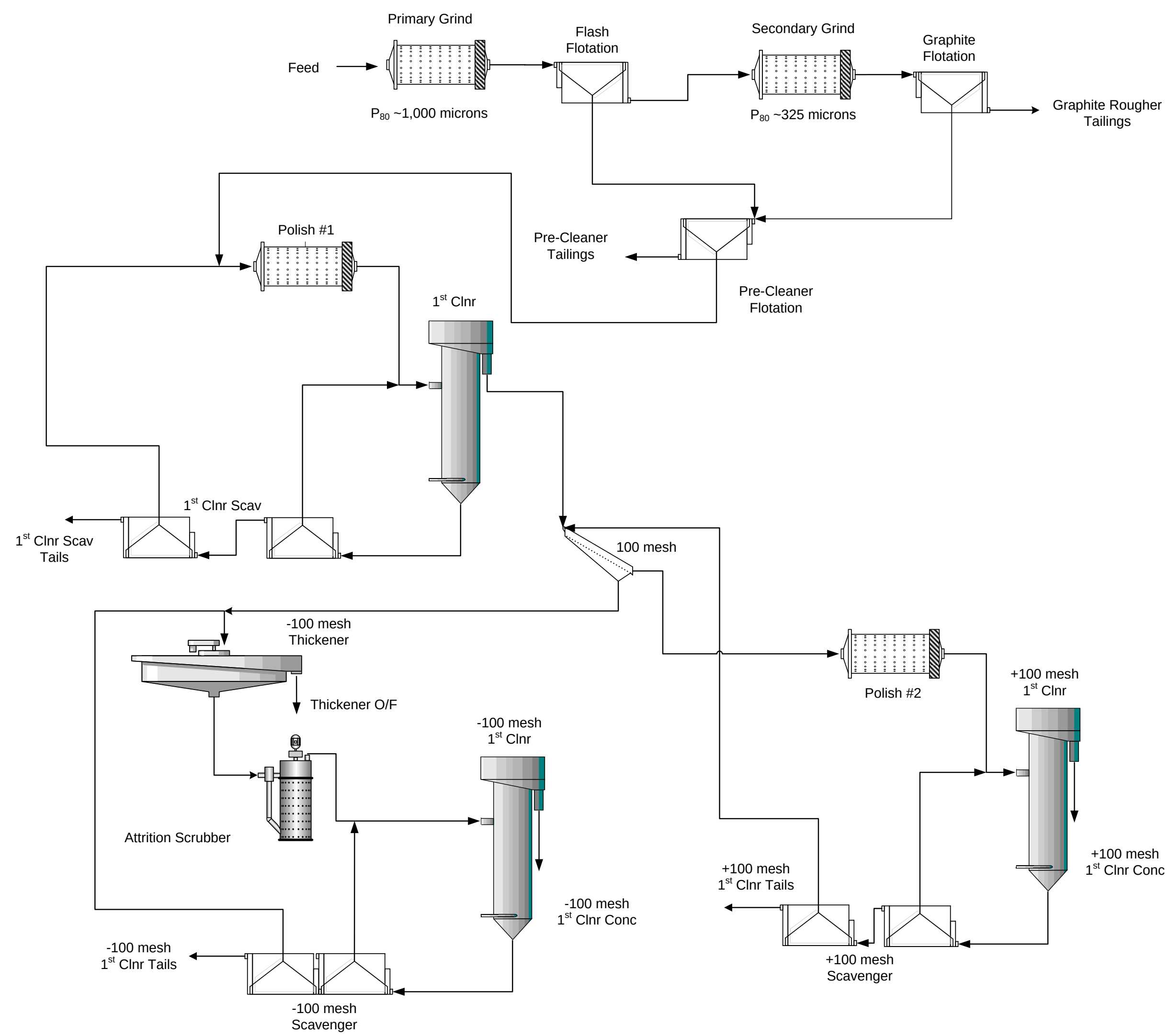
The last two cleaner tests employed the optimized conditions based on the results of the cleaner tests F4 to F11. The difference between the two tests was the elimination of the pre-cleaner in test F12 and inclusion in test F12A. The average combined flash and rougher concentrate grade of the two tests yielded 26.3% total carbon at 12.2% mass recovery. The pre-cleaner stage in test F12A increased the intermediate concentrate grade to 39.9% total carbon and reduced the mass recovery to 7.8%. The carbon losses to the pre-cleaner tailings were 0.7% (0.6% to 1.0% in all eight batch cleaner tests with pre-cleaner stage) and these losses typically occur as very small graphite particles locked with gangue minerals. A visual inspection of the pre-cleaner tails did not identify any liberated and larger graphite flakes.

The combined concentrate grade using the average values between direct and reconstituted grades using the size fraction analysis results yielded 97.3% total carbon in both tests. Test F12 without the pre-cleaner stage produced a slightly higher carbon recovery of 94.2% compared to 91.9% in the test with pre-cleaning. However, the majority of these higher losses were linked to the minus 100 mesh cleaning circuit rather than the pre-cleaning stage, which only contributed with 0.7% to the increased losses.

Based on these results, a decision was made to include the pre-cleaner flotation stage in the proposed flowsheet as earlier scoping level tests included samples displaying a gangue entrainment problem that could be mitigated with the incorporation of a pre-cleaner flotation stage. While gangue entrainment was not identified as a major concern for the flowsheet development composite, including the pre-cleaner stage provides a more robust flowsheet. Further, the mass recovery into the primary polishing mill was reduced significantly by 37%, thus resulting in capital and operating cost savings.

The flowsheet development program culminated in the flowsheet that is depicted in Figure 13-18.

Figure 13-18: Proposed Coosa County Flowsheet



13.6 Concentrate Purification Testwork

A program of preliminary testwork at a North American commercial testwork facility was completed in 2015 to examine the amenability of Coosa Project flotation concentrates to a thermal purification process. For reasons of commercial confidentiality, Alabama Graphite

Corp. does not wish to disclose the identity of the metallurgical laboratory contracted for the Company's ongoing graphite purification work.

A series of purification tests were carried out on several flotation concentrates from the 2014 SGS flotation program to produce high purity graphite (> 99.95%). These samples of graphite concentrate were considered representative of a range of Coosa Project concentrate materials. The graphite purity also ranged from low to high in these samples.

Graphite can be purified by a number of metallurgical techniques typically involving either thermal upgrading or chemical upgrading – the latter can involve a caustic roast process, chlorination or acid leaching. In the 2015 testwork, samples of Coosa flotation concentrate were subjected to purification by chlorination. The carbo-chlorination process involves reacting the graphite concentrate with chlorine gas at elevated temperatures to produce gaseous silicon tetrachloride ($SiCl_4$) which is subsequently swept away. It is well known that silicon tetrachloride (boiling point 57^{0}C) can be produced by reacting silicon or silicon dioxide (also called silica) with carbon at elevated temperatures in a stream of chlorine gas, based on the following chemical reactions:

$$Si + 2CL_2 = SiCl_4 \qquad (1)$$

$$SiO_2 + 2Cl_2 + 2C = SiCl_4 + 2CO \qquad (2)$$

The main purification trials were conducted on a range of samples of Alabama concentrate produced by SGS Mineral Services of Lakefield, Ontario, Canada as presented in Table 13-7

Table 13-7: Range of Alabama Graphite Concentrates used for Purification Tests

Concentrate	Graphite purity, %
Bama	85.634
Coosa County	89.042
Mixed Clean (Coosa plus Bama)	96.505

The purity of the graphite concentrate was determined by conducting a 48-hour loss-on-ignition test at 1000^{0}C. It was assumed here that the main impurity was SiO_2, however, it is noted that other gangue impurity oxides that may be present such as CaO or Al_2O_3 also chlorinate. While a number of reactor configurations could have been used for the laboratory tests, for convenience, a laboratory-scale packed bed reactor common in chemical and metallurgical processing was adopted. This consisted of a cylindrical graphite crucible to contain the graphite concentrate charge and fitted with a tuyere hole in the bottom for chlorine gas injection. The reactor was also fitted with a graphite lid, with the upper section of the crucible walls arranged with several holes to allow gaseous products of the reaction to escape. The graphite crucible used for the tests with the larger charge noted below was sized 64 mm in diameter.

For the tests, the reactor was filled with a pre-weighed quantity of graphite concentrate and then supported in a vertical laboratory tube furnace which was operated at temperatures ranging from 1350^{0}C to 1400^{0}C. In preliminary tests, a charge size of 50 grams was used, followed by tests with a larger charge - from 260 gm to 412 gm. Computations carried out using the FACTSAGE^{TM} thermochemical package showed that Equation (2) above goes to completion at these temperatures and at 1 atmosphere pressure, with all SiO_2 present reacting to produce gaseous $SiCl_4$.

In the laboratory tests, it was found that an excess of chlorine was required to complete the reaction, in part believed to be due to the small size of the laboratory tests and the possible less-than-perfect gas-solid contacting with the relatively small chlorine gas flowrate (for the 64 mm diameter graphite crucible which as noted above was used for the tests with 260 to 415 gm of graphite concentrate charge, a chlorine flowrate of approximately 140 mL/min was adopted).

The time used for each test depended in part on the purity of the starting graphite concentrate (thus influencing the quantity of chlorine required), while in each case, the Stoichiometric Ratio, [Chlorine injected/Stoichiometric chlorine] was typically 5 or 6.

The results of the laboratory testing with the larger charges noted above are presented in Table 13-8. This shows that a graphite purity of over 99.95% was achieved in the laboratory tests. It is noted that the preliminary tests with the 50 gm charge reported similar high impurity levels, ranging from 99.735% to 99.996%.

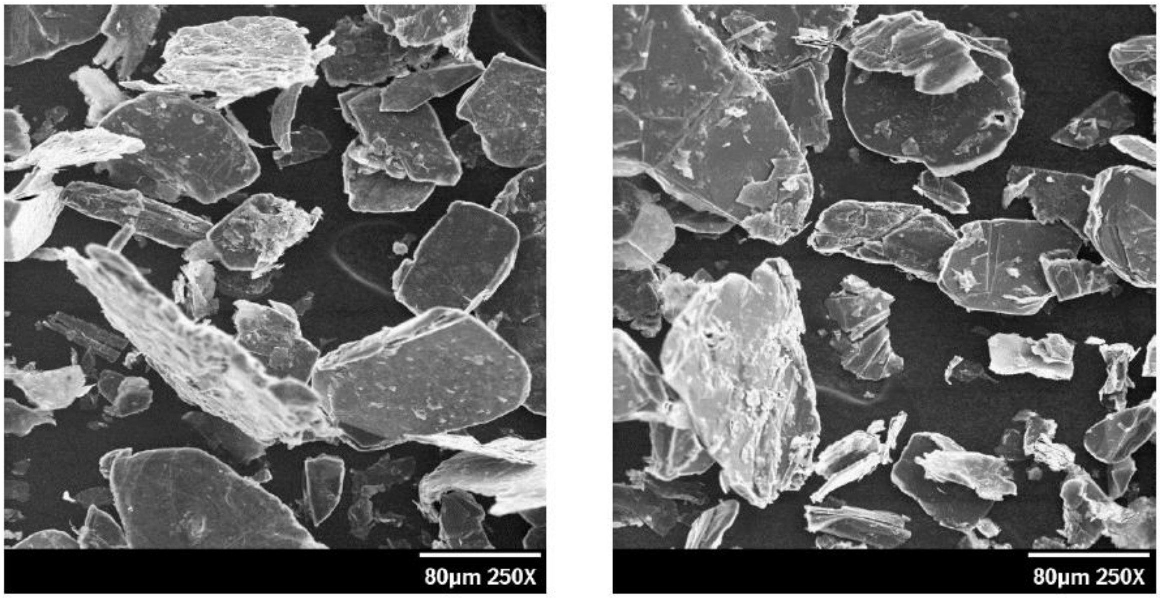
Table 13-8: Results of Laboratory Purification Testing on Samples of Alabama Graphite Concentrate

Concentrate	Graphite purity, %	Test No.	Charge size, gm	Chlorine Injected (Stoichiometric ratio)	Run time, mins	Purity of graphite product, %
Mixed Clean	96.507	5	412	6	450	99.988
Large Coosa	89.042	6	260	6	930	99.971

An SEM analysis was carried out on the starting graphite concentrate and the also on the purified product. Thus was done to check whether the purification process had any effect on the morphology of the graphite flakes. These measurements showed that there was no difference between the non-purified graphite concentrate and the purified concentrate product, refer to Figure 13-19.

This work demonstrates the capability of the proposed purification process to produce high purity graphite flakes (>99.95%) and provides the basis for the development of preliminary cost estimates for this stage of the processing facility.

Figure 13-19: SEM Images of the Non-Purified Graphite Concentrate (left) and the Purified Product (right)



13.7 Performance & Recovery Estimates

13.7.1 Flotation

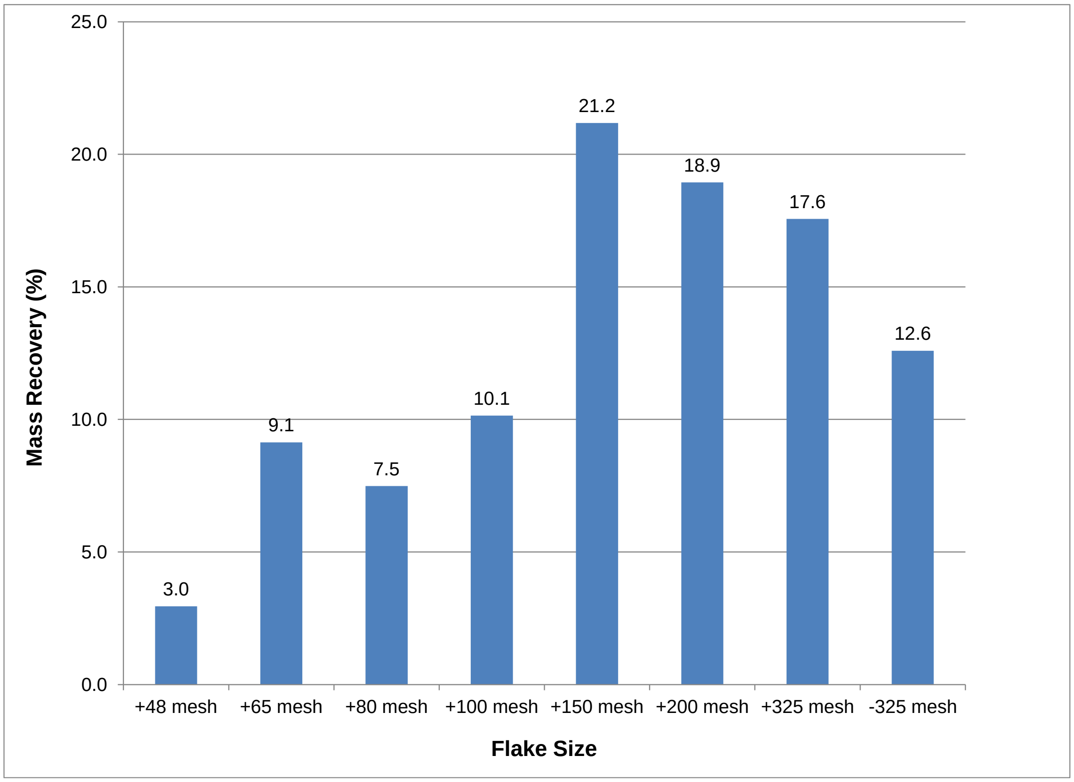
The proposed flowsheet for the processing plant consists of a conventional two stage milling circuit at P_{80} of 1,000 microns in the flash flotation stage and 325 microns in the graphite rougher circuit. The process will generate a flotation concentrate that will then be subjected to purification in a hydrometallurgical process.

It is projected that Coosa County mineralization will produce a flotation concentrate grading 95% total carbon at 92% recovery. This projection is based primarily on the test results of the August 2015 flowsheet development program as previous programs were preliminary in nature.

The average grade of the last six cleaner tests in the August 2015 program was 96.0% total carbon with an open circuit carbon recovery of 93.3%. However, given the fact that these results were obtained using a master composite consisting of a limited number of small samples from different areas within the Coosa County prospect, AGP has elected to use more conservative projections for the PEA. Further no locked cycle tests have been completed today that simulate closed circuit performance.

The projected flake size distribution is depicted in Figure 13-20 and is based on the average values of the last batch cleaner tests of the flowsheet development program using the optimized flotation conditions.

Figure 13-20: Projected Flake Size Distribution



13.7.2 Purification

The prediction of purification process performance is based on the results of preliminary test work conducted in 2015. PEA financial models are based on the following key performance projections.

- The recovery of graphite through the purification process is estimated to be 90%
- Initial results have shown that graphite concentrates of 99.99% Cg can be achieved. For the PEA, AGP has assumed a 99.90% Cg product grade.
- Approximately 70% of the purification plant product will have suitable shape and grade to be considered for sale into the various high value markets (e.g. lithium ion batteries). The remaining 30% will be offered for sale into the micronized market and will thus attract a lower price.

14 MINERAL RESOURCE ESTIMATES

This Mineral Resource statement describes the mineral resource evaluation prepared for the Coosa Project in accordance with the Canadian Securities Administrators' National Instrument 43-101.

The estimate is based on 20,414.4 feet of drilling (both sonic and core) from 69 exploration holes. The resource estimation work was completed by Scott Wilson, CPG, (AIPG Membership Number 10965) an appropriate "independent qualified person" as this term is defined in National Instrument 43-101.

This section describes the resource estimation methodology and summarizes the key assumptions considered by MMC. In the opinion of MMC, the resource evaluation reported herein is a reasonable representation of the global carbon mineral resources found in the Coosa Project at the current level of sampling. The mineral resources have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

The database used to estimate the Coosa Project mineral resources was audited by MMC. MMC is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries of mineralization and that the assay data are sufficiently reliable to support mineral estimation.

Vulcan Software Version 9.1 was used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block models, estimate carbon grades and tabulate mineral resources.

Grades were estimated using Inverse Distance estimation techniques. These resources are a reasonable representation of the graphite found at the Coosa Project given the current level of sampling.

The Coosa Mineral Resources are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, political or other relevant issues. The estimates of Mineral Resources and Mineral Reserves may be materially affected if mining, metallurgical, or infrastructure factors change from those currently anticipated at the Coosa Project.

14.1 Coosa Estimation Procedures

The modeling for Coosa Project was undertaken using Vulcan Software. All exploration sampling has been used in the geological modeling process.

The drill hole database was de-surveyed, transformed and validated in the Vulcan software, which was then used for the Mineral Resource modeling. The statistics have been completed using a combination of Vulcan and Microsoft Excel. Compilation of the final model was undertaken in Vulcan.

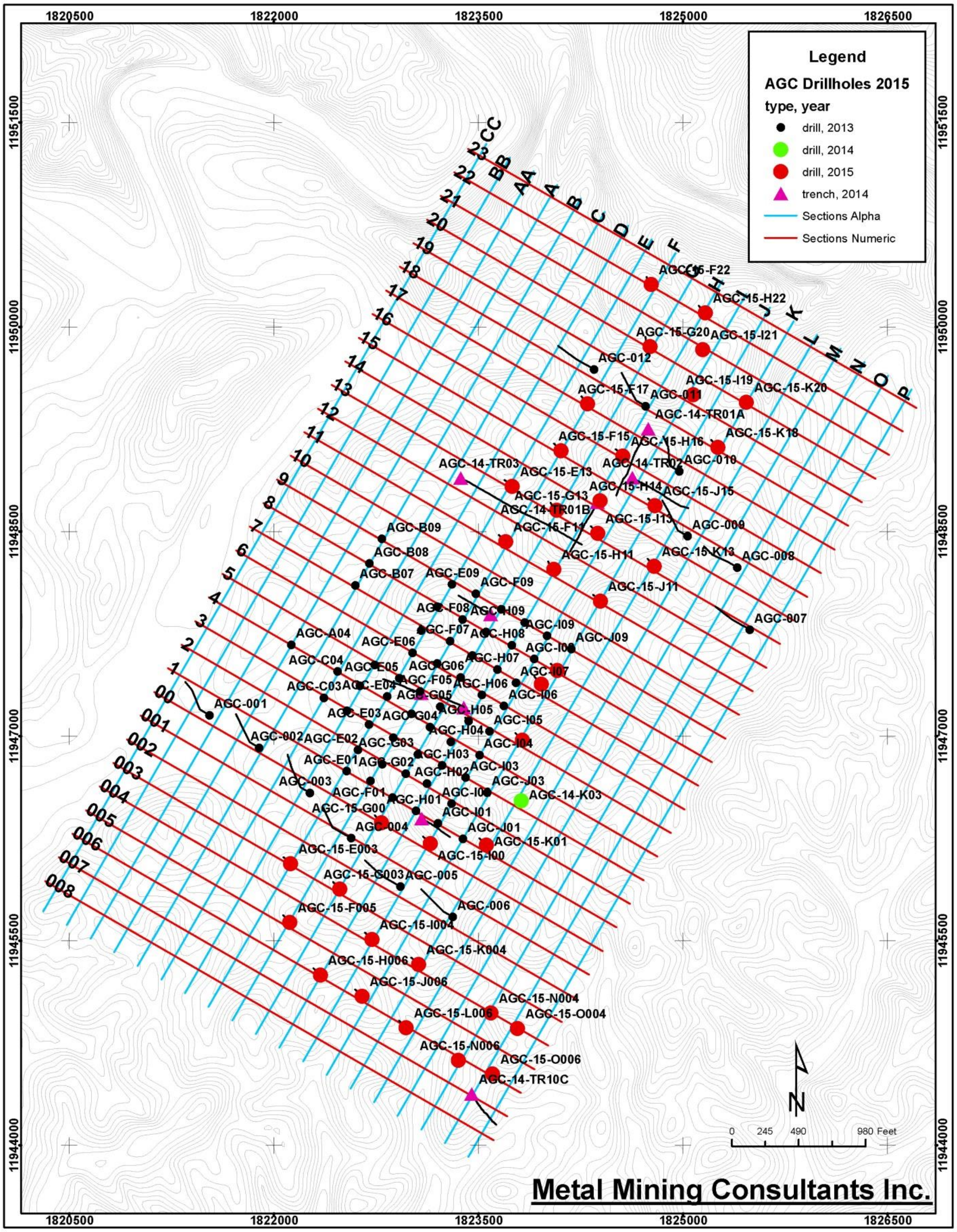
Vulcan software in common with other mining software systems and relies on a block modeling approach to represent deposit as a series of 3-D blocks to which grade attributes, and virtually any other attributes can be assigned. The software provides numerous means by which attributes can be assigned, and optimization routines are provided that allow block splitting, such that complex deposit outline details are not lost or smoothed out by regular size blocks. The approach used to build the final model is described briefly below.

14.2 Database for Geologic Model

Drill hole data for the Coosa Project is logged in the field and entered and maintained in Excel spreadsheets by Alabama Graphite. Drill hole logs are manually reviewed for discrepancies and inconsistencies.

Alabama Graphite provided MMC with a drill hole database in the form of Microsoft Excel spreadsheets which contained collar locations, surveys, assays and lithology data. MMC has validated the database and has certified the data to be clean and error free. The drill hole database has been converted to a Vulcan® Isis database with the identifier of "AGC_2015.dhd.isis"– a naming convention that works with Vulcan Software. The database contains 109 unique drill holes totaling 25,905 feet of drilling plus 9 trenches totaling 3,800 feet of sampling, treated as drill holes, which contain the graphitic carbon assays used in this resource estimation analysis.

Figure 14-1: Plan View of Coosa Drilling (UTM Grid in feet)



14.3 Density

A total of 242 specific gravity measurements were used to define the density of each lithology block in the model according to the oxidation state. Lithology's without sufficient density measurements in oxide and transitional material were determined by using a conversion factor. The conversion factor was calculated by using data from QGS samples. The data suggests that the density decreases by 5% from each oxide state. Table 14-1 displays the rock types by oxidation state and number of samples used to calculate the average density used in the model.

Table 14-1: Rock Types and Corresponding Density Values

Lithology (oxidation state)	Count	Min S.G	Max S.G	Average of S.G	Calculated	Final Density	Conversion to tons/cu ft.
QBGS	17	2.73	3.03	2.875		2.875	0.08973
RD	17	2.73	3.03	2.875		2.875	0.08973
TR	0	0	0	0.000	2.730	2.730	0.08521
OX	0	0	0	0.000	2.461	2.461	0.07681
QGS	171	2.08	2.88	2.689		2.689	0.08392
RD	159	2.17	2.88	2.709		2.709	0.08455
TR	5	2.44	2.68	2.572		2.572	0.08028
OX	7	2.08	2.47	2.319		2.319	0.07237
QMBGS	54	2.52	2.81	2.691		2.691	0.08398
RD	53	2.58	2.81	2.694		2.694	0.08408
TR	1	2.52	2.52	2.520		2.520	0.07866
OX	0	0	0	0.000	2.272	2.272	0.07091
Total	242	2.08	3.03	2.702		2.702	0.08434
Conversion Factor - Reduced to Transitional				0.950			
Conversion Factor - Transitional to Oxidized				0.901			
Conversion Factor - Grams/cu cm -> lbs./ cu ft.				62.428			

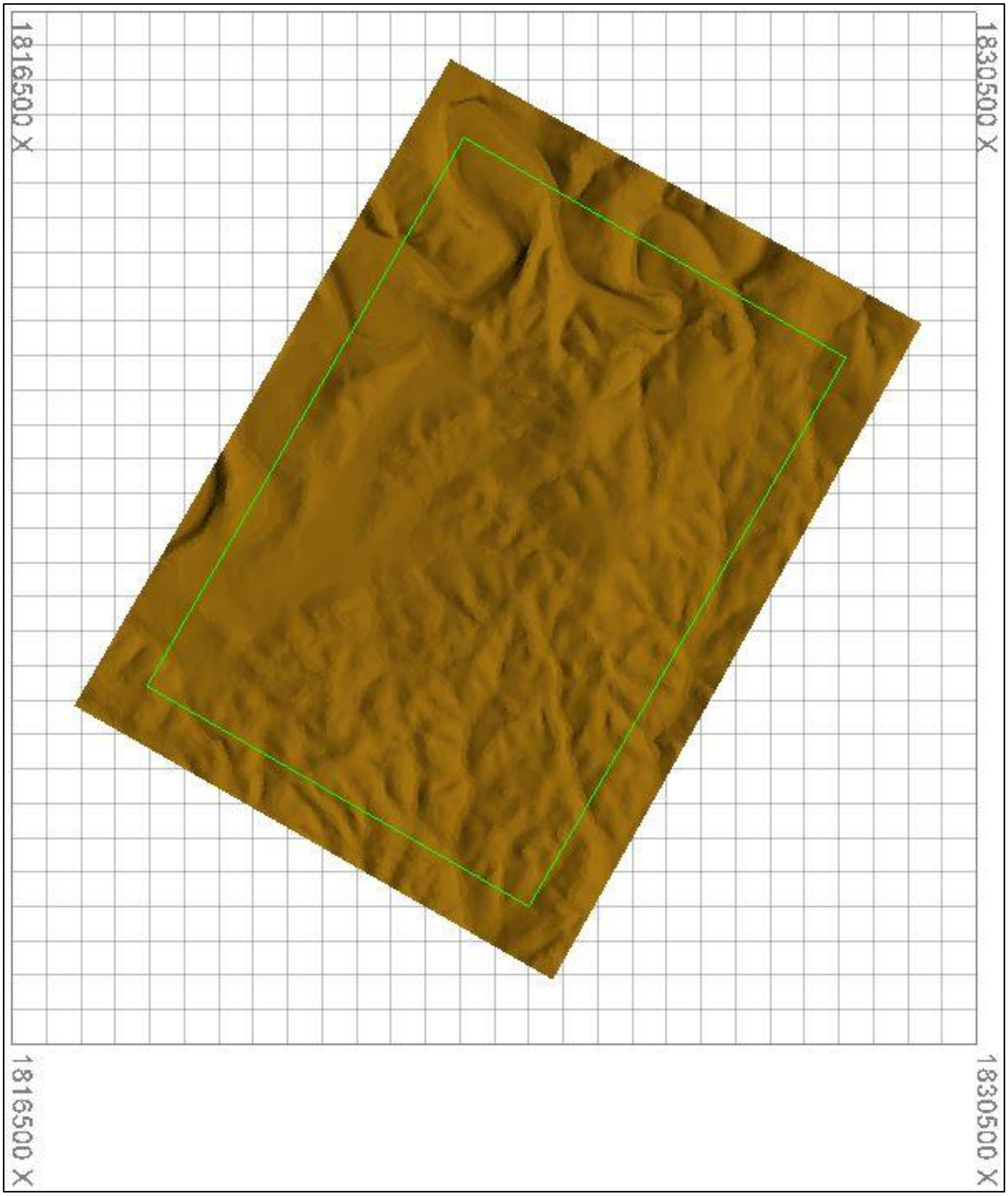
NOTE:

1. RD – reduced or unweathered, sulfide-bearing rock
2. TR – partly oxidized with some sulfides, transitional between RX and OX
3. OX – oxidized

14.4 Topographic Data

Alabama Graphite Corp. contracted Perry Remote Sensing, LLC (PRS) to purchase a precision digital elevation model (DEM) data for 91 sq. km of the project area. The DEM data was used to make a triangulation surface in Vulcan which is used to code the block model with blocks that are in the ground. Drill Hole collars were also registered to this surface. (Figure 14-2).

Figure 14-2: Triangulation of LiDAR Surface with Block Model Limits (Green Polygon)



14.5 Geologic Boundaries

For the Coosa deposit, geologic shapes were created using Vulcan. Geologic boundaries were constructed using by a series of wireframes that were digitized on section. Shapes were created for the QGS, QMBGS and QBGS rock types. In Vulcan, these triangulations were used to constrain the estimation appropriately within each geologic boundary. Figure 14-3 illustrates a typical cross section through the Coosa deposit and Figure 14-4 displays all of the geologic boundaries from an isometric view point.

Figure 14-3: Coosa Geologic Shapes in Cross Section, Central Grid Area

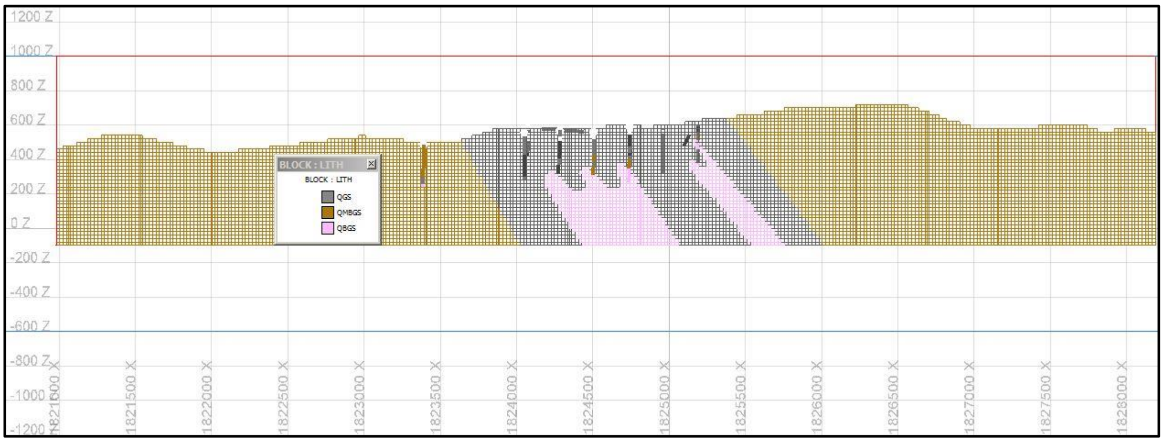
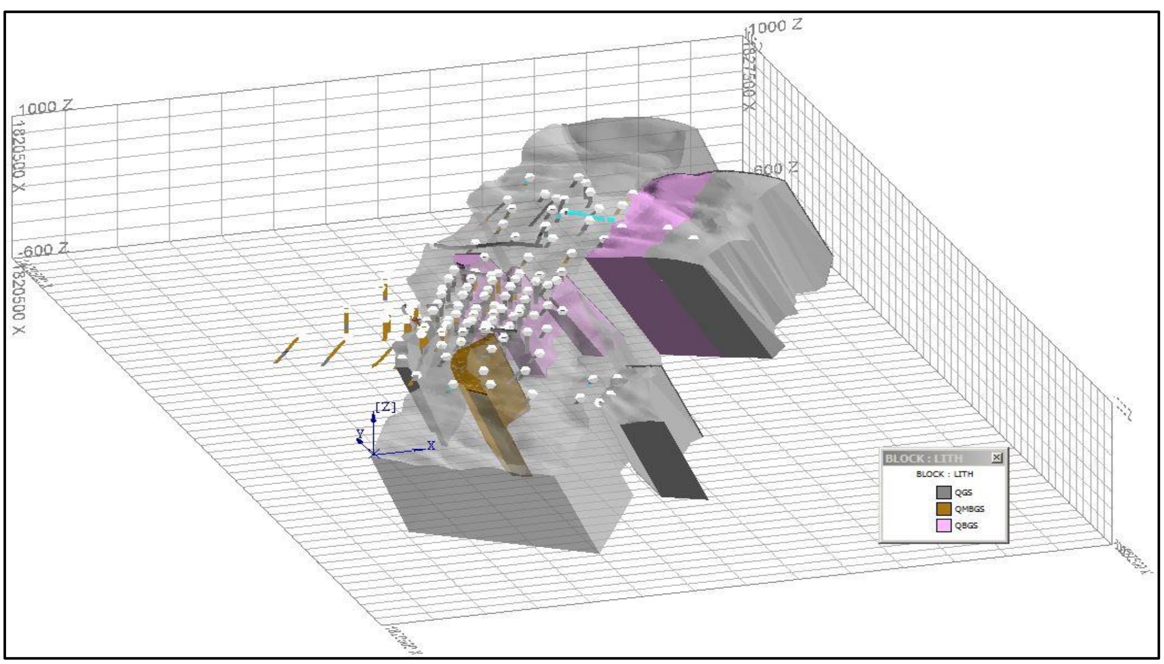


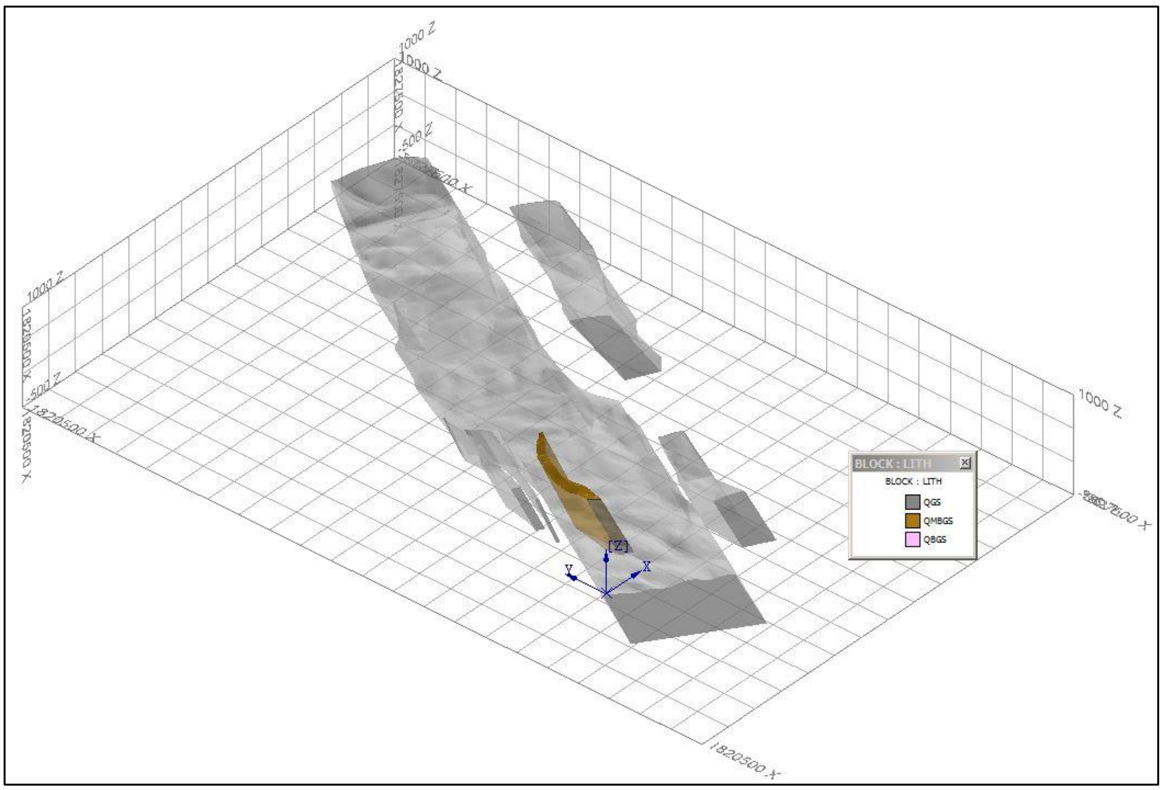
Figure 14-4: Isometric View of Coosa Geologic Boundaries



14.5.1 QGS and QMBGS Boundary

The QGS (Quartz-Graphite-Schist) surfaces and solids were created in Vulcan and were interpreted from data in the drill hole database data foliation measurements collected in the field. Based on the drilling and geologic interpretation, the QGS boundary represents known high grade graphitic schist and intermediate (INT) QGS to QMBGS (Quartz-Muscovite-Biotite-Graphite-Schist), which was defined through the drilling. The QGS boundary was used to constrain the grade estimation within this graphite rich zone and limit the influence of graphite grades in the surrounding lower grade QMBGS. Figure 14-5 illustrates the triangulations that were used to flag the geologic block model and constrain the resource estimation.

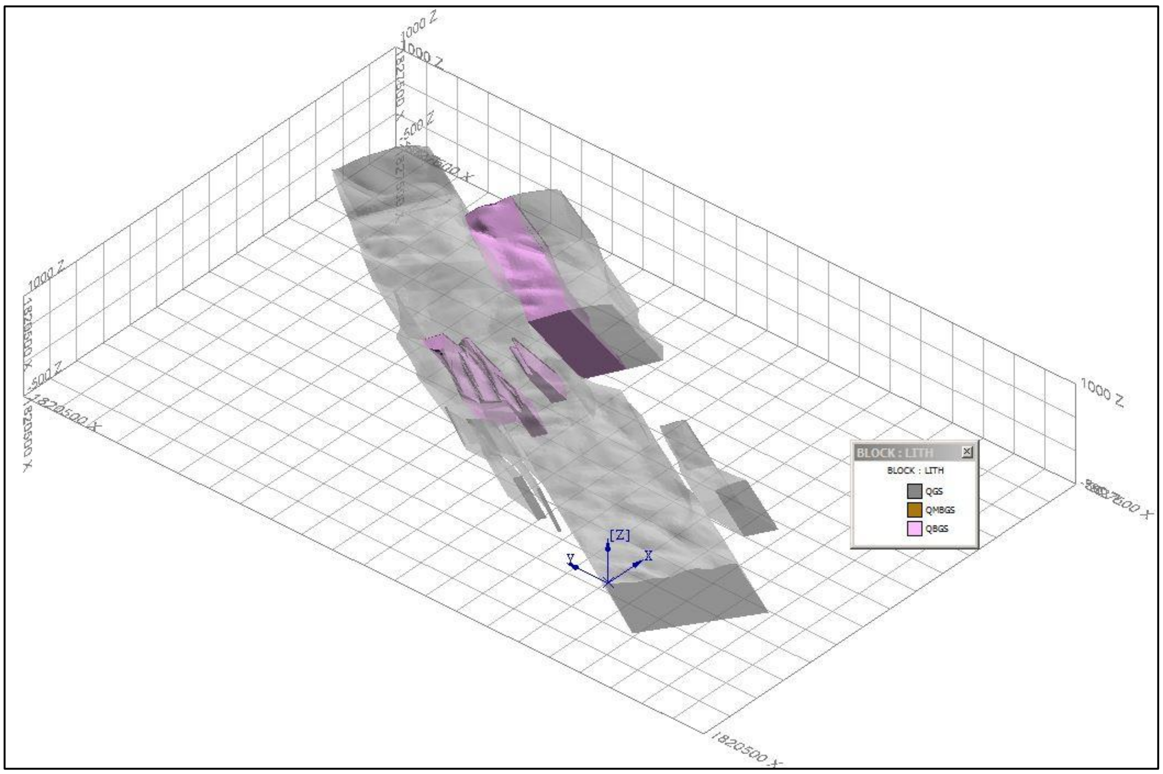
Figure 14-5: Triangulations of Coosa QGS and QMBGS Boundary



14.5.2 QBGS Boundary

The QBGS (Quartz-Biotite-Garnet-Schist) boundary was created using Vulcan and was derived from the drill hole database. This boundary represents a lithological unit that has a lower carbon content. The QBGS solids were constructed in areas with supported drill hole data. The solids were not extrapolated to great distances because of the unknown nature of the lithology. The drill holes located in the grid did not cross or fully penetrate the QBGS. This boundary was used to constrain the grade estimation. Figure 14-6 illustrates the geologic relationship between the QGS, QMBGS and the QBGS.

Figure 14-6: Triangulations of the QGBS Boundary



14.5.3 Oxide, Transitional and Reduced Surfaces

The oxide, transitional and reduced boundaries were modeled using drill hole data collected by AGC. Oxide material is strongly weathered (saprolite) and friable. Transitional material is weak to moderately weathered and oxidized. Reduced material does not exhibit any weathering or oxidation. The boundaries between these zones were modeled as surfaces and were used to assign density values to the blocks and used during the resource classification. Figure 14-8, is a cross-section showing the oxide-transitional-reduced boundaries modeled.

14.6 Drill Hole Compositing and Statistics

Drill Hole assays for Coosa were composited using ten foot down the hole composite lengths. A total of 2,754 10 foot composites were constructed. Intervals with missing assays or intervals with less than 50% recovery were ignored and a new composite was generated at that point. Codes were assigned based on the location of the composite centroid relative to the geologic triangulations and were utilized during the estimation process. Basic statistics were compiled and analyzed on the composites which include; distribution of composite lengths (Figure 14-7) and carbon graphite values (Table 14-2).

Figure 14-7: Composite Sample Lengths Statistics

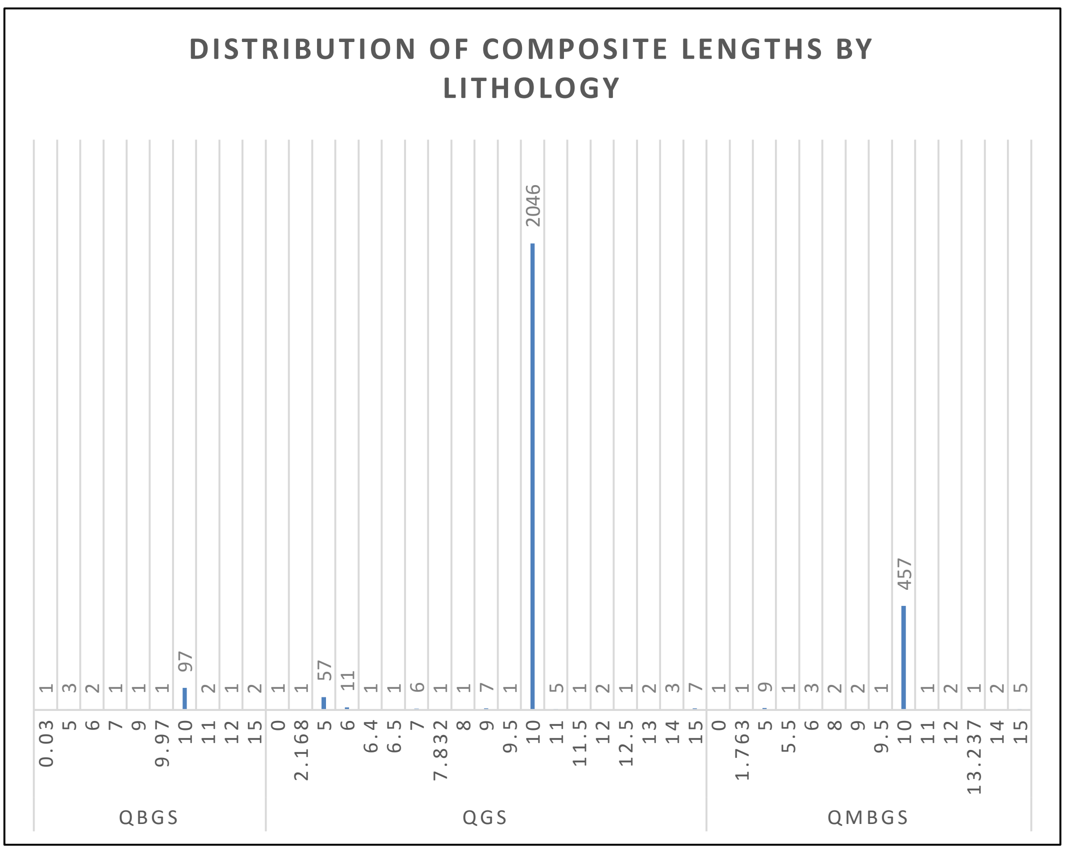


Table 14-2: Composite Capping Statistics by Lithology Type.

Capping Statistics				
Lithology	**Count**	**Min CG%**	**Max CG%**	**Average CG%**
QBGS	111	0.0025	1.49	0.33
QGS	2155	0.04	7.18	2.37
QMBGS	488	0.1	4.07	0.99
Grand Total	2754	0.0025	7.18	2.04

14.7 Block Model

The resource model for Coosa was constructed with Vulcan software using a block model. All of the required information about the deposit is stored in each individual block. This includes estimated characteristics carbon and statistical characteristics such as number of samples used in an estimate, distances to the nearest sample, number of drill holes used, etc., are stored in each individual block. Geologic triangulations were also used to identify the rock type of each block. Geologic codes stored in the block model were also used to assign the density within specific geologic boundaries. Table 14-3 outlines the framework of the Coosa block model.

For Coosa, blocks bound inside the QGS, QBGS and QMBGS geologic triangulation were estimated only using samples also within these boundaries. Samples within the QGS were not used to estimate grade within the QMBGS or QBGS. For example: only samples flagged in QGS were used to estimate blocks within QGS and only samples flagged QMBGS were used to estimate blocks within the breccia boundary.

Table 14-3: Coosa Block Model Framework

	East	North	Elevation
Block Model Reference Point	1,824,000	11,942,000	-100
Number of Blocks	460	320	55
Parent Block Size Feet	20	20	20
Orientation (absolute bearing of x axis around Z axis)	30 degrees		

14.8 Grade Estimation Parameters

Grades for graphitic carbon were estimated using inverse distance to the second power (ID2). Search ellipsoid orientations were determined by evaluating regional orientation, foliation and plunge of the graphitic schist observed in the field. Drill Hole data was used to define a semi-variogram which was used to determine the appropriate search distance for the major axis (Figure 14-8). Semi-variogram model results for the semi major and minor axis were poor so dimensions were selected to tie together mineralization from surface trenches and drill holes (semi-major axis) and limit the distribution of grades perpendicular to the dip of the graphitic schist (minor-axis). Blocks estimated on the first pass were flagged accordingly and omitted during the second pass. A two pass estimation was performed in the main host (QGS) using different minimum and maximum sample counts, minimum number of drill holes and search ellipsoids for each pass. A single pass estimation was performed on the secondary host (QMBGS). Only samples flagged in the composite file as QGS were used during the estimation of QGS blocks and only samples flagged in the composite file as QMBGS were used to estimate QMBGS blocks. Composites with a length of less than 5ft were omitted from the estimation. Table 14-4 below outlines the grade estimation parameters used.

Figure 14-8: Semi-Variogram Model (Major-Axis)

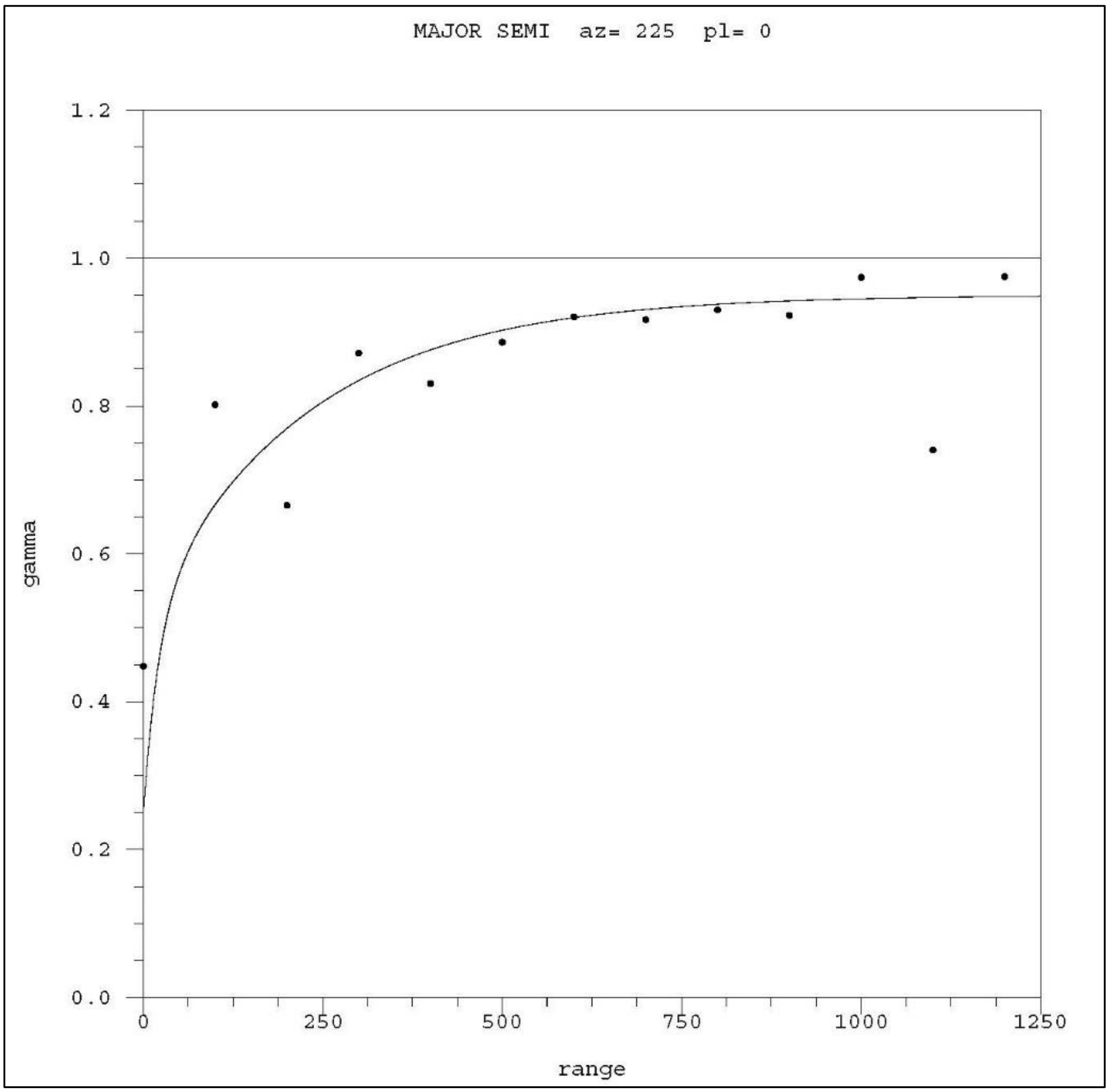


Table 14-4: Coosa Estimation Parameters

Gold Rock Estimation Parameters			
Estimation Type	**Inverse Distance Cubed (ID2)**		
Search Ellipsoid	**Bearing**	**Plunge**	**Dip**
QGS	35	10	-45
QMBGS	35	10	-45
Search Distance	**Major Axis**	**Semi-Major Axis**	**Minor Axis**
QGS Pass 1	375	250	25
QGS Pass 2	750	350	25
QMBGS Pass 1	250	100	25
Samples	**Min**	**Max**	
QGS Pass 1	4	10	
QGS Pass 2	2	20	
QMBGS Pass 1	3	20	
Maximum Samples per Drill Hole	**Max**		
QGS Pass 1	3		
QGS Pass 2	4		
QMBGS Pass 1	4		

14.9 Post Processing

14.9.1 Limiting Surface

In order to meet the prospects of eventual economic extraction, a limiting surfaces was created approximately 50ft below the TD of the drill holes and was used to “zero out” the CG% grades. Blocks below this surface are believed to be too speculative to classify and were not used to optimize a whittle resource pit.

14.9.2 Resource Pit Parameters

The Coosa resource was constrained to a Whittle optimized pit to demonstrate reasonable prospects of economic extraction. The block model was exported from Vulcan and then imported into Whittle. Input parameters were based on publically available graphite project costs. Table 14-5 outlines the parameters used for Whittle pit optimization.

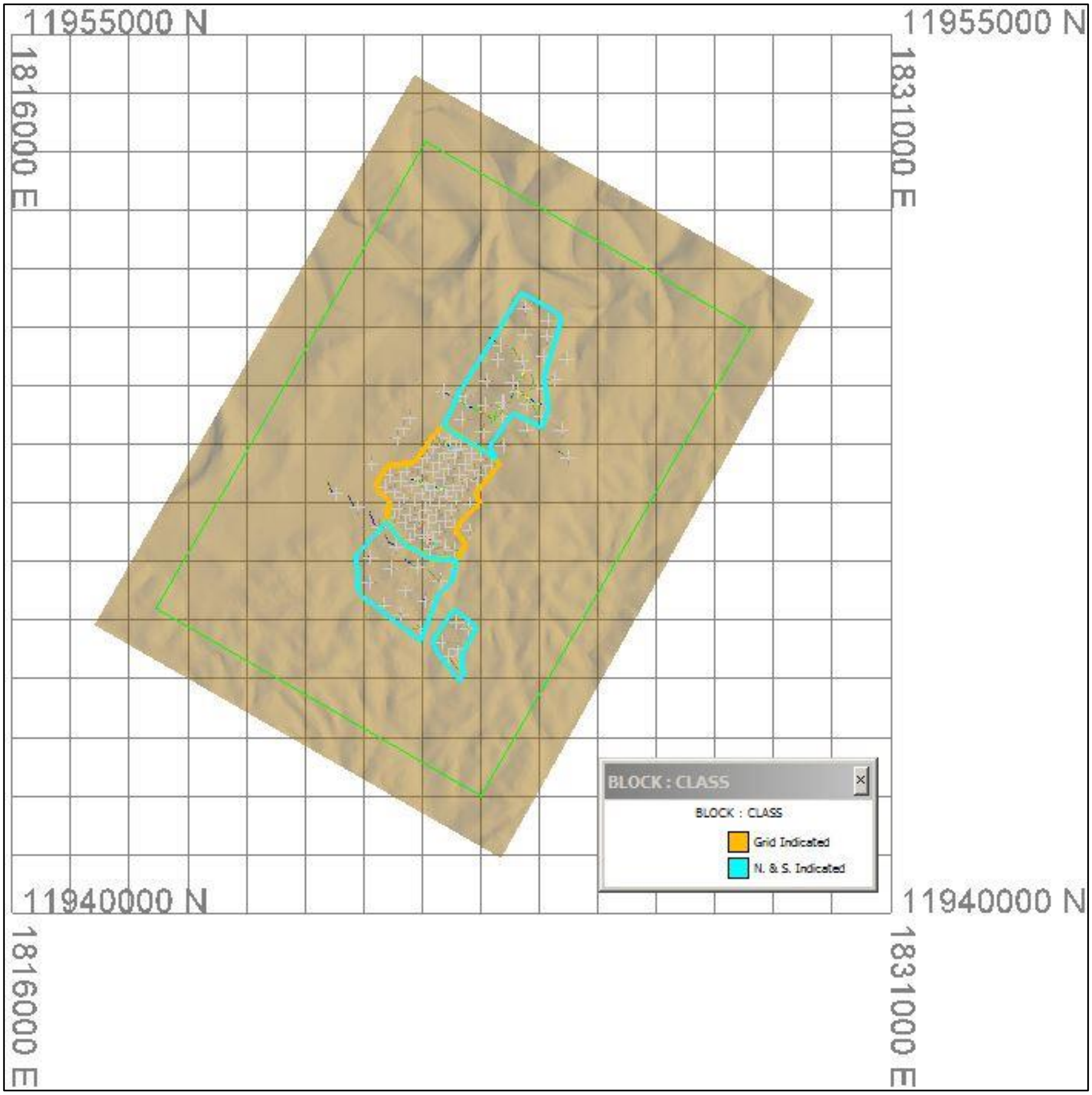
Table 14-5: Resource Pit Parameters

Resource Pit Parameters		
Parameter	Unit	Value
Mining Cost	USD/ton mined	$4.81
Processing Cost	USD/ton Processed	$14.56
G&A	USD/ton Processed	$4.54
Total Processing Cost	USD/ton Processed	$19.10
Graphite Selling Price	USD/ton	$1,450.00
Pit Slopes	°	60°
Graphite Recovery	%	82.8%

14.9.3 Measured, Indicated and Inferred Classification

The Coosa Project resource was classified as Indicated Mineral Resources or Inferred Mineral Resources. In the opinion of the QP the company has not obtained adequate information regarding metallurgy, graphite flake size and geologic continuity to classify any part of the resource as Measured Mineral Resource. The grid holes demonstrated adequate spacing and continuity of graphite grade to be classified as indicated as well as oxide and transitional material, to the North and South of the grid, where surface trench samples were taken and continuity is demonstrated with drill holes. Resource blocks estimated, to the north and south of the grid, in the reduced zone may demonstrate reasonable prospects of economic extraction however warrant addition drilling at depth to prove grade continuity within the reduced graphitic schist before they can be upgraded from the Inferred to the Indicated Mineral classification. Resource blocks were flagged using triangulations that represent the zones described above Figure 14-9displays the Indicated and Inferred domains that were used to flag the block model. All blocks in the orange domain were classified as Indicated and blocks in the blue domains were classified as Indicated (oxide and transitional) and Inferred (reduced).

Figure 14-9: Resource Classification Domains



14.10 NI 43-101 Mineral Resource

14.10.1 Indicated Resource

The 13 October 2015 Coosa Project Indicated Mineral Resources in Table 14-6 are reported at a graphitic carbon cutoff grade of 1%. The resources are based on 109 unique drill holes totaling 25,905 feet of drilling plus 9 trenches totaling 3,800 feet of sampling, treated as drill holes. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-6: Coosa Project Indicated Mineral Resources, 13 October 2015

	Indicated Resource		
Cutoff %	Tons	C%	C Tons
0.5	85,112,000	2.27	1,932,000
1	**78,488,000**	**2.39**	**1,876,000**
1.5	69,842,000	2.53	1,767,000
2	54,436,000	2.74	1,491,000

14.10.2 Inferred Resource

Inferred mineral resources represent that material that MMC considers to be too speculative to be included in economic evaluations at this point. Additional drilling will be required to convert Inferred Mineral resources to Measured or Indicated Mineral Resources. Table 14-7 lists the Inferred Mineral Resources for the Coosa Project and is reported at a graphitic carbon cutoff grade of 1%.

Table 14-7: Coosa Project Inferred Mineral Resources, 13 October 2015

	Inferred Resource		
Cutoff %	Tons	C%	C Tons
0.5	93,223,000	2.3	2,144,000
1	**79,443,000**	**2.56**	**2,034,000**
1.5	70,283,000	2.74	1,926,000
2	58,207,000	2.94	1,711,000

14.11 Mineral Inventory

14.11.1 Indicated Mineral Inventory

The Indicated Mineral Inventories (Table 14-8 through Table 14-10) has been classified and separated by material type (oxide, transitional and reduced) and reported within the same resource pit used to report the Mineral Resource.

Table 14-8: Indicated Oxide Mineral Inventory

	Indicated Oxide - Mineral Inventory		
Cutoff %	Tons	C%	C Tons
0.5	24,525,000	2.45	601,000
1	23,231,000	2.54	590,000
1.5	21,071,000	2.68	565,000
2	17,604,000	2.85	502,000

Table 14-9: Indicated Transitional Mineral Inventory

Indicated Transitional - Mineral Inventory			
Cutoff %	**Tons**	**C%**	**C Tons**
0.5	19,372,000	2.32	449,000
1	17,733,000	2.46	436,000
1.5	15,610,000	2.63	411,000
2	12,734,000	2.82	360,000

Table 14-10: Indicated Reduced Mineral Inventory

Indicated Reduced - Mineral Inventory			
Cutoff %	**Tons**	**C%**	**C Tons**
0.5	41,215,000	2.13	878,000
1	37,524,000	2.26	848,000
1.5	33,161,000	2.39	793,000
2	24,098,000	2.62	631,000

14.11.2 Inferred Mineral Inventory

The Inferred Mineral Inventories (Table 14-11 through Table 14-13) has been classified and separated by material type (oxide, transitional and reduced) and reported within the same resource pit used to report the Mineral Resource.

Table 14-11: Inferred Oxide Mineral Inventory

Inferred Oxide - Mineral Inventory			
Cutoff %	**Tons**	**C%**	**C Tons**
0.5	10,184,000	2.13	217,000
1	8,063,000	2.47	199,000
1.5	6,555,000	2.76	181,000
2	5,115,000	3.03	155,000

Table 14-12: Inferred Transitional Mineral Inventory

Inferred Transitional - Mineral Inventory			
Cutoff %	**Tons**	**C%**	**C Tons**
0.5	7,661,000	2.02	155,000
1	5,758,000	2.43	140,000
1.5	4,640,000	2.72	126,000
2	3,552,000	3	107,000

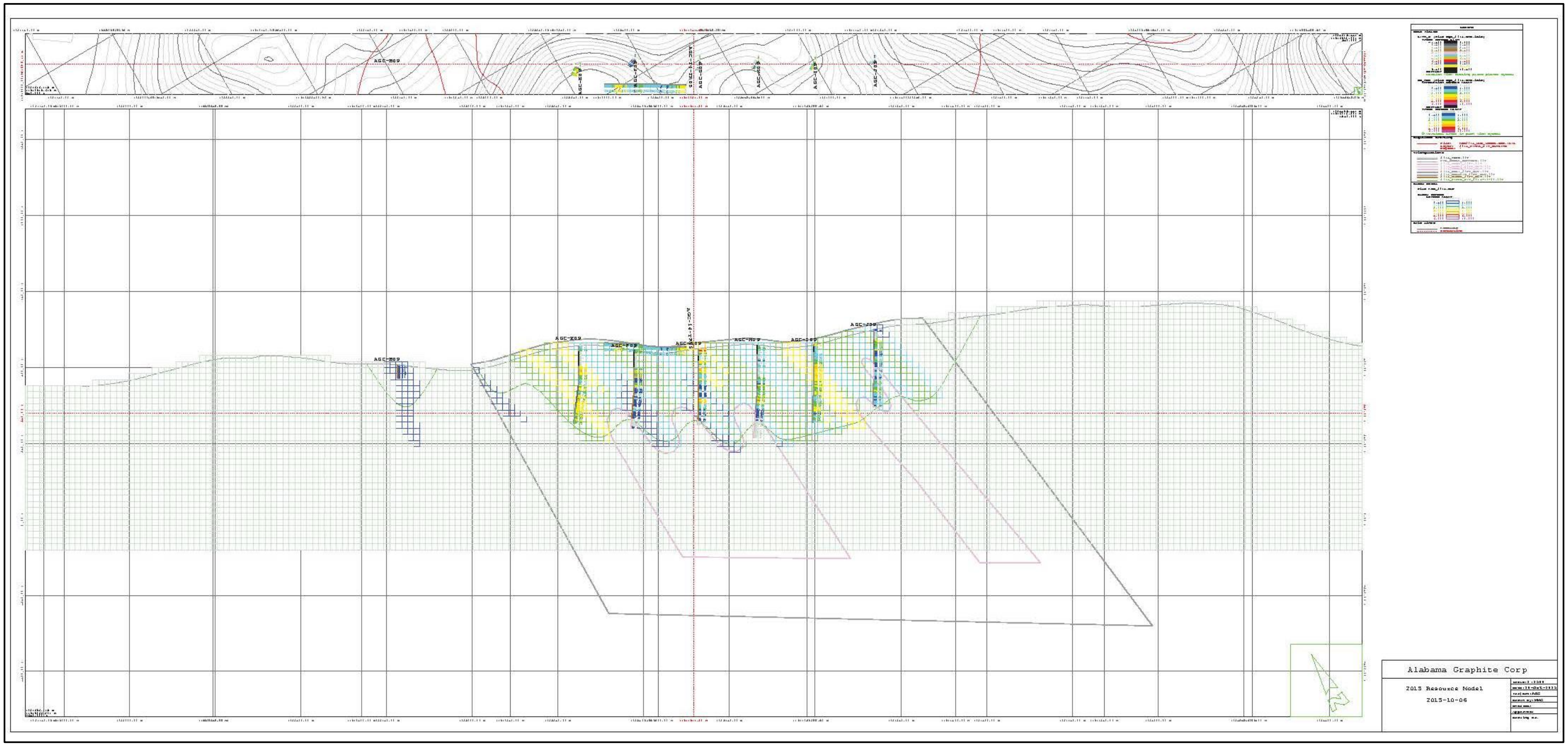
Table 14-13: Inferred Reduced Mineral Inventory

Inferred Reduced - Mineral Inventory			
Cutoff %	**Tons**	**C%**	**C Tons**
0.5	75,378,000	2.36	1,779,000
1	65,622,000	2.59	1,700,000
1.5	59,088,000	2.74	1,619,000
2	49,540,000	2.92	1,447,000

14.12 Block Model Validation

Statistical and visual checks were performed by MMC of the estimated block model to ensure there were no discrepancies in the grade estimation routines and to ensure the geometry of mineralization meets the configuration that the geologists expected (Figure 14-10).

Figure 14-10: Block Model Validation



15 MINERAL RESERVE ESTIMATES

The Coosa Graphite Project is at a PEA level of study and therefore has no reserves at this time.

16 MINING METHODS

16.1 Geotechnical

During the site visit by AGP in 2015, overburden depths of approximately three feet were observed in several road cuts and test pit areas. Geotechnical data relevant to the open pit slopes is limited at this stage of study, which is typical of most mine development projects at the PEA stage. Geotechnical data will be needed for future geotechnical evaluations in the area of proposed pit slopes, waste dumps, and overburden storage areas.

At this stage of study, no information is available regarding the hydraulic conductivity of the rock mass on the Coosa project.

16.1.1 Preliminary Open Pit Slope angles

Drill core data alone is insufficient to develop slope design criteria. The depths of the oxidized horizon in the project area ranged from approximately 10 ft. to slightly more than 100 ft. In order to lower capital and operating costs of the project, pit targets were restricted to the oxidized horizon near the surface of the deposit. It is assumed that no blasting will be required for mining this horizon.

Final haul roads can be placed along the oxidized/transitional horizon contacts in pit bottoms and typically tied into surface topography. As a result, the pit walls were designed using the inter-ramp angles as shown in Figure 16-1 and Table 16-1.

Figure 16-1: Open Pit Slope Geometry Definitions

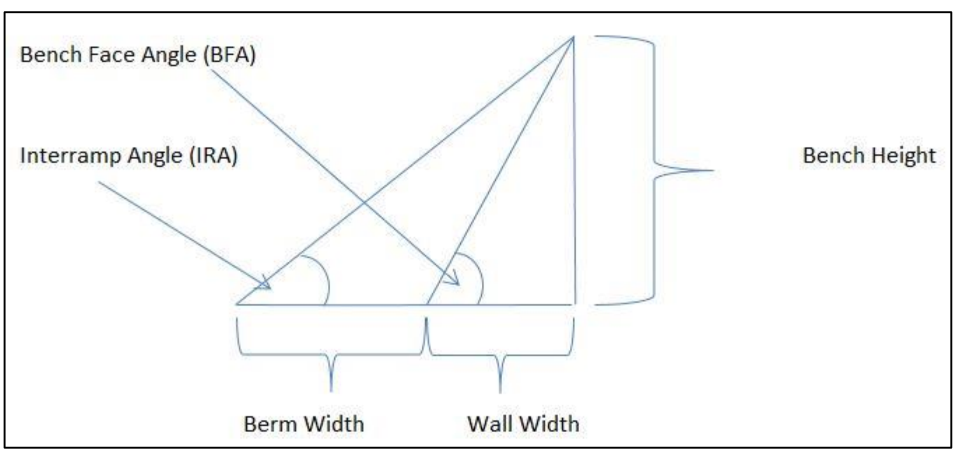


Table 16-1: Coosa Pit Slope Design Parameters

Catch Bench Geometry			Inter-Ramp Geometry
Height (ft)	Angle (°)	Width (ft)	Angle (°)
20	65	10	46

16.2 Open Pit Mining

16.2.1 Introduction

This section discusses the mine planning performed for the 2015 PEA study. Open pit mining was selected as the mining method to examine the Coosa deposit based on the decision to restrict mining to the shallow oxidized horizon as well as the very low strip ratio of the deposit. As there is very little waste rock, and mining would not occur in competent rock, underground bulk mining was not considered to be a practical mining method for this deposit.

16.2.2 Geological Model Importation

The resource model developed by MMC using Vulcan software (Section 14) is used as the basis for the examination of the open pit potential of the Coosa deposit. From this model AGP created an engineering block model in MineSight with additional items for use in block valuation and economic pit shell generation. Table 16-2 shows the boundaries used for the model.

Table 16-2: Model Boundaries

Model Dimension	X	Y	Z
Minimum (ft)	0	0	-100
Maximum (ft)	9,200	6,400	1,000
Dimension (ft)	20	20	20

All the grade item values in the engineering model remain the same as in the geological model.

16.2.3 Economic Pit Shell Development

Determination of the potential open pit was accomplished with the use of the Lerch-Grossman routine in the MineSight software. The input parameters for the economic pit shells are shown in Table 16-3.

Table 16-3: Economic Pit Shell Parameters

Operating Cost Parameter	Units	Parameter Value
Mining Cost	$/ton	3.65
Process Cost	$/ton mill feed	14.56
General and Administrative	$/ton mill feed	4.54
Process Plant Recovery	%	92.0
Purification Recovery	%	90.0
Combined Metallurgical Recovery	%	82.8
Net Graphite Price	$/ton product	5,721

Mining costs do not include blasting as the pit shells will be restricted to the oxidized horizon.

A restriction was applied to the northern portion of the deposit so that the pit expansions would not disturb the nearby Weogufka creek.

Economic pit shells were generated with net graphite prices ranging from $572/ton of product to $5,721/ton of product. The target was to supply approximately 25 years of mill feed to the mill for a 16.5k tpa product process rate. The ultimate pit shell was then determined to be Pit 06 which had an $1,287/ton net graphite price (net of filter cake transportation to purification, as well as purification, shaping and coating) with a marginal cut-off grade of 0.5% Cg. That shell provided 21.6 million tons of mill feed with a grade of 2.26% Cg and a strip ratio of 0.06:1 (tons waste/ton mill feed).

Sets of internal shells were examined to assist in developing the pit shells with the highest value for starter pits. These provided guidance on the phasing shapes and sequence and is discussed in more detail in the Pit Design and Phase Development section.

16.2.4 Model Dilution

Prior to detail design of the pit phases commencing, model dilution was calculated for each block.

Losses and dilution were evaluated with consideration of the size of the mining excavators, deposit geometry, bench height, and the visual differentiation between barren and mineral laden material. Distinguishing between plant feed material and waste or low grade will require additional assessment with blast hole assaying.

The dilution and mining losses were modelled on a block by block basis. The block model dimensions are 20 ft. long, 20 ft. wide and 20 ft. tall. (20x20x20). The block is considered to be internally diluted but contact dilution needs to be applied to the block. Smaller loaders and backhoes are proposed for mining of the material which provides for the opportunity of selective mining. The proposed loader has a bucket width of 12.5 ft. AGP estimates that the use of 3 ft. of dilution on any block side would be appropriate. Selectivity with the specified equipment may not be able to clean closer than that without additional sampling or visual cues to make the determination where the dig line should be placed.

Utilizing the block dimensions, a calculation of the percentage of dilution for each block depending on the number of sides with diluting material was estimated. The dilution estimate varies depending on the number of sides that are in contact with blocks below cut-off. The percentage of dilution per side is shown below in Table 16-4:

Table 16-4: Dilution Percentages Applied

Number of Diluting Blocks (sides)	Dilution Percentage (%)
0	0%
1	13.0%
2	23.1%
3	31.0%
4	37.5%

The script employed to calculate the dilution considered the grade of the diluting block. Blocks that were entirely surrounded by above cut-off material were not diluted. The script was completed with a base milling cut-off of 1% Cg. Blocks below 1% Cg were considered waste.

The net result of the dilution calculation was a slightly less than 1% drop in overall grade. This is a result of the fairly homogenous nature of the deposit with the contacts being gradational.

The dilution percentage is what AGP expected the feed losses to be. For this reason, the dilution was assumed to be equal to the feed loss and only a diluting effect on the grade was applicable.

16.2.5 Pit Design and Phase Development

Due to the decision to only mine the oxidized mill feed initially, all waste and overburden material was to be hauled outside of potential mining areas. Therefore, no backfilling of pits was to be planned for so that future potential mining below the oxidized horizon could be preserved. The phase sequence was determined on the contained unit value.

With these considerations, four different starter phases were identified, followed by six small pits. Some were standalone pits and some were pushbacks of other phases. In all cases the pit designs adhered to the geotechnical criteria of Table 16-1.

The numbering of the phases did not always follow the mining sequence, but rather followed the design development sequence based on contained value. During detailed designs, significant effort was placed on raising the grade of pits where reasonable. This resulted in somewhat smaller final pit designs than the original economic pit shells. The detailed tons and grade of the phases and pits have been shown in Table 16-5 by category.

Table 16-5: Pit Phase Design Tonnages and Diluted Grade by Category

	Mill Feed (ktons)	Grade (DCg%)	Mill Feed (ktons)	Grade (DCg%)	Overburden (ktons)	Rock (ktons)	Total Waste (ktons)	Strip Ratio
	Indicated		Inferred					
Phase 1	663	3.26	347	3.42	60	5	65	0.06
Phase 2	955	3.16	10	3.05	52	84	137	0.14
Phase 3	1,457	3.19	54	2.98	62	48	110	0.07
Phase 4	1,323	3.06	1	3.19	59	0	59	0.04
Subtotal	**4,398**	**3.16**	**411**	**3.35**	**233**	**138**	**371**	**0.08**
Pit P1	-	-	474	3.22	31	216	247	0.52
Pit P2	476	2.79	11	2.67	28	19	47	0.10
Pit P3	2,831	2.75	973	2.76	191	376	567	0.15
Pit P4	440	2.90	198	2.64	40	33	73	0.11
Pit P5	136	2.78	181	2.75	29	145	175	0.55
Pit P6	4,302	2.60	369	2.90	210	54	264	0.06
Subtotal	**8,186**	**2.68**	**2,205**	**2.87**	**528**	**843**	**1,372**	**0.13**
Total	**12,584**	**2.85**	**2,616**	**2.95**	**761**	**981**	**1,742**	**0.11**

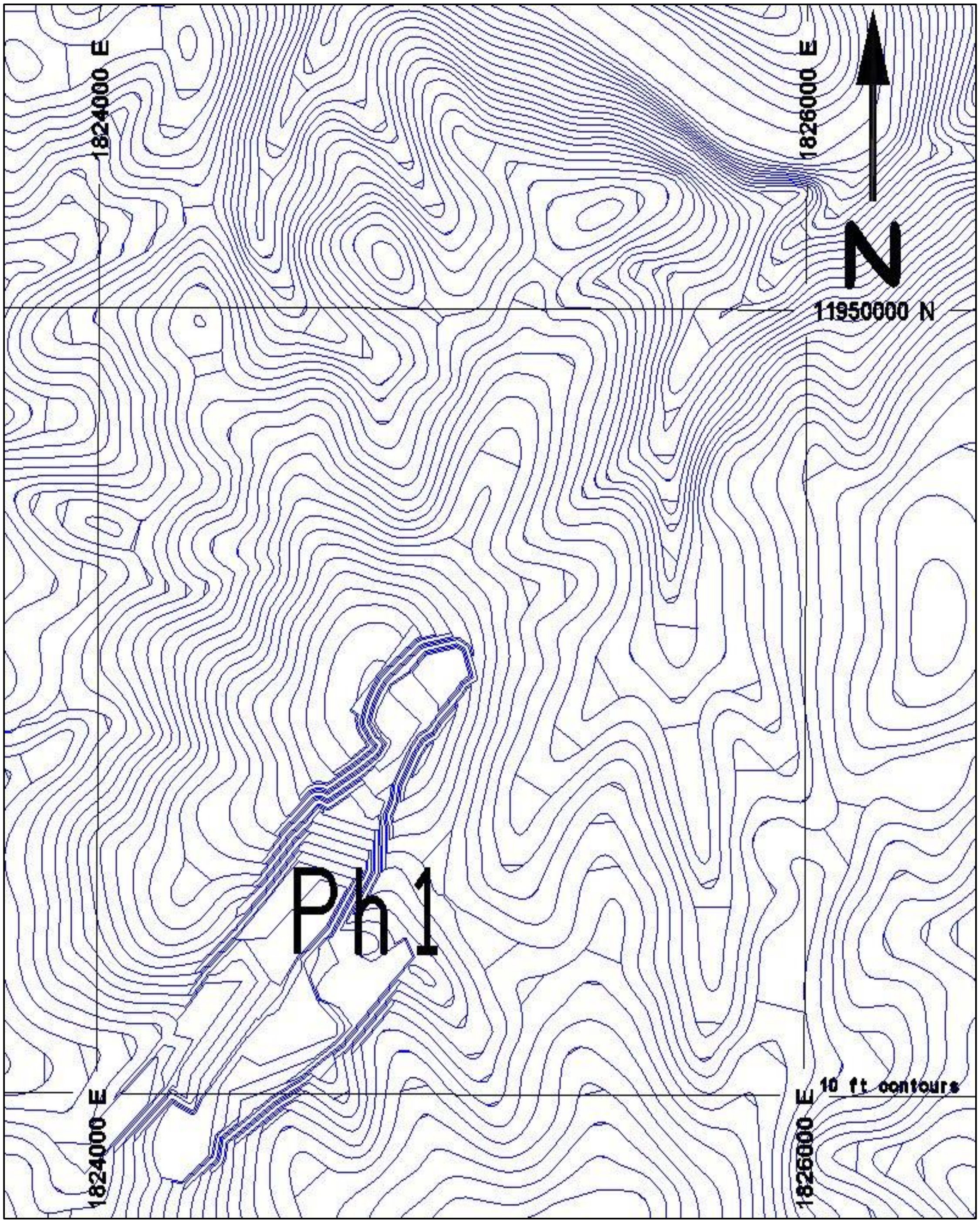
The marginal milling cut-off grade was calculated to be 0.5% Cg. An elevated cut-off grade of 1% DCg (diluted graphitic carbon) was applied for bench summaries, but this had a negligible effect on the overall grades as very little material falls in the 0.5-1.0% DCg range.

The phase designs were generally developed in order of highest value to the lowest value. Pit P1 is an exception to this as it has high diluted graphitic carbon grades (DCg), but there is very limited drilling within it to support moving it into the starter phases. It also contains more waste than the other starter phases.

Surface topography is made up of fairly gentle slopes, so eight percent haul roads can be easily developed to access benches. Haul road widths of 60 ft. will accommodate the small haul trucks envisaged for the Coosa project. All phases and pits will be mined down to the gently sloping oxide/transitional material contact, with the majority of in-pit haul access along this contact. The detailed designs for each have been illustrated in the following figures.

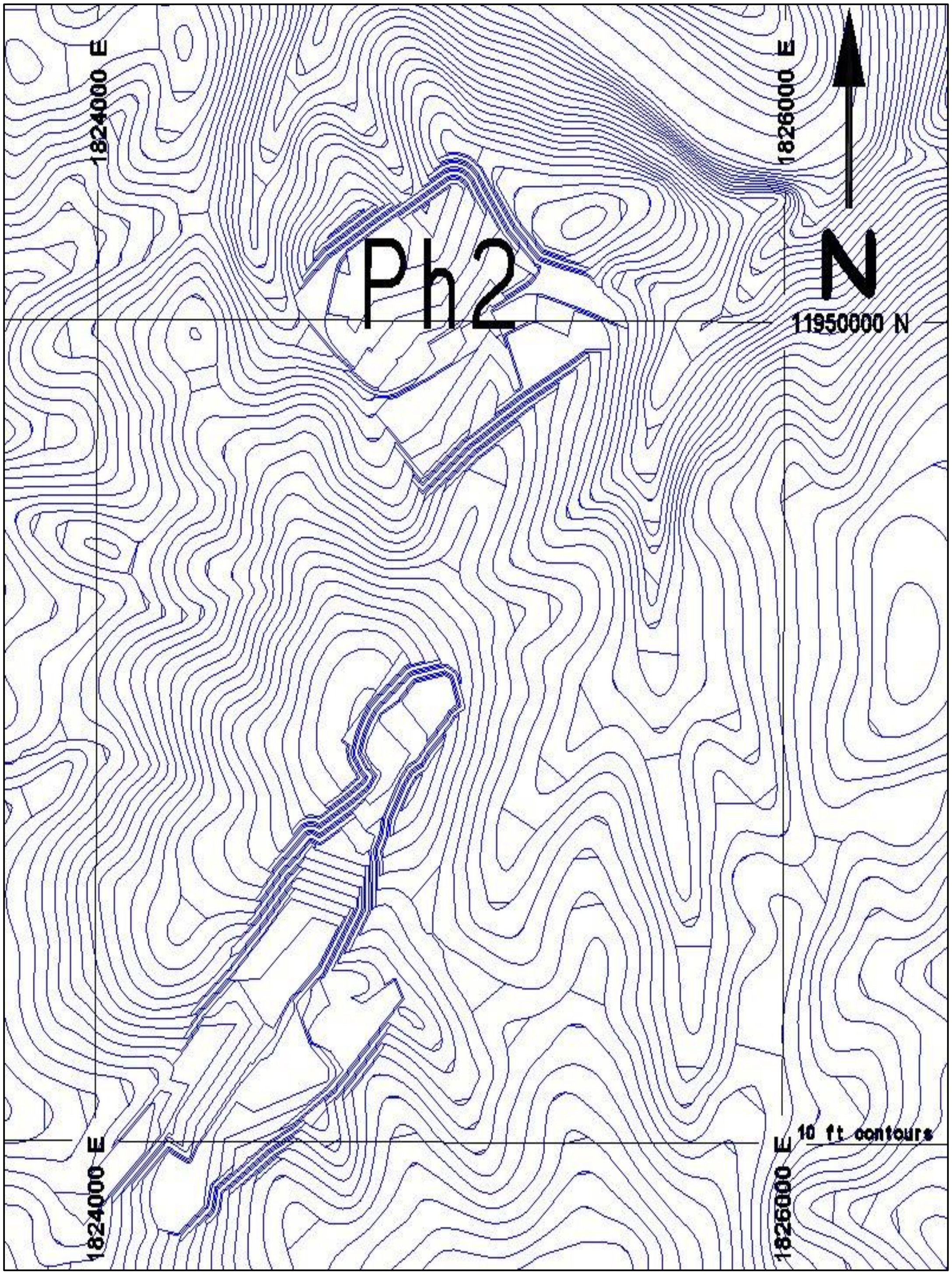
Phase 1 shown in Figure 16-2 is developed in the northern area of the deposit and represents the highest grade starter phase. This phase also has a very low strip ratio of 0.06 tons' waste per ton of mill feed, so there will be minimal waste stripping required to reach mill feed. Mill feed and waste will be hauled to the east along topography, while a small amount of overburden will be hauled downhill to the southwest. Mining will go to the footwall of the oxide zone and follow the footwall of the deposit leaving a ramp along the north and east sides.

Figure 16-2: Plan View of Phase 1



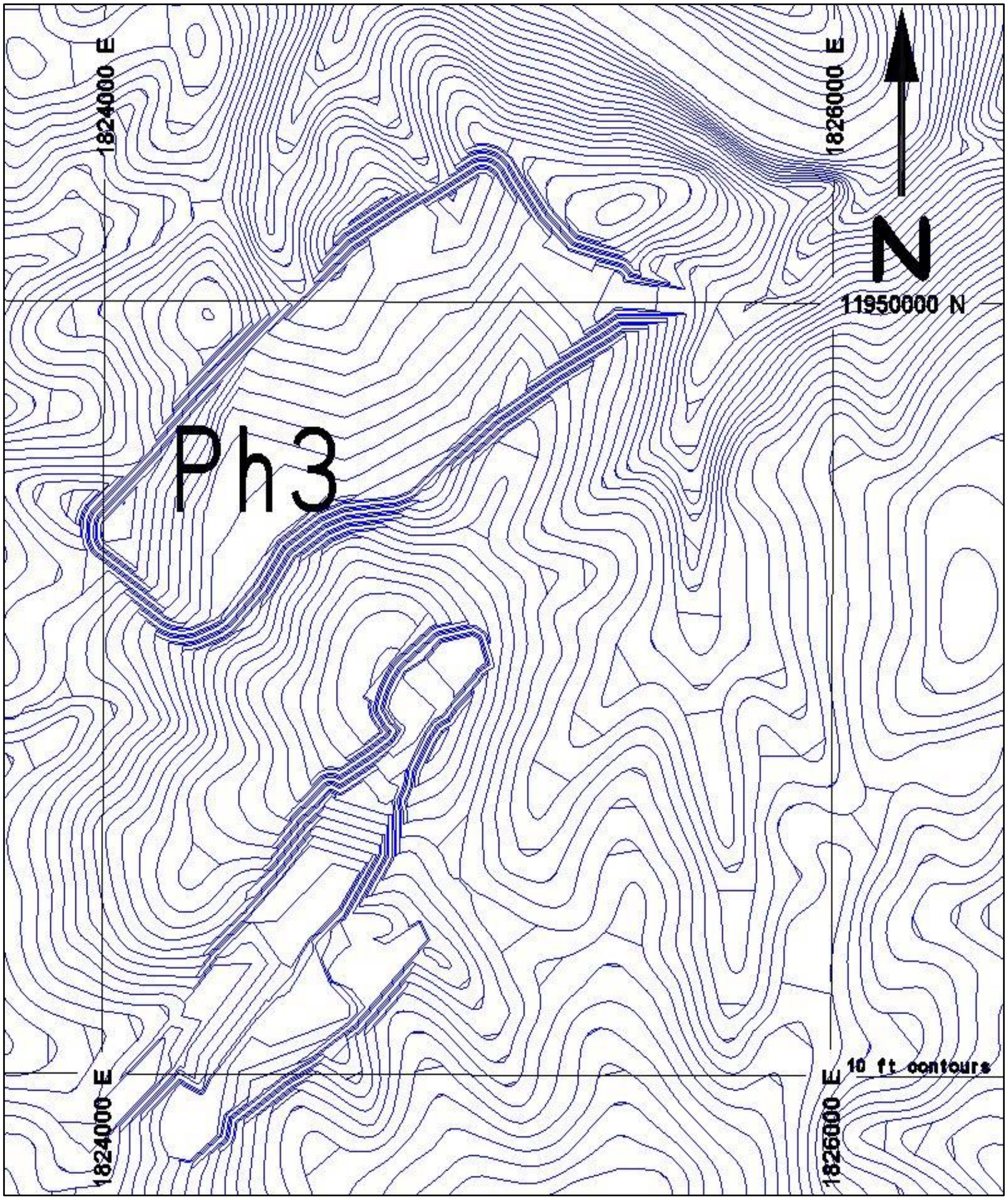
Phase 2, shown in Figure 16-3 is developed next and is the furthest north mining area of the project. It utilizes portions of the same waste and mill feed haul roads to the southeast as Phase 1. Overburden will again be hauled to the southwest.

Figure 16-3: Plan View of Phase 2



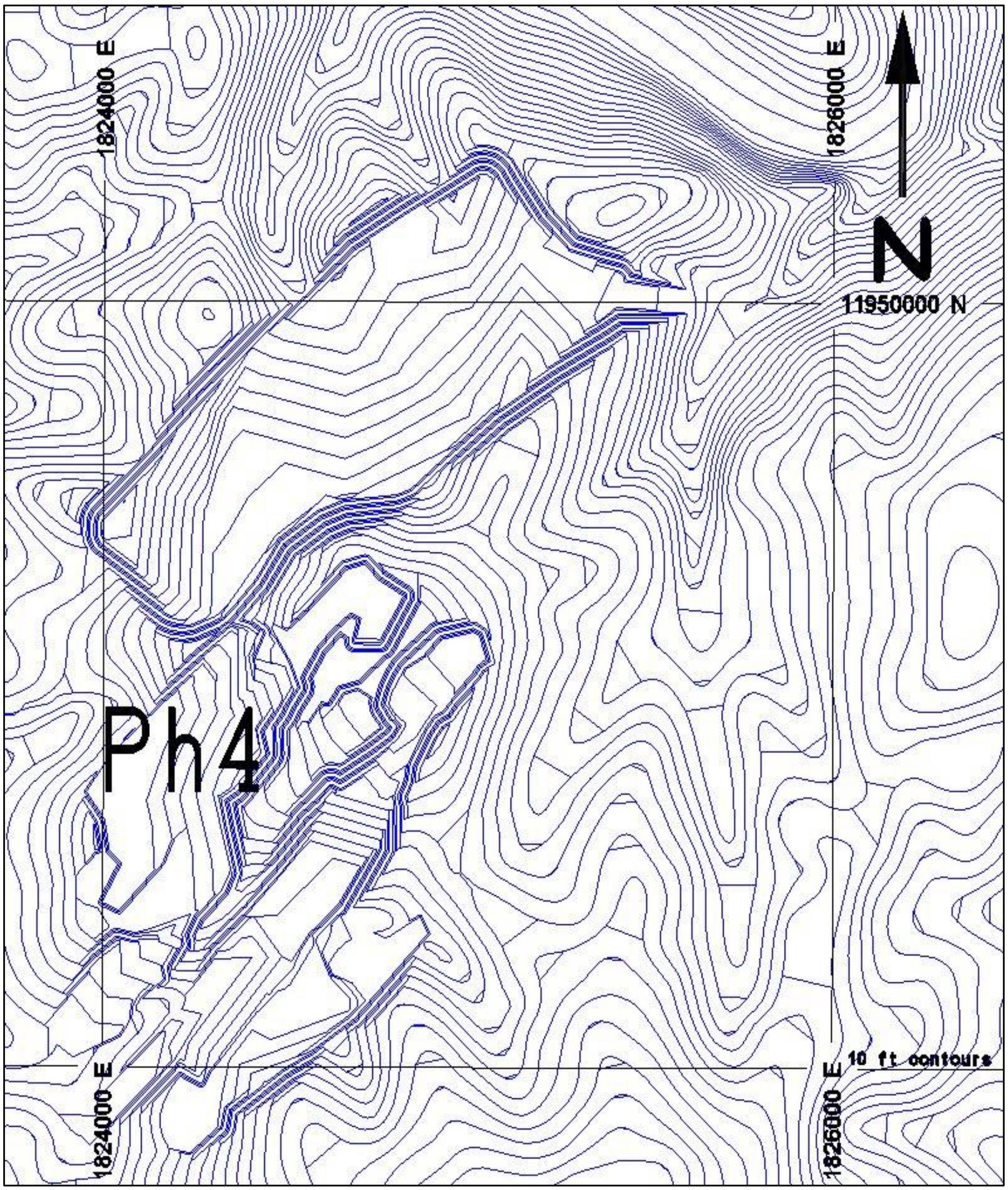
Phase 3, shown in Figure 16-4 has access to the upper benches along topography, but then utilizes the same ramp system from Phase 2 to advance the pit development to the southwest.

Figure 16-4: Plan View of Phase 3



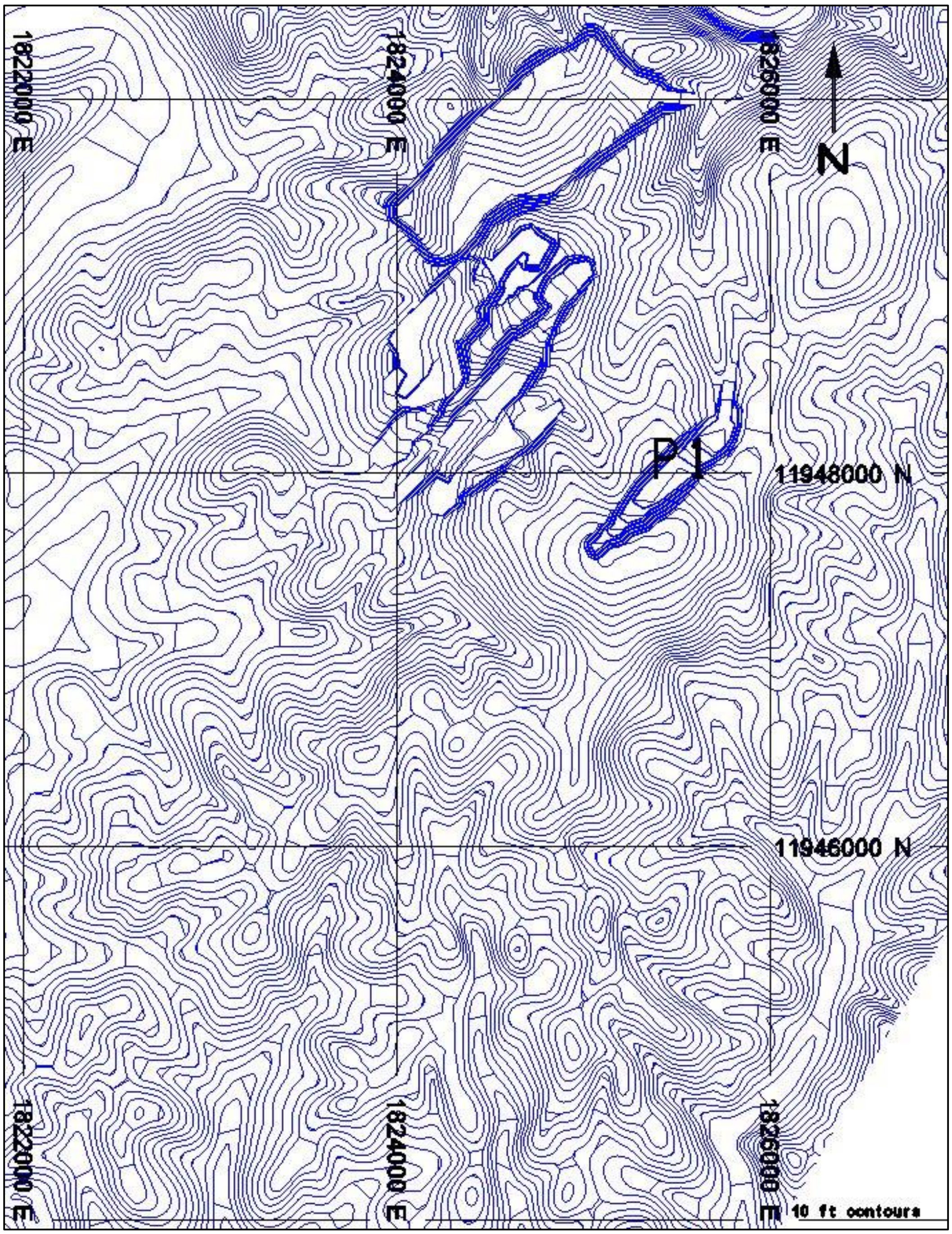
The Phase 4 design shown in Figure 16-5, represents a mining area between Phases 1 and 3. Waste and mill feed haul roads will be developed around or through Phase 1 and then tie into existing haul roads in order to send material to the east. Overburden will continue to be sent the southwest and will also connect with existing roads to minimize new road development.

Figure 16-5: Plan View of Phase 4



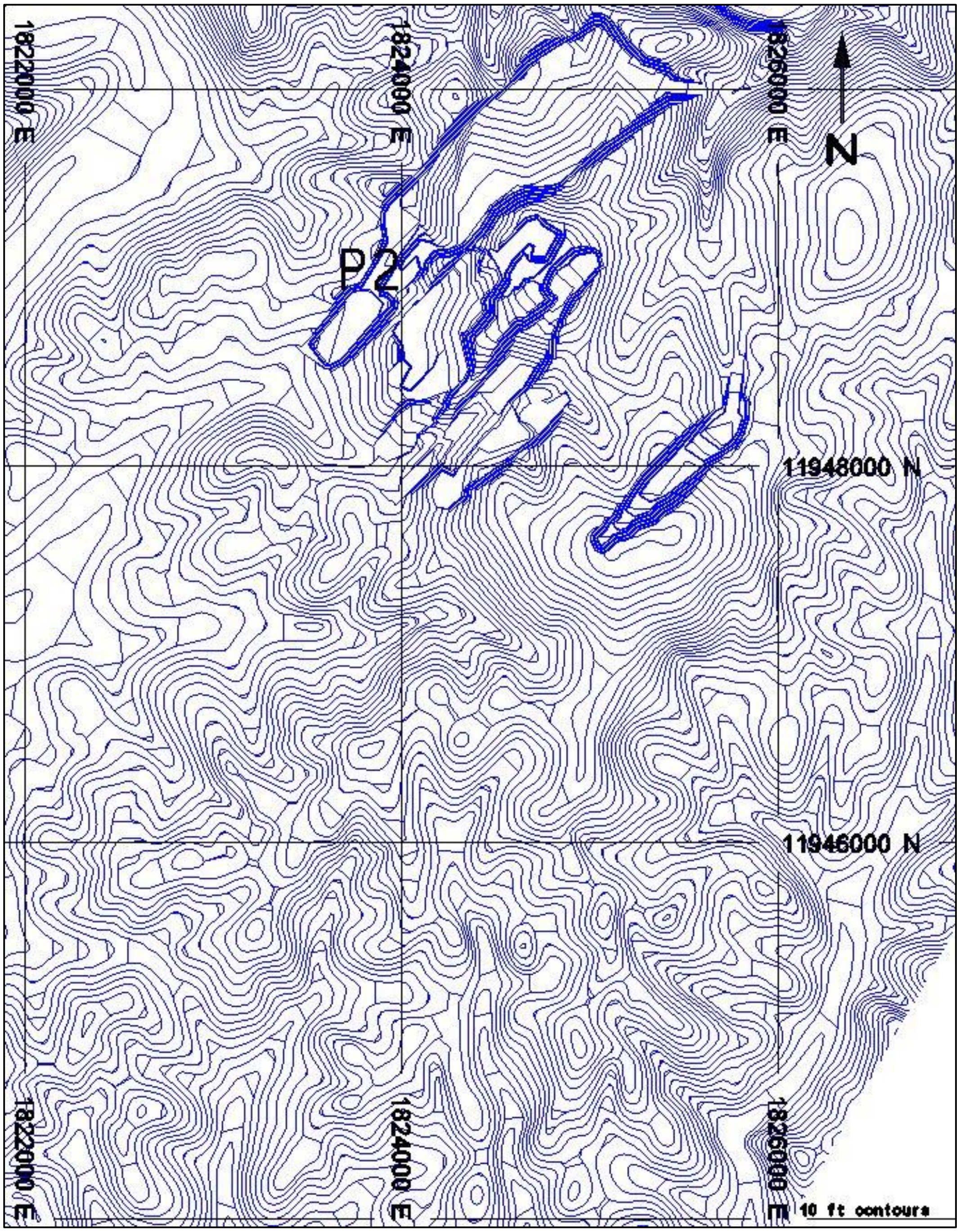
The Pit P1 design shown in Figure 16-6, is a small mining area at the east side of the project. It is located very close to the proposed plant and waste storage area, so will have short hauls. This pit was delayed in the mining sequence primarily due to the lack of drilling in this area. Overburden will be sent along a new haul road which extends south along the east side of the pit and then heads due west to the overburden storage area.

Figure 16-6: Plan View of Pit P1



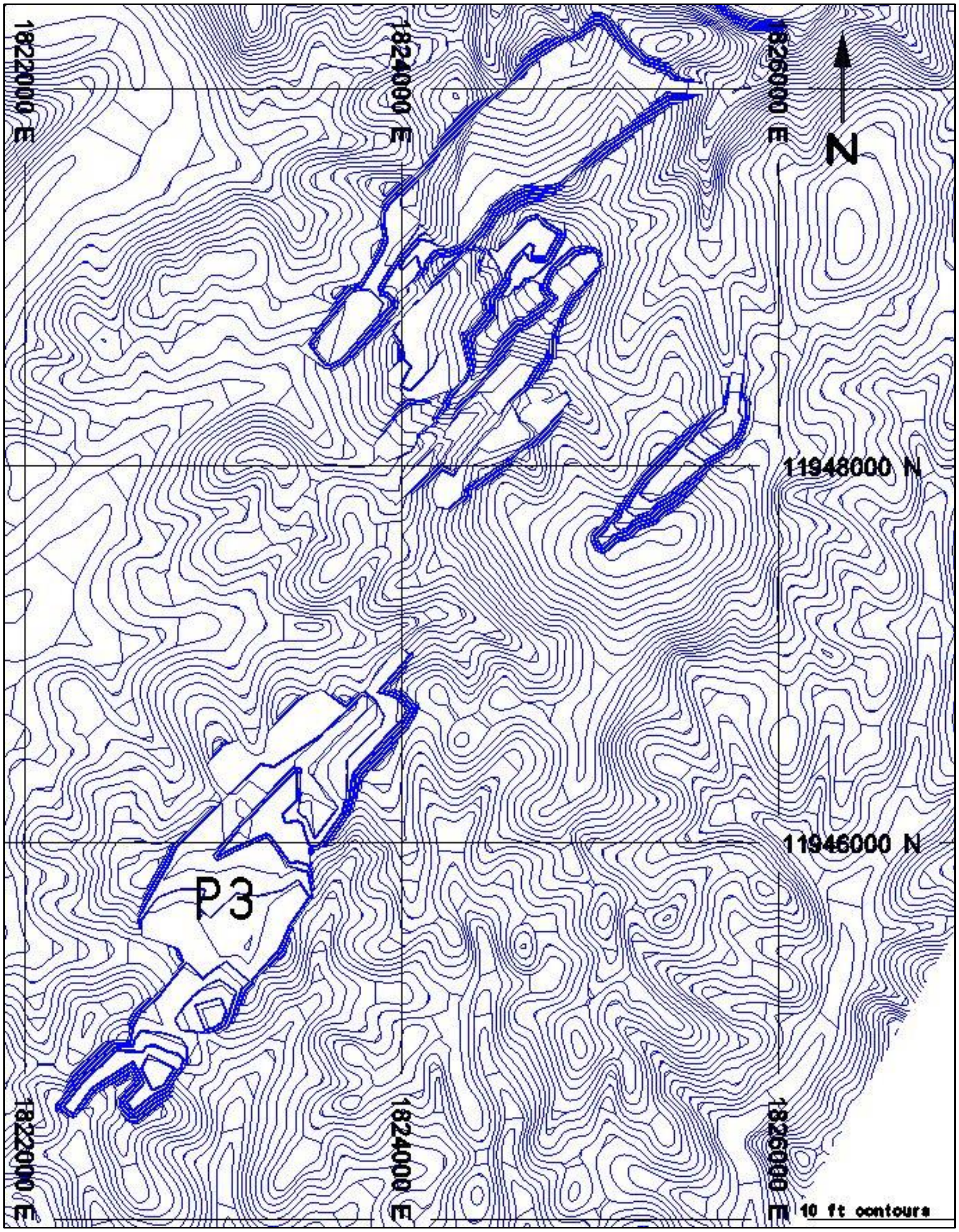
The Pit P2 design shown in Figure 16-7, is a lower grade southwest extension of Phase 3. Waste and mill feed haul routes will connect with existing haul roads from Phase 3. Overburden hauls will likewise connect with existing Phase 3 overburden haul roads for hauls to the southwest.

Figure 16-7: Plan View of Pit P2



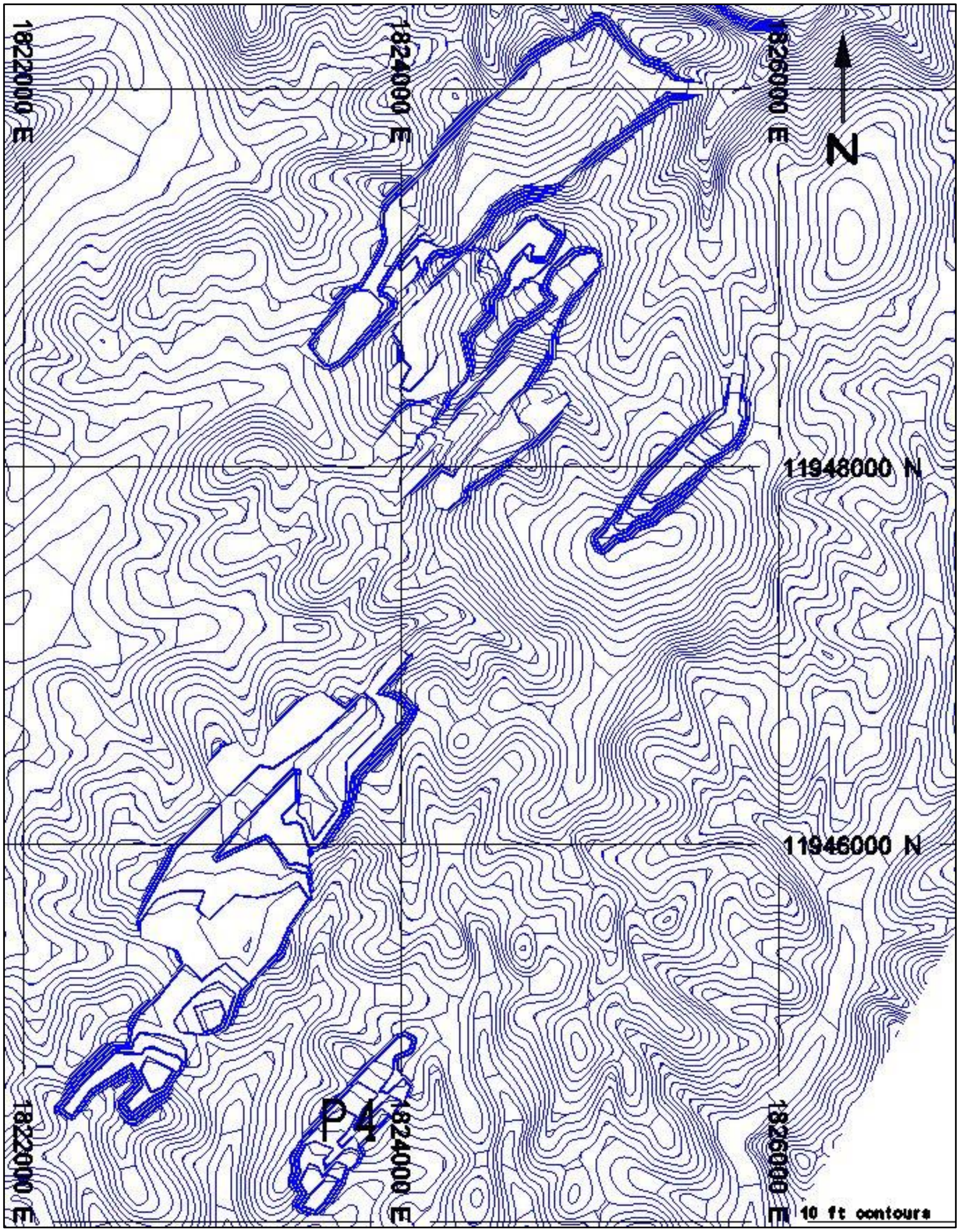
The Pit P3 design shown in Figure 16-8, is the first significant mining area in the southern half of the project. Mill feed and waste hauls will be directed to the northeast, while overburden hauls will be sent to the northwest.

Figure 16-8: Plan View of Pit P3



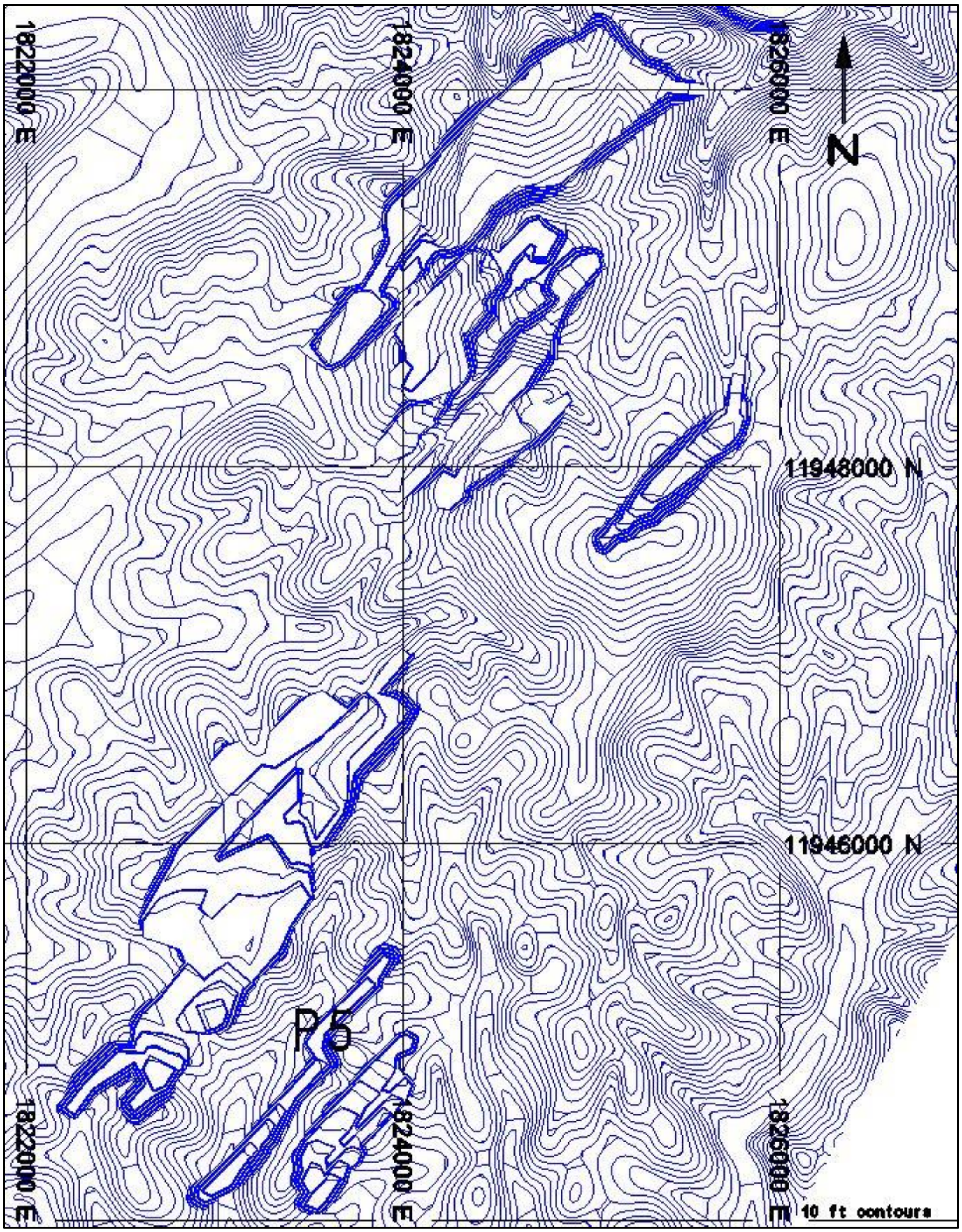
The Pit P4 design shown in Figure 16-9, is the furthest south of any mining area. Mill feed and waste hauls will be directed to the northeast, while overburden hauls will be sent to the northwest.

Figure 16-9: Plan View of Pit P4



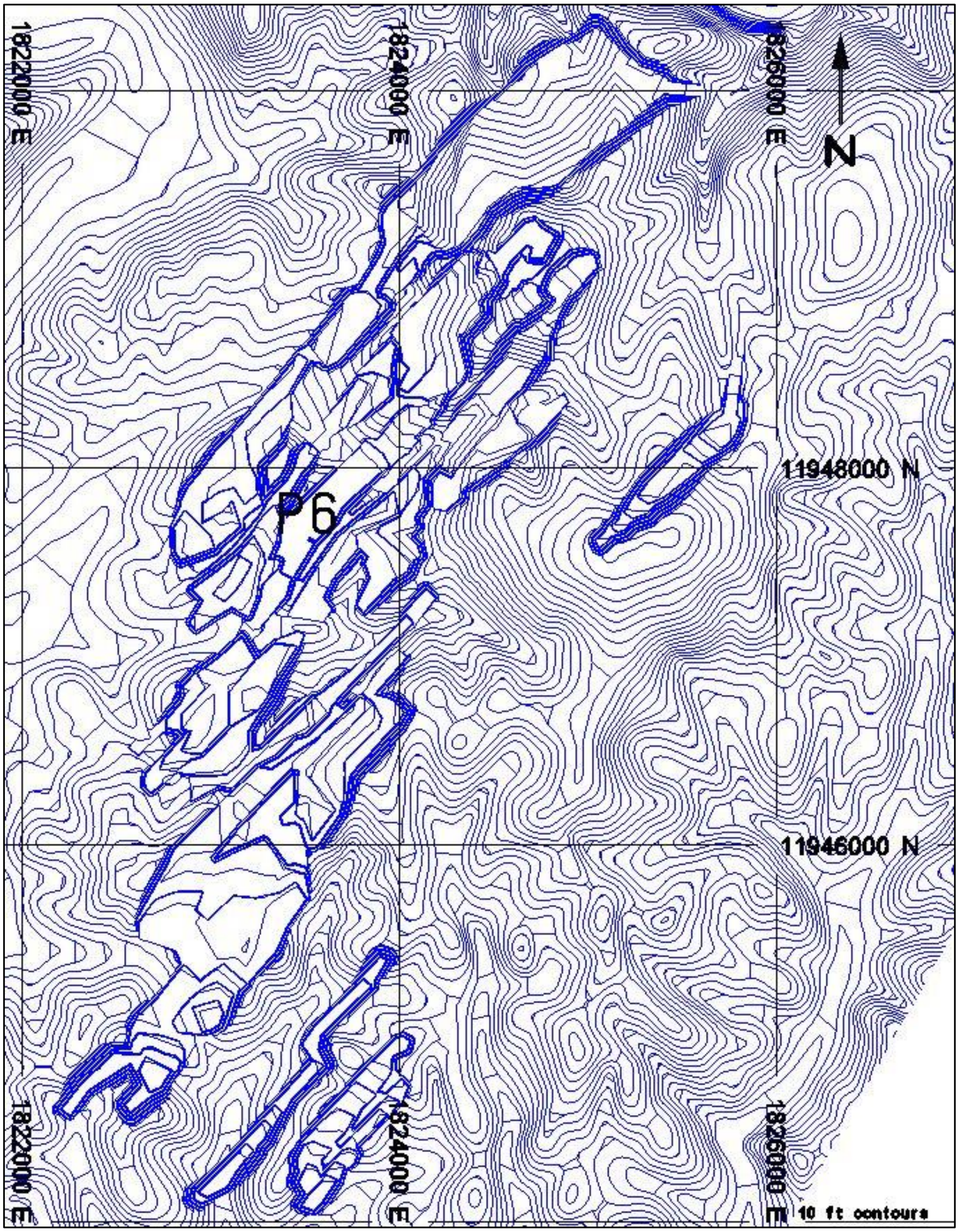
The Pit P5 design shown in Figure 16-10, is a narrow mining area which will need to be mined at the same time as other areas to maintain adequate mill feed. Mill feed and waste hauls will be directed to the northeast, while overburden hauls will be sent to the northwest.

Figure 16-10: Plan View of Pit P5



The Pit P6 design shown in Figure 16-11, effectively connects the north and south mining areas. It is generally a lower grade area that will most likely be mined near the end of mining. Mill feed and waste hauls will be directed to the northeast, while overburden hauls will be sent to the west.

Figure 16-11: Plan View of Pit P6



16.2.6 PEA Mine Schedules Examined

Three different schedules were developed with annual reporting periods so that various product throughput rates could be evaluated at a cash flow level. Schedule 1 had a product throughput rate target of 5,511 tons per year (tpa). Schedules 2 and 3 had their product throughput rates ramped up to 16,535 tpa and 27,558 tpa respectively. A common cut-off grade of 1% DCg (diluted graphitic carbon) was applied in all three schedules.

As Schedule 1 was mined at a much lower rate than the other schedules, its mining area was restricted to the initial 4 phases which contained high grade material. The plant feed tonnages by year for Schedule 1 are shown in Figure 16-12, and feed grade in Figure 16-13.

Figure 16-12: Plant Feed Tonnages (5511 tpa schedule)

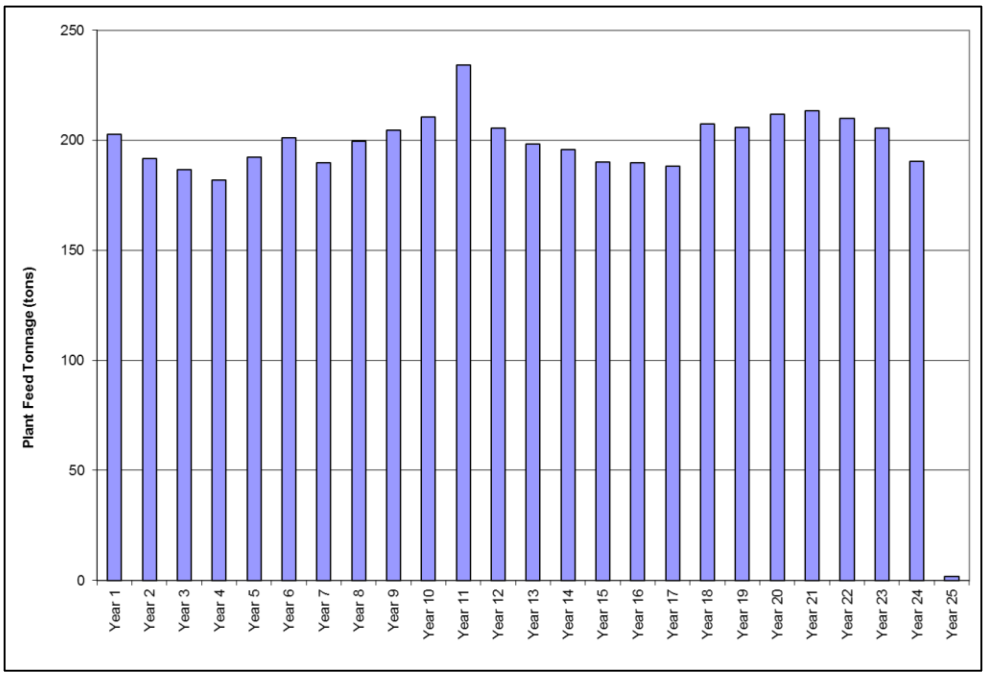
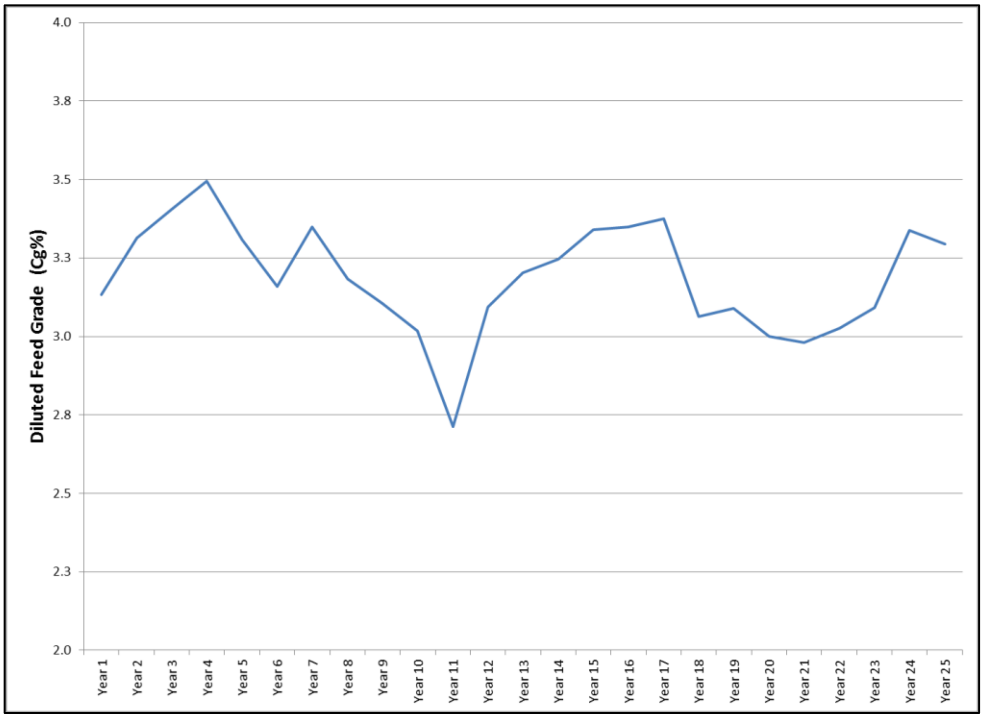
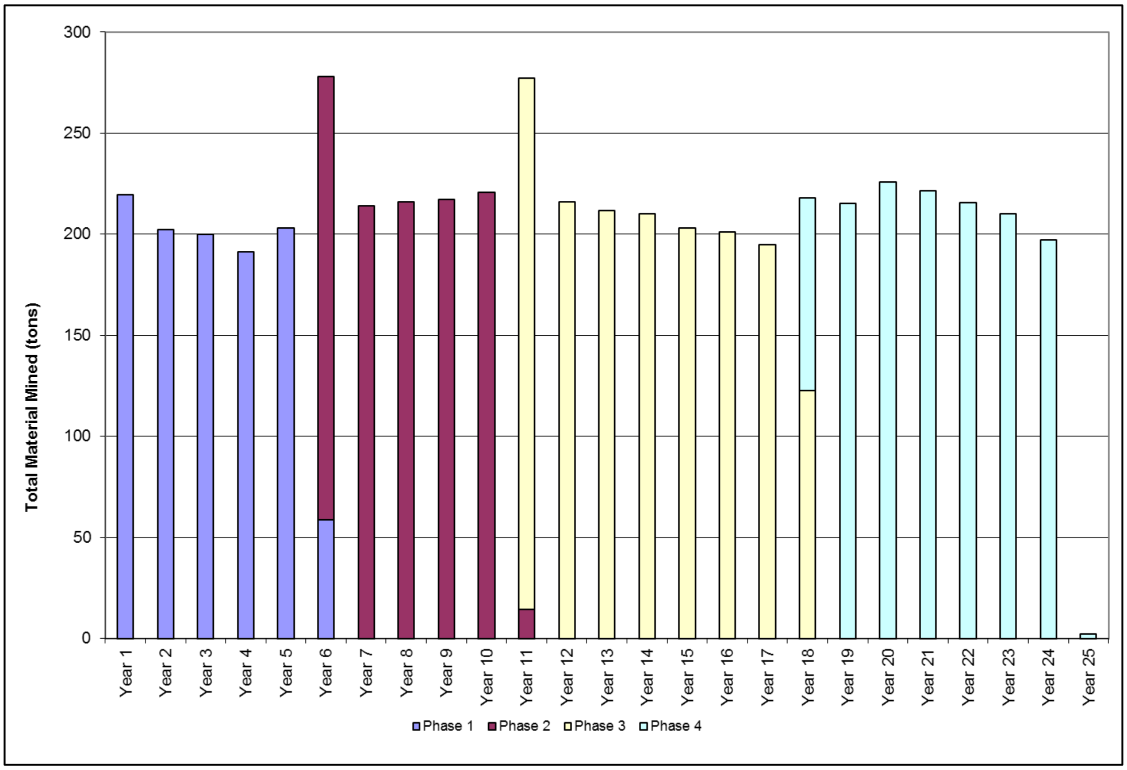


Figure 16-13: Plant Feed Grade (5511 tpa schedule)



Mined tonnages are consistently slightly more than 200 ktons per year for Schedule 1. As shown in Figure 16-14, Years 6 and 11 have higher mine tonnages due to initial waste stripping for new phases. Mined tonnages include mill feed, rock and overburden. Mining and processing are completed in Year 25 since no stockpiles have been utilized. A total of 4.4 million tons of Indicated mill feed grading 3.16% Cg (diluted) and 0.4 million tons of Inferred mill feed grading 3.35% Cg (diluted) will have been processed over 25 years in Schedule 1. Mined overburden and rock tonnages will be 233 ktons and 138 ktons respectively.

Figure 16-14: Total Material Mined by Phase (5511 tpa schedule)



Schedule 2 was mined at the same rate (5511tpa) as Schedule 1 for the initial five years, but transitions in Year 6 to a slightly higher rate, then reaches full plant product throughput in Year 7 at 16535 tpa. After the initial four phases are mined complete, the average grade of mill feed trends downward for the remainder of the schedule where Pits 1-6 are mined. The plant feed tonnages by year for Schedule 2 are shown in Figure 16-15, and feed grade in Figure 16-16.

Figure 16-15: Plant Feed Tonnages (16535 tpa schedule)

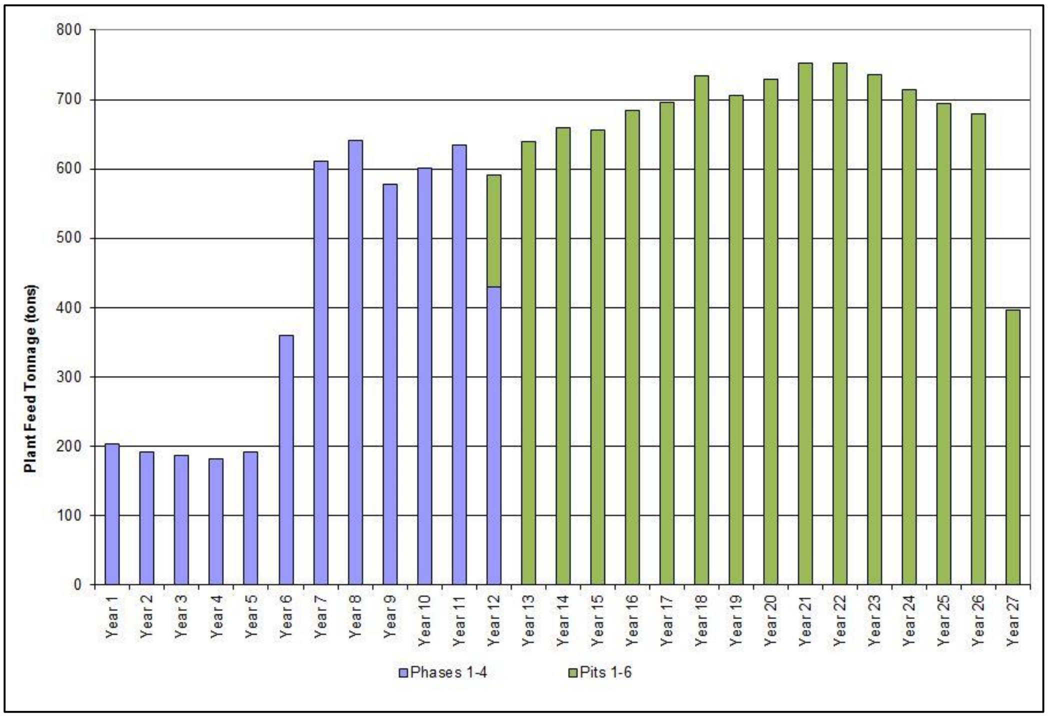
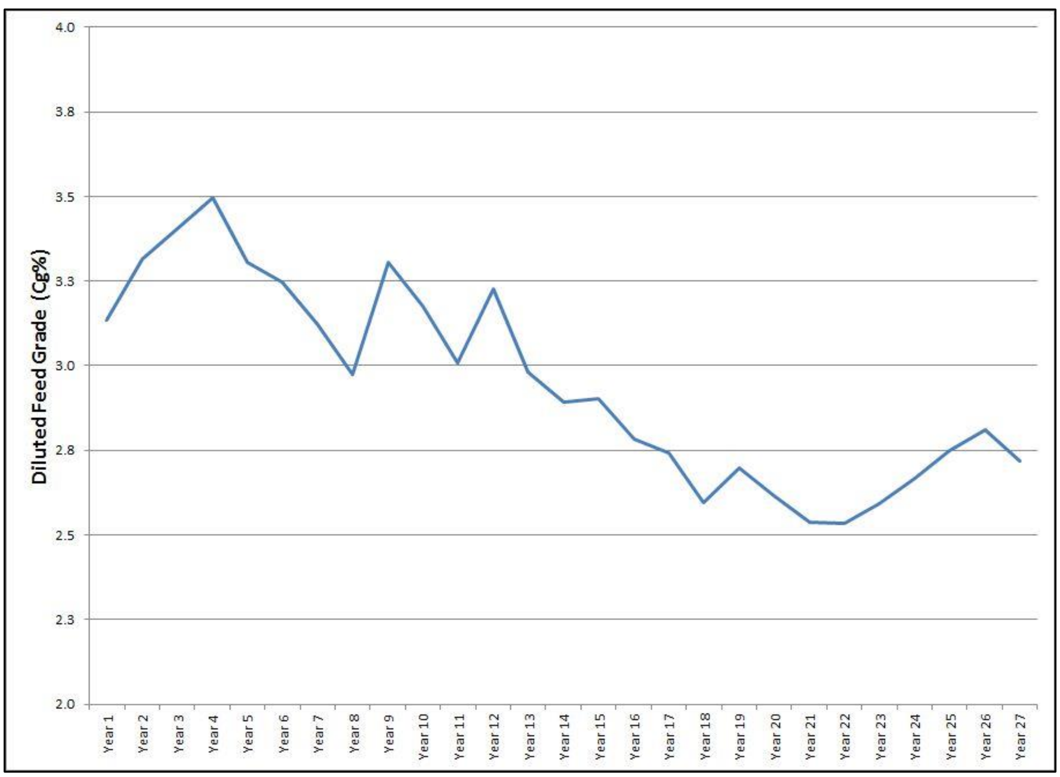
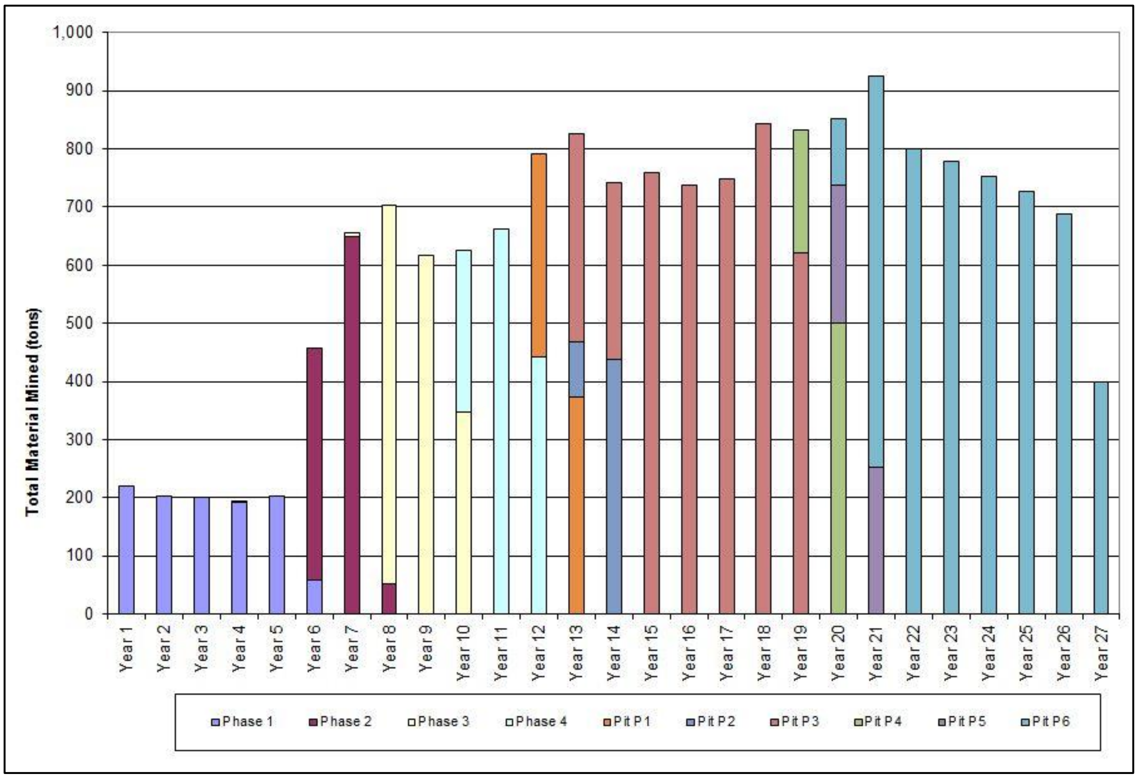


Figure 16-16: Plant Feed Grade (16535 tpa schedule)



Mined tonnages are increased to consistent levels between 700 and 800 ktons per year in Schedule 2. As shown in Figure 16-17, several years have higher mine tonnages than this range but are not excessive. Mined tonnages include mill feed, rock, and overburden. Mining and processing are completed in Year 27 since once again no stockpiles have been utilized. A total of 12.6 million tons of Indicated mill feed grading 2.85% Cg (diluted) and 2.6 million tons of Inferred mill feed grading 2.95% Cg (diluted) will be processed in Schedule 2. Mined overburden and rock tonnages will be 761 ktons and 981 ktons respectively.

Figure 16-17: Total Material Mined by Phase (16535 tpa schedule)



Schedule 3 was mined at the same rate as Schedule 2 until Year 9, but transitions in Year 10 to a slightly higher rate, then reaches full plant product throughput in Year 11 at 27558 tpa. The plant feed tonnages by year for Schedule 3 are shown in Figure 16-18, and feed grade in Figure 16-19. The plant feed varies by year due to grade variations. As the grade drops the feed tonnage increases to make the required final product.

Figure 16-18: Plant Feed Tonnages (27558 tpa schedule)

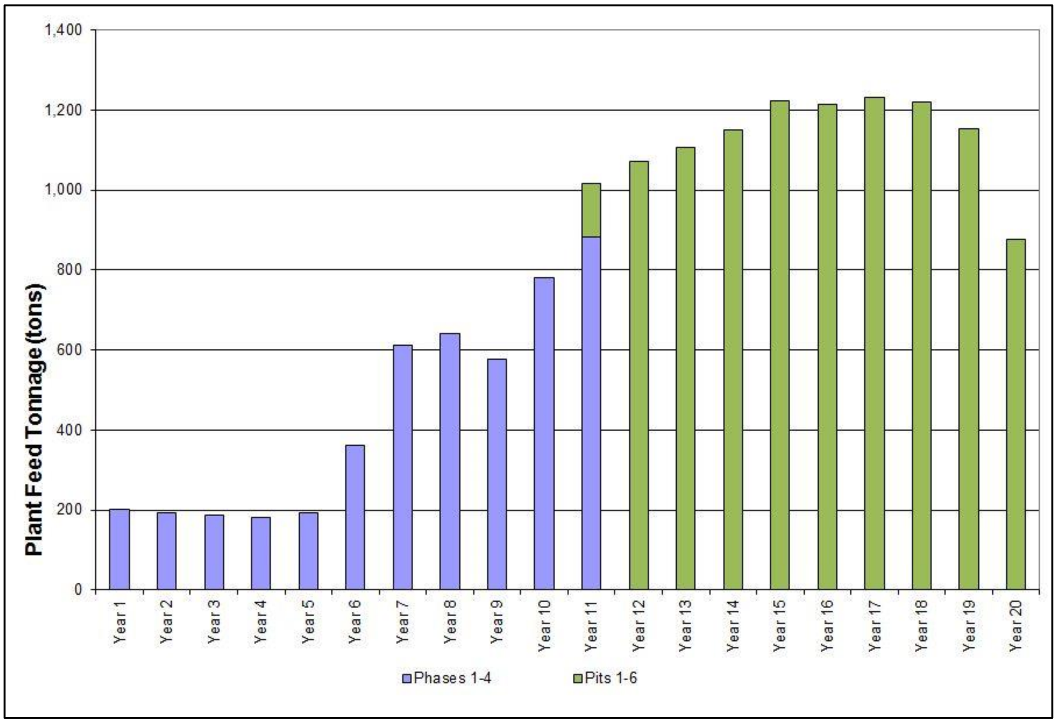
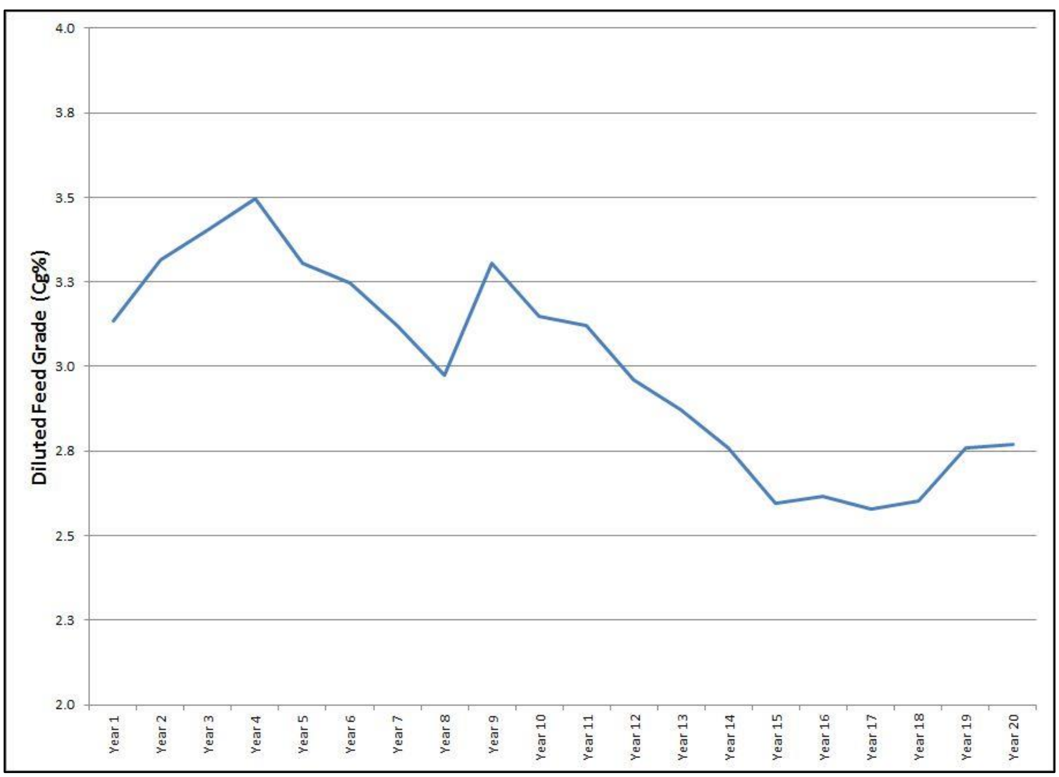
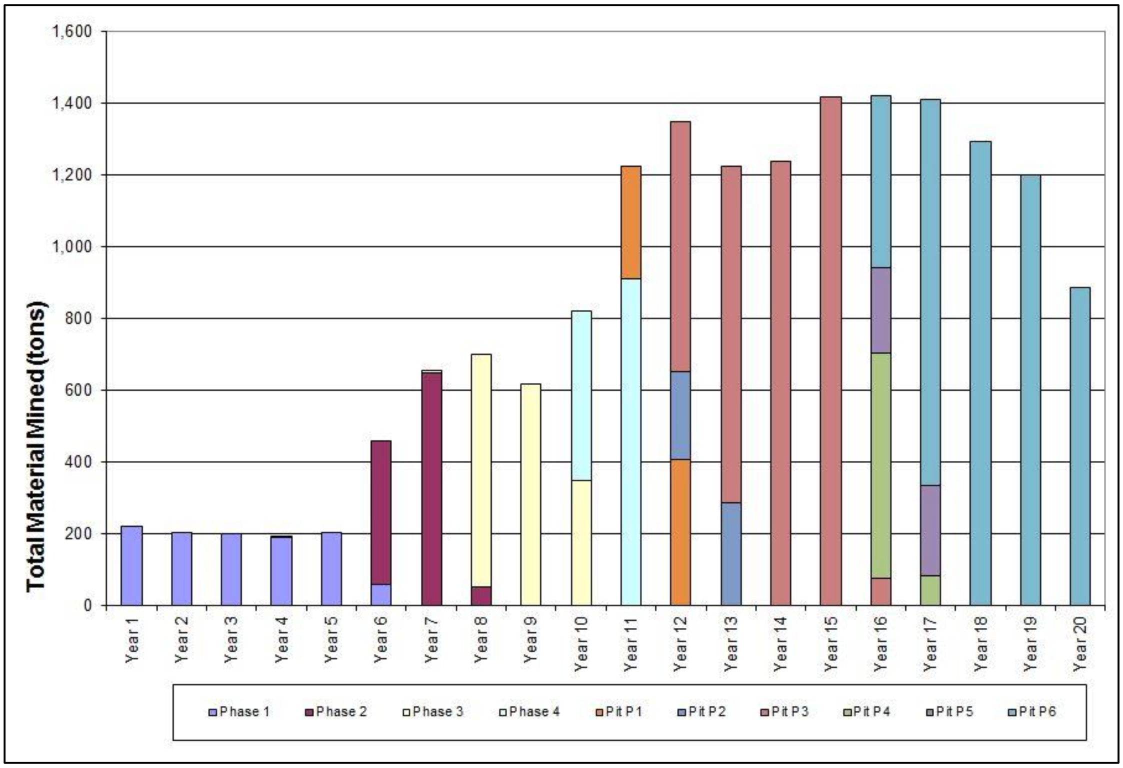


Figure 16-19: Plant Feed Grade (27558 tpa schedule)



Mined tonnages are increased to a peak of approximately 1.4 million tons per year in Schedule 3. As shown in Figure 16-20, the range of annual mine tonnages is fairly smooth at each production level. Mined tonnages include mill feed, rock and overburden. Mining and processing are completed in Year 20 with no stockpiles being utilized. Total material quantities and grades are the same as Schedule 2.

Figure 16-20: Total Material Mined by Phase (27558 tpa schedule)



16.2.7 *Mine Production Schedule*

The final schedule selected based on a cashflow analysis (to be discussed in Section 22) was Schedule 2. This was production at a rate of 5511tpa for the initial five years, but transitions in Year 6 to a slightly higher rate, then reaches full plant product throughput in Year 7 at 16535 tpa. After the initial four phases are mined complete, the average grade of mill feed trends downward for the remainder of the schedule where Pits 1-6 are mined.

16.2.8 *Mine Equipment Selection and Mining Method*

Mining equipment was selected based on a desire to minimize dilution and minimize flake breakage in the mining of the mill feed. This maximizes the value of the resulting product.

The ability to mine efficiently without the need for drill and blast operations also was a factor in the equipment selection. Equipment with sufficient weight and durability, even with the lower production requirements meant that larger equipment than normally expected was chosen.

Loading is expected to be completed with a 6.0 cubic yard excavator in the pit. The machine has the capability to dig in excess of 20 ft. efficiently and the weight sufficient to mine the oxide material. Assistance to the excavator will be with a 400+ horsepower dozer equipped with a triple shank ripper to loosen and push material to the excavator for loading.

Haulage is accomplished with the use of 45 ton articulated trucks. These trucks are better suited to work on the undulating oxidization footwall and poorer footing conditions in wet periods. They can also handle the road grades with ease and the softer overburden storage facility.

Support to the mine and plant will be provided by graders and smaller dozers to dress roads and waste/overburden storage facilities. A smaller backhoe will also be used to ensure the settling ponds are cleaned regularly to avoid mine material leaving the project site. Proper ditching of the roads will direct water to storage areas for settlement of particles.

A single shift of mining, Monday to Friday is expected to be required to provide sufficient feed to the process facility for the first two schedules. When mining increases to a rate sufficient for 27558 tpa of final product, an afternoon shift would be required Monday to Friday.

The process facility will operate around the clock to ensure it is capital efficient. For this to occur, the mine will stockpile material at the crusher and a 5.8 cubic yard loader will ensure the crusher is full at all times when the mine is idle. It will also be used to load dry stack material in the trucks for the backhaul of tailings to the dry stack storage facility.

For the operating cost determination, rock, overburden and mill feed haul profiles were developed for several benches in each mining area, then pro-rated to benches between them. All ex-pit haul profiles used a maximum of 8% gradient to ensure productive use of haul roads.

16.2.9 Waste Rock Management Facilities Design

The waste rock management facilities for Schedule 1 are shown in Figure 16-21. This figure is a snapshot of the pit and waste storage facilities in Year 25 for Schedule 1 (or Year 11 in the case of Schedule 3) just as active mining in phases 1 to 4 is complete. Figure 16-22 contains a snapshot of the pit and waste storage facilities after mining is complete for phases 1 to 4 and pits P1 to P6. This represents a maximum extent for the currently scheduled oxide mill feed material. For the purposes of this study, waste material from the Coosa deposit has been assumed to be non-acid generating.

Figure 16-21: Waste Rock Management Facility (after phases 1-4)

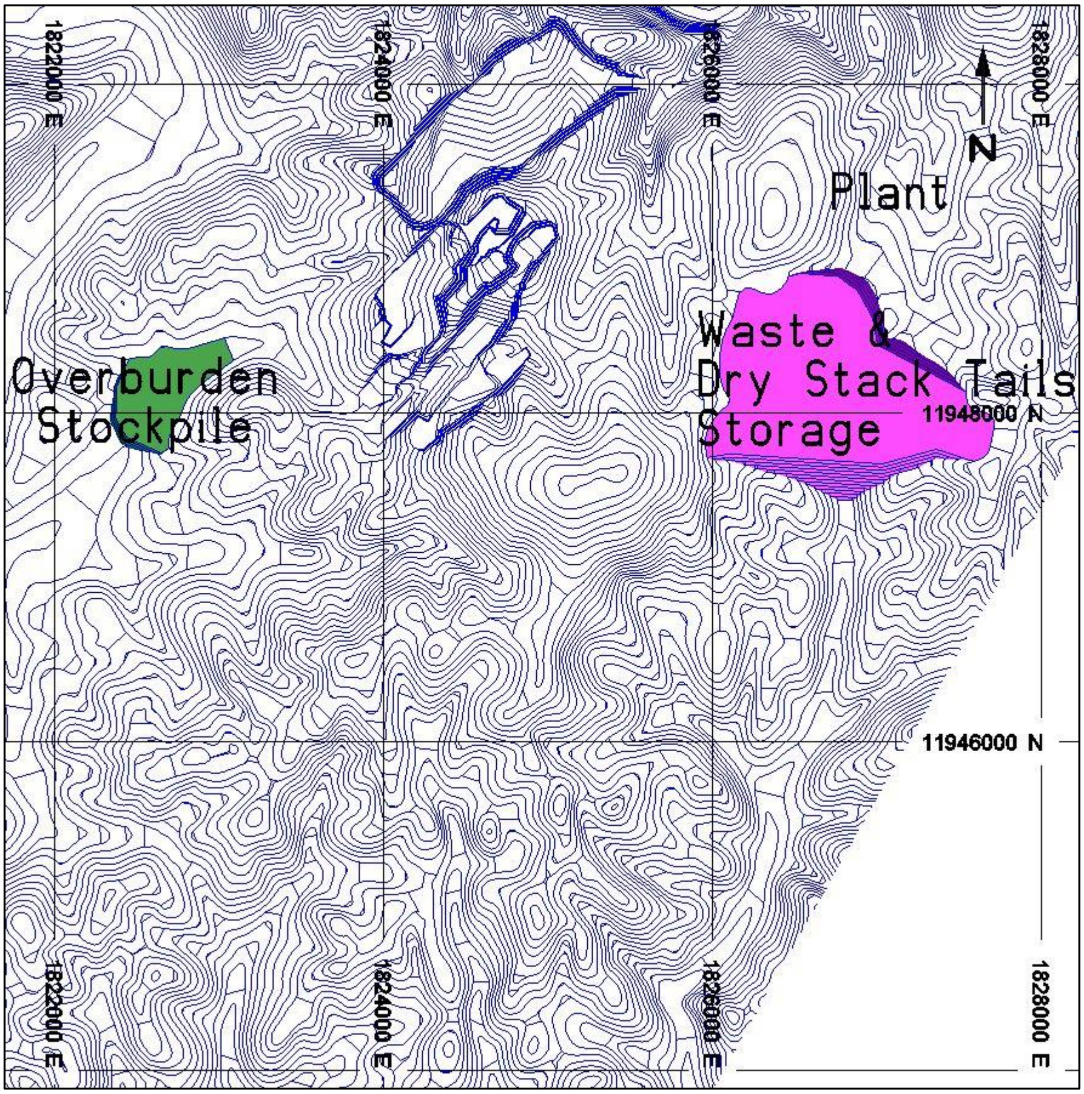
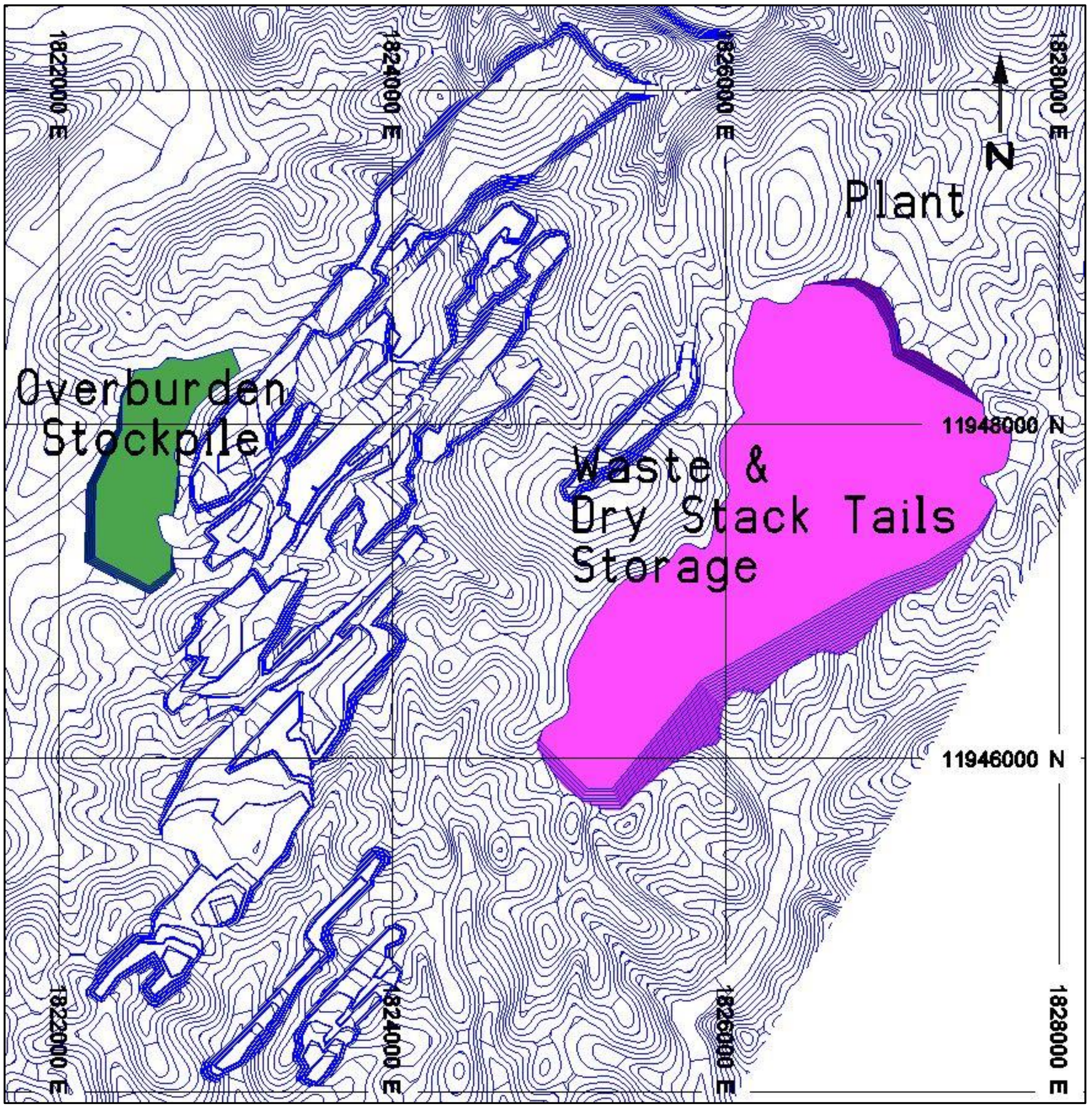


Figure 16-22: Waste Rock Management Facility (after all phases and pits)



17 RECOVERY METHODS

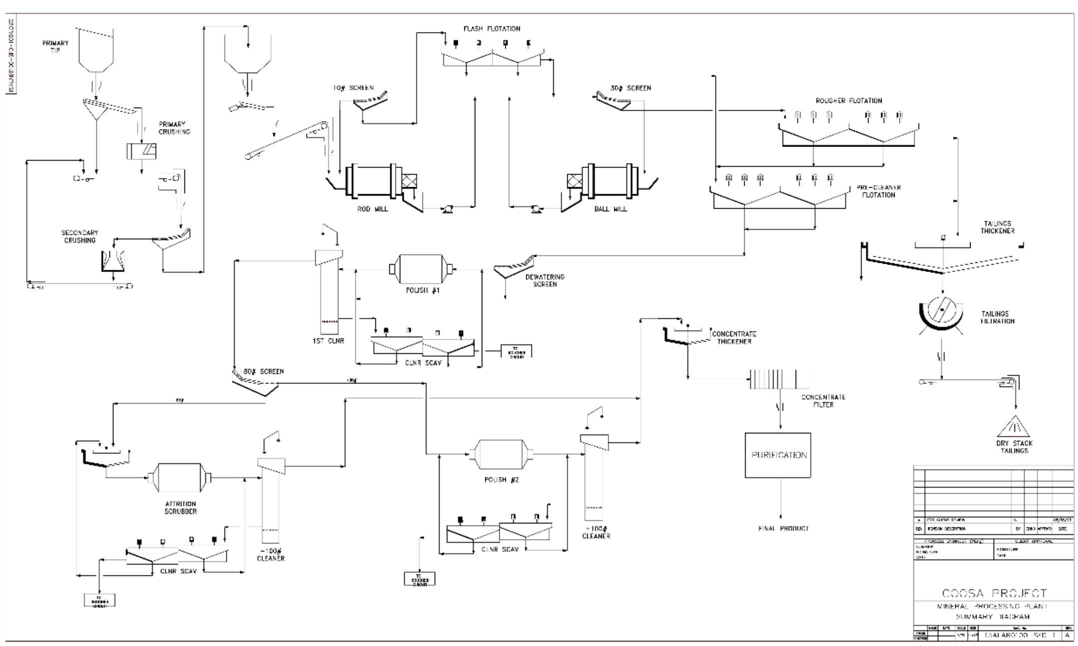
17.1 Introduction

The proposed method of graphite recovery from the Coosa deposit consists of conventional crushing and milling, followed by rougher and cleaner froth flotation. The cleaner flotation concentrate is then transported to a purification plant that includes micronization and spheronization prior to upgrading by thermal purification to produce the final product. This section describes the flowsheet, design criteria, and process description for a concentrator and purification plant to produce 5500 tons per annum of final (purified) product. Note that the process plant capital cost presented in Section 21 includes this throughput, as well as an upgrade scenario for future expansion to increase final production to the 16,000 tons per annum and 27.5 tons per annum cases used in the economic model.

17.2 Process Flowsheet

From the testwork conducted, a flowsheet was developed consisting of two-stage crushing, rod and ball mill grinding, froth flotation, concentrate dewatering, thermal purification, spheronization, and tailings thickening. A schematic of the proposed flowsheet is presented in Figure 17-1.

Figure 17-1: Flowsheet for the Coosa Processing Plant



17.3 Design Criteria

Based on the available data, a set of preliminary plant design criteria was developed which provides all the specific unit operation process details required for the equipment sizing and selection. A summary of some of the key criteria is presented in Table 17-1.

Table 17-1: Summary of Process Design Criteria

Parameter	Design Data
Graphite Head Grade	3.0 %
Plant Throughput	514 tpd
Crushing Circuit Availability	86.0 %
Grind/Float Circuit Availability	92.0 %
Rod Mill Feed Size, F_{80}	14 mm
Rod Mill Transfer Size, T_{100}	1.68 mm
Ball Mill Grind size, P_{80}	450 µm
Rougher Flotation Time	8 min
Mass Recovery to Rougher Concentrate	6.0 %
Pre-Cleaner Flotation Time	8 min
1st Cleaner Column Flux	2 t/hr/m2
1st Cleaner Mass Recovery	70 %
1st Cleaner Concentrate +177µm	45 %
Mass Recovery to Final Flotation Concentrate	3.0%
Cg Recovery to Final Flotation Concentrate	97.0 %
Cg Grade of Final Flotation Concentrate	97.0 %
Tailings Moisture Content	18.0 %
Process Plant Fresh Water Consumption	7.0 cu.ft/t
Process Plant Power Consumption	23.2 kW/t

17.4 Process Description - Concentrator

This section describes the parameters used to design a new graphite concentrator for the Coosa Project near Montgomery, Alabama, for the base-case operating scenario of 5,500 tons per annum of purified graphite product.

Note: the fundamental design criteria for the processes described herein have been developed using limited testwork and should be considered preliminary. With this in mind, AGP has taken a conservative approach to plant design, using commonly available technology and flowsheet configuration.

17.4.1 Summary

The Coosa concentrator is designed for a nominal 16,500 tons per month run of mine feed.

Mine haul trucks will tip into a surge bin feeding a primary jaw crusher designed for 86% availability. The primary crushed mill feed will be conveyed to a 5/8" (15mm) sizing screen, with the oversize reporting to the secondary crusher and the undersize fed forward to the rod mill circuit.

A 150-ton fine feed bin provides surge capacity for the circuit and helps to ensure a constant feed rate to the rod mill, and operates in closed circuit with a 10 mesh scalping screen. Undersize from the screen is fed to a flash flotation stage, with the flash cell tailing pulp gravitating to the ball mill classification screen. The screen cuts at a P_{80} of 450µm, with the oversize reporting to the mill and the undersize going to the feed box in rougher flotation.

The rougher flotation stage consists of six conventional ("trough") cells in series. Each cell will have independent air flow control. The rougher concentrate is combined with the flash concentrate and pumped to a pre-cleaner flotation circuit consisting of five trough cells in series.

The pre-cleaner concentrate is dewatered at 230 mesh, with the oversize reporting to the first of two polishing scrubbers. The scrubber discharge is combined with the screen undersize (solution) to serve as feed to the first cleaner column float cell. Two stages of scavenging on the first cleaner tail return concentrate to the column feed and screen feed, respectively.

First cleaner flotation concentrate is screened at 80 mesh (177µm), with the oversize reporting to the second polishing scrubber ahead of column flotation. The screen undersize is thickened prior to attrition scrubbing and column flotation. Final cleaner concentrate from both circuits is combined, thickened and press filtered to serve as feed for the purification step.

Flotation tailings from the rougher and pre-cleaner circuits are thickened, filtered, and transported to the tailings management facility.

Reagents are stored, mixed, and distributed from a central reagents area. Frother, collector, and lime are pumped from the reagents area to the flotation section using peristaltic reagent pumps to accurately dose the process.

17.4.2 Crushing: Area 100

Note: the information gathered from ongoing pilot work suggests that run of mine material from the Coosa deposit will be extremely friable and will contain large quantities of fine material. However, for the purposes of this preliminary study, AGP has elected to include a more conservative design that includes a jaw crusher for oversize material.

Ore will be delivered to the primary tip by 20t haul trucks at a frequency averaging two trucks per hour. Peak delivery rate is assumed to be 60 stph. Feed is discharged directly into a rail-lined, 30-ton surge bin.

A vibrating grizzly feeder at the bottom of the surge bin discharges +60 mm oversize into the primary crusher feed chute. Grizzly undersize drops onto the classification screen feed conveyor. The primary crusher consists of a 15" x 24" jaw crusher with a closed side setting of 50 mm, and can accept a top size of 400 mm. Crushed feed discharges onto a sacrificial conveyor, which in turn discharges onto the classification screen feed conveyor. The screen feed conveyor is equipped with an overhead magnet for removing tramp metal ahead of the secondary crusher.

A 4 ft. x 8 ft. vibrating inclined screen is used to classify the primary crushed feed and close the circuit around the secondary crusher. The screen deck consists of 19 mm slots with the screen undersize chute feeding the fine feed bin conveyor. The fine feed bin conveyor is equipped with a weight-o-meter for recording crushing circuit production. Oversize from the classification screen is fed by a lined chute into the secondary crusher.

The secondary crusher consists of a standard roll crusher with a closed side setting of 15 mm. Crushed product discharges via a chute onto the recycle conveyor which returns the material to the classification screen feed conveyor.

The crushing circuit is expected to run at 86% availability, seven days per week. A dust collection system is included in the classification screen and secondary crushing area.

17.4.3 Grinding: Area 200

The milling circuit and all subsequent areas of the plant operate at 92% availability on a 24/7 schedule. Crushed run of mine material is stored in a 150-ton fine feed bin with approximately six hours operating capacity. An overflow chute at the top of the bin allows overflow to a stockpile with an additional six hours of capacity. The stockpile material can be reclaimed to the fine feed bin conveyor by front end loader, as needed.

A vibrating pan feeder draws material from the bottom of the fine feed bin onto the mill feed conveyor. The conveyor is equipped with a weight-o-meter to measure mass flow and allow for control of the mill feed rate.

The primary grinding mill consists of a 6 ft. diameter by 10 ft. long overflow discharge rod mill with a 120 kW, single pinion drive. The discharge end of the mill is fitted with a trommel screen with 10 mm openings. Oversize tramp material, woodchips, etc. drop through a chute into a drum or skip. The material passing through the trommel is collected in a chute and fed to the screen pump feed box.

The mill discharge is diluted with process water and is then pumped to a vibrating, inclined classification screen with 10# (1.68mm) openings. The screen oversize is returned to the rod mill feed chute, whereas the undersize is pumped forward to the flash flotation step.

Flash flotation consists of two 100 ft^3 trough flotation cells in series. Concentrate from the flash circuit combines with the rougher concentrate as feed to the pre-cleaner circuit. Flash tails are fed by gravity to the discharge pump box in the ball mill circuit.

The secondary grinding stage consists of a 5 ft. diameter by 8 ft. long overflow discharge ball mill with a 60 kW, single pinion drive. The discharge end of the mill is fitted with a trommel screen with 6 mm openings. Oversize tramp material, woodchips, etc. drop through a chute into a drum or skip. The material passing through the trommel is collected in a chute and fed to the screen pump feed box.

Ball mill discharge and flash tailings are pumped to an inclined, 6 ft. x 10 ft., classification screen with 595µm (30#) openings. The oversize gravitates to the feed chute of the ball mill, whereas the undersize is pumped to the feed box in the rougher flotation circuit.

Spillage contained in the grinding area is pumped to the mill discharge sump for re-treatment. Ball mill grinding media is delivered to the plant in bulk and is stored in the ball mill ball bunker. The ball bunker is serviced by a crawl and electric hoist arrangement allowing balls to be lifted into a kibble using the ball loading magnet and tipped into the mill via a ball loading chute.

17.4.4 Rougher and Pre-Cleaner Flotation: Area 300

Ball mill classification screen underflow is pumped to the feed box of a bank of six DR30 type rougher flotation cells with a combined residence time of just over ten minutes. Flotation air to each cell is supplied by flotation blowers via a low pressure manifold and is manually controlled. Pulp level is maintained by modulating dart valves.

Rougher concentrate is collected in a common launder and flows by gravity to the rougher concentrate froth pump, which pumps to the pre-cleaner feed box. The pre-cleaner float bank consists of five DR18 type flotation cells with twelve minutes of residence time. The concentrate collected from the pre-cleaner float is collected in a common launder and flows by gravity to the same froth pump as the flash concentrate to be pumped to the cleaner circuit. Rougher and pre-cleaner tailings are the final tailings from the flotation circuit and report to the tailings pump feed box and are then pumped to the tailings dewatering circuit.

Spillage in the rougher section is collected in a common sump and pumped back into the first rougher cell using a submersible spillage pump.

17.4.5 Cleaner Flotation: Area 320

Feed to the cleaner flotation circuit, in the form of pre-cleaner concentrate, is fed to a 4 ft. x 8 ft. vibrating dewatering screen with 230 mesh (63µm) panels. Dewatered solids from screen deck fed to polishing scrubber #1, a 4 ft. x 13 ft. tire-mounted drum scrubber with a 15kW drive. The scrubber discharge combines with dewatering screen underflow to serve as feed to the first of three cleaner flotation columns. Each column is 3 ft. in diameter by 20 ft. high with PLC level control and froth washing.

Concentrate from the first cleaner column is pumped to an 80-mesh (177µm) sizing screen. The screen oversize gravitates to a second polishing scrubber in the +80-mesh cleaning circuit. Tailings from the first cleaner column go to conventional scavenging circuit consisting of four DR18 trough cells in series. Concentrate from the first two scavenging cells is returned to the column feed, whereas the concentrate from the last two cells reports to the dewatering screen.

The +80 mesh cleaning circuit has the same flowsheet as the first cleaner circuit, except there is no dewatering screen. All of the classification screen oversize is fed to the second polishing scrubber ahead of the +80 mesh cleaning column. The +80 mesh cleaner column concentrate is pumped to the concentrate thickener, while the column tails report to the scavenger circuit, with concentrates returned to the column feed and scrubber feed.

The -80 mesh cleaning circuit has the same flowsheet as the first cleaner circuit, except that there is a thickener in place of the dewatering screen. Thickener underflow is fed to an attrition scrubber ahead of the -80 mesh cleaning column. -80 mesh cleaner column concentrate is combined with the +80 mesh concentrate and pumped to the concentrate thickener, while the column tails report to the scavenger circuit, with scavenger concentrates returned to the thickener feed and scrubber feed.

Cleaner scavenger tailings are pumped to the rougher circuit, or the tailings thickener. The cleaner area spillage is collected in bermed areas and directed into the cleaner area spillage pump, which pumps back to the first cleaner feed box.

17.4.6 Concentrate Dewatering: Area 400

Final graphite concentrate is pumped to the concentrate thickener sampling box and sampler before entering the concentrate thickener for dewatering. The thickener is equipped with rake lift, bed level detection, and bed mass monitoring. Thickener overflow gravitates to the spray water tank for recycling, while the thickener underflow is withdrawn from the cone by a centrifugal underflow pump and pumped forward to the concentrate storage tank, or recycled to the thickener feed if of insufficient density.

The concentrate is pumped from the mechanically agitated storage tank to a horizontal pressure filter for dewatering. The press will contain fifteen 1000 mm x 1000 mm plates, and have a filtration volume of 448 L. Filtrate from the press is recycled back to the concentrate thickener feed launder.

Concentrate filter cake is discharged from the press via two cake discharge chutes onto the cake transfer belt which transfers cake to the concentrate storage shed where it is stored prior to purification. Concentrate dewatering area spillage is recovered by pumping back to the concentrate thickener.

17.4.7 Tailings Dewatering: Area 450

The combined rougher/pre-cleaner tailings pumped from the flotation area discharge into a sampling launder on top of a 10 m diameter high-rate thickener. The thickener is equipped with rake lift, bed level detection, and bed mass monitoring. Thickener overflow gravitates to the process water tank, while the thickener underflow is withdrawn from the cone by a centrifugal underflow pump and pumped forward to the tailings filter.

The tailings filter consists of a continuously operated vacuum belt filter that produces a tailings product at ~18% moisture. Filtrate is returned to the feed launder of the tailings thickener. The filter cake discharges from the press via a chute onto the tailings conveyor which transfers the tailings to a storage pile within a three sided concrete walled shed. A front end loader is used to load a 30-ton dump truck to transport the tailings to the dry stack area at an average rate of approximately one truckload every hour.

17.4.8 Services: Area 500

Process water is stored in a 200 m^3 tank and is distributed to the plant by a process water pump. Plant hosing/flushing water is provided by the hose-down water supply pumps.

The process water tank is also used to feed the diesel powered fire water pump from a separate (lower) offtake, thus guaranteeing availability.

Clean water is piped into the plant from wells and stored in the plant clean water tank. From the storage tank, water is pumped around the plant for use as reagent mixing water, slurry pump gland seal water, and as required for mill lubrication system cooling.

Plant and instrument air is provided by two compressors. Air quality is maintained by a filter. Instrument air is dried using a refrigeration drier. An air receiver is provided for compressed and instrument air lines, to allow for surges in demand.

Low pressure air is supplied to the flotation plant by two separate blowers. The blowers are fixed speed, with manifold pressure controlled by a modulating valve on an exhaust line.

17.4.9 Reagents: Area 600

PH Modifier – Calcium Hydroxide

Lime (calcium hydroxide) is delivered to the plant in 1-ton bulk bags and loaded into the lime hopper. Dry lime is metered from the hopper into the agitated mixing tank by a screw feeder and mixing plate. Mixed lime slurry at 10% solids is pumped to an agitated dosing tank. A circulation pump supplies lime to the flotation circuit via a ring main.

Collector – Fuel Oil

Diesel fuel is delivered to the plant storage tank in bulk. Peristaltic hose pumps meter the reagent directly from the storage tank to several addition points throughout the plant.

Frother - MIBC

Liquid Methyl Isobutyl Carbinol (MIBC) is delivered to site in $1m^3$ totes. As delivered (100% strength) MIBC is pumped directly to the dosing points by dedicated peristaltic pumps.

Flocculant – Magnafloc 10

Flocculant powder is delivered to site in one ton bags and stored in the reagent storage area. Bags are lifted by the reagent area crane and added to the flocculant powder hopper. Powder is withdrawn by the flocculant screw feeder and blown through a venturi to a wetting head located on top of the mechanically agitated mixing tank.

From the mixing tank, mixed flocculant can be fed forward to the storage tanks, or recycled back into the mixing tank to aid mixing. From the storage tank, flocculant is pumped directly to the tailings and concentrate thickeners.

Reagent spillage is pumped to the tailings thickener, or stored in totes for disposal.

17.5 Process Description - Purification Plant

The purification plant is a secondary processing stage that enables the production of speciality graphite products and will include micronization, spheronization, purification, and coating processes. The process will not include the use of hazardous reagents such as hydrofluoric acid (HF) or sodium hydroxide (NaOH), but rather is based on a new continuous carbo-chlorination process with internal recycling of the chlorine gas and capture of the graphite impurities as solid oxides. This process has been defined by preliminary batch testwork and order of magnitude capital and operating costs for a plant producing 5500 tons per annum of purified graphite have also been estimated (Section 21).

Metallurgical testwork (described in Section 13) has determined that a purified product of over 99.95% C(g) is possible using carbo-chlorination.

Note: the fundamental design criteria for the purification processes described herein have been developed using limited testwork and should be considered preliminary. The testwork was carried out in 2015 by a reputable North American Laboratory with a well-established track record in pyrometallurgical process development. For reasons of commercial confidentiality, Alabama Graphite Corp. does not wish to disclose the identity of the metallurgical laboratory.

17.5.1 General

Current plans allow for the construction and operation of the purification plant within one of the many vacant industrial facilities near the project site.

The PEA financial models and production schedules assume a capacity expansion to 16,500 tons per annum of purified graphite in Year 7 To accommodate the expansion, AGP has assumed that a new 11,000 tons per annum purification plant would be built nearby (Phase 2), with the option to double capacity to 22,000 tons per annum at a later stage (Phase 3).

The following process descriptions consider the 5,500 tons per annum plant only.

17.5.2 Raw Material Handling & Storage

Filter cake from the concentrator will be delivered in bulk containers or one ton bags. The cake, containing approximately 10% moisture, would be transferred using a container tilter or bag ripper and various receiving vessels and conveyors to feed the crude graphite storage silo. This silo is sized to provide approximately seven days' storage capacity at nominal throughput rates.

The damp cake is transported to a second storage silo (two days' capacity) ahead of the micronization mills via a heat exchanger unit designed to reduce moisture levels using recycled gas from the purification furnaces.

17.5.3 Micronization and Spheronization

The part-dried flotation concentrate will be conveyed from the storage silo into the micronization and spheronizing lines, where the graphite will be finely milled, classified, and shaped to produce a blend of spherical and micronized graphite products. The milling process also assists in the drying of product prior to purification.

Spheronized and micronized products will be conveyed to the purification area storage/feed bins.

17.5.4 Purification and Coating

Several purification reactors will be installed in parallel, allowing a nominal 60% on-time for individual units. Spheronized graphite concentrate will be fed from the various feed bins into the reactors so as to effect the carbo-chlorination process. The reactors will be heated electrically to the treatment process temperature of approximately 1400 °C.

The reactors will be fed and emptied on a continuous basis and hot gas would be recycled through appropriate circuits (described below). Coarse and fine material would be purified separately.

Testwork conducted to date has shown that purified products of up to 99.95% C(g) are achievable using the carbo-chlorination process.

The purified fines would be transferred to storage hoppers in the products area. The coarse purified product would be passed from the purification vessel discharge directly into to a coating vessel, which allows the particles to be coated with a thin layer of carbon, thereby reducing surface area; required for durability in Li-ion anode production. Coated, spheronized, purified graphite (CSPG) would be discharged from the coating vessel and transferred to the product storage hoppers for packaging and transportation.

17.5.5 Product Storage & Packaging

Two products are presently anticipated, and are discussed in Section 19.0 in more detail.

- CSPG – for Li-ion battery anodes (high purity)
- Micronized, high purity graphite.

These speciality products will be transferred from the shaping plant, and stored in hoppers prior to bagging. A vendor supplied bagging plant will package the final products in bulk bags. Weigh scales would be provided for accounting purposes.

17.5.6 Gas Handling & Auxiliaries

The hot chlorine off-gas exits the purification and coating equipment and is ducted to a gas handling circuit that will:

- remove the chlorinated impurities as solid oxides
- vent and scrub excess gasses
- recycle the chlorine (assumed 90% efficiency)
- recycle excess energy via heat exchangers within the graphite feed circuit

An area dedicated to storage of the required chemicals (chlorine, sulphuric acid, toluene, oxygen) will be provided outside the plant.

Blowers for the gas handling system would be located in a dedicated area within the plant, alongside compressors for general plant use.

18 PROJECT INFRASTRUCTURE

18.1 Infrastructure and Site Layout

The Coosa Graphite Project infrastructure consists of the following:

- open pits
- process plant, mobile equipment and maintenance shops
- office/administration area
- waste rock management facility
- dry Stack tailings management facility
- electrical substation

This infrastructure is required to support the final facility generating 16,500 tons per annum (15,000 tonnes per annum) of spherical and micronized final product.

18.1.1 Mine/Mill Site Operations

The mill and processing facility is to be located to the northeast of the open pit phases. This is at the head end of the valley where the waste rock storage and dry stack tailings management facilities will be located. Refer to Site Plan **Error! Reference source not found.**.

The purification plant is to be located in the vicinity of Rockford, Al. This is 19 miles from the mine site with access via County Roads 29 and 22. Access to natural gas in this location is key for the purification plants furnaces.

18.1.2 Warehouse, Office, and Dry

The small scale nature of the mine and proximity to nearby urban areas allows the size of the facilities to be minimal. Use of a maintenance contract means that warehousing requirements are minimized. The use of sea containers for immediate parts storage is considered with larger parts stored at the equipment vendor's offices in the nearby city.

Office and dry facilities will be portable offices located near the mill.

18.1.3 Fuel Storage and Handling

Fuel storage will be sufficient for two days of fuel for the diesel power plant (used in the initial years of the project) and mine equipment. The fuel storage area will be equipped with containment and spill basins. It will be sized to store 5500 gallons of diesel.

18.1.4 Roads

The site access road is approximately 3.4 miles from Coosa County Road 29. It will be upgraded to allow truck traffic to carry the flake concentrate from the mine to the purification plant. This road will be maintained as a gravel road with the proper ditches and culverts for surface drainage.

Site roads will be used to access the mining areas, waste and dry stack tailings management facilities, overburden stockpile, and drainage settling ponds. Site roads for mine truck traffic will be 60 ft. wide including ditches and berms for proper drainage.

18.1.5 Water Balance System

Approximately 90,000 gal/d of fresh water will be required to satisfy water demand for the process plant. This will come from water collection in the settling pond below the dry stack tailings management facility as well as from a series of water wells.

18.1.6 Service/Potable Water

Water for site services will be pumped to a water storage tank and pumped to the locations where service water is required. Potable water will be provided in bottled containers throughout the site.

18.1.7 Sewage Treatment

Sewage will be managed with the use of portable facilities with refuse trucked off site for proper disposal. Waste water will be treated as part of the plant operations and discharged to the environment.

18.1.8 Refuse Management

A recycling policy to minimize both industrial and domestic refuse will be implemented. Recyclable products will be stored at an appropriate site storage facility and removed on a regular basis. Waste products will be transported to their appropriate landfills, off site.

18.1.9 Power Supply and Distribution

Electrical power for the site will be supplied initially with diesel generation. This comes from a 1 MW unit for the plant with an additional 600 kW unit for site requirements and additional capacity for the mill should it be required.

When the production rate is increased to 16,500 tons per annum of final product, electrical grid power will be brought to site.

Access to the Alabama Power transmission line is possible with a 3.75-mile line connecting to the west of the project. The power line would follow existing roads to facilitate easy installation and periodic checks.

At full capacity the plant will have a connected load of 2.7 MW with an operating load of 1.5 MW. The mill will be serviced with a main substation and electrical power distributed by a combination of cable ducts and aerial lines. The existing diesel generation will be used as back-up power.

Power to pumps in the settling pond below the dry stack tailings management facility will be diesel powered initially. With grid power being brought to site, a 0.75 mi spur line would be added and the pumps converted to electrical energy.

18.2 Tailings Management Facility

18.2.1 General

The proposed dry stack tailings management facility is illustrated in **Error! Reference source not found.**. The material will be placed in this location by articulated dump trucks loaded by the plant loader.

18.2.2 Design Basis and Operating Criteria

The principal objective of the dry stack tailings management facility is to provide secure containment of all tailings solids generated by the milling process. The mill throughput is designed at just over 1,700 tons per day with total mill through put of 12.6 million tons of Indicated material and 2.6 million tons of Inferred material. Using a 112 lbs/cu.ft. (1.8 t/m3) dry stack density, the facility must accommodate 8.7 million cubic yards of tailings.

18.2.3 Tailings Management Facility and Reclaim Water System

The dry stack tailings management facility is located to the south of the plant location. The material will be placed with an articulated dump truck and mixed with waste material. Final sloping of this material is at a 27-degree slope for long term stability.

Seepage from the dry stack tailings facility will be collected in the settling pond further down the valley and pumped back to the plant for use as process water.

The tailings material stored in the facility has been assumed to be non-acid generating for the PEA Study. This will need to be verified with material from the pilot plant testwork and further study.

18.3 Plant/Mine Site Water Management

The plant site drainage will be collected in a settling pond with disposal to the process water pond.

Mine water will be collected in settling ponds located to the north, west, and south of the project. Two settling ponds will be located at the north end of the initial pits near the Weogufka creek to collect any potential mine runoff.

Another set of settling ponds below the overburden stockpile to the west of the pit will contain drainage from the mine on the west side.

An old dam from a previous graphite operation is located to the south of the current plant location. This would be replaced with a proper settling pond to ensure seepage and any sediment from the waste/tailings facility are contained.

These settling ponds would be cleaned annually, or as required, to ensure sufficient sedimentation control exists at all times. Water from the ponds would be used to control dust on the mine haulroads.

Surface water will be diverted away from the mine where possible to ensure contact with active mining areas does not occur. If contact does occur, it will be directed to the mine settling pond.

18.4 Waste Rock Management Facility

There will be two waste rock management facilities developed over the course of the mine life:

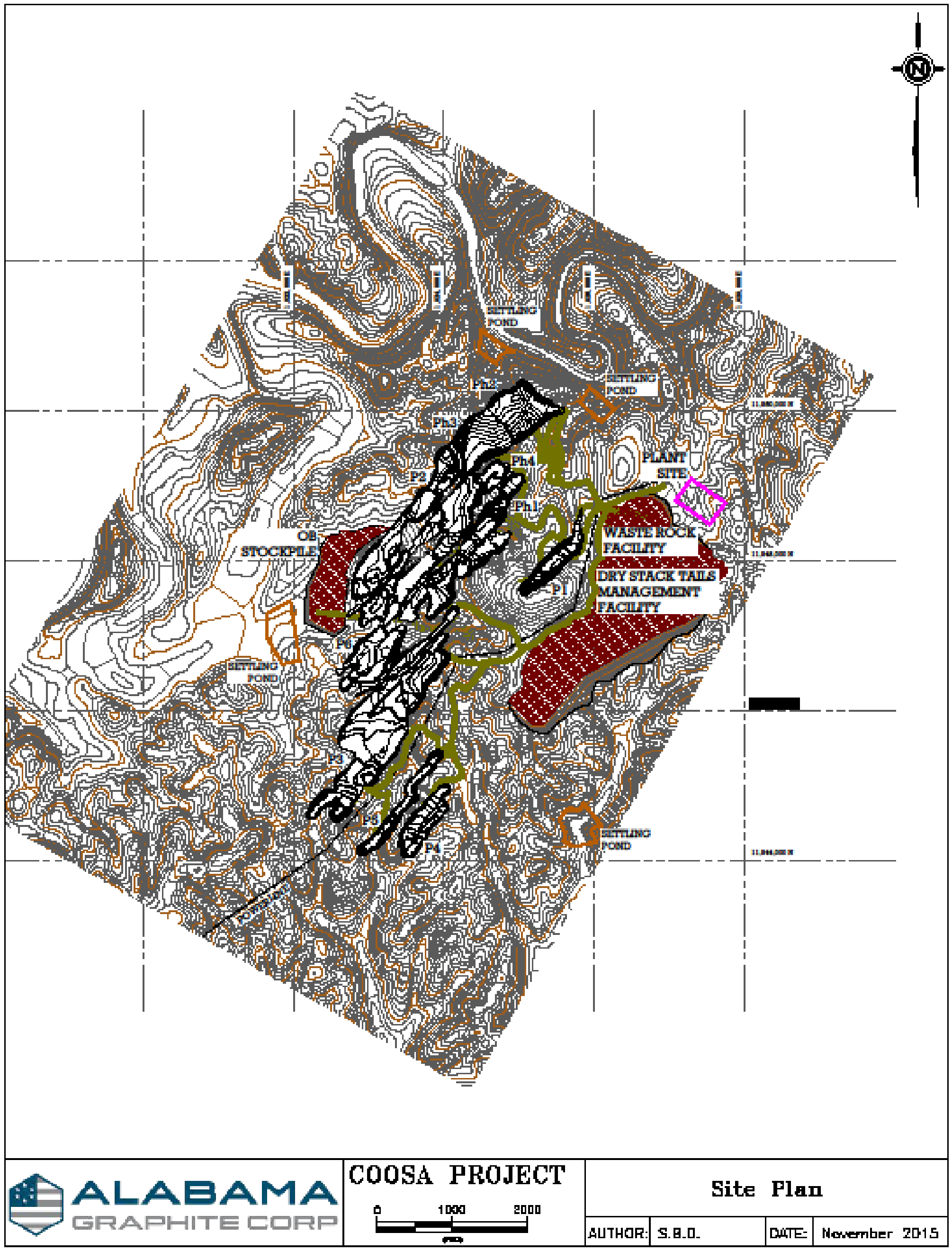
- overburden storage facility
- waste rock management facility

The overburden storage facility is situated to the west of the pits. This material will be used upon final mining of the deposit to reclaim the pits.

The waste rock management facility only requires 620,000 cubic yards of volume for the full waste storage. This material is mixed with the dry stack tailings facility material to create a single dump platform for maintenance and reclamation purposes.

ARD testing of the waste rock material is recommended for later stages of study. For the PEA, waste rock has been assumed to be non-acid generating.

Figure 18-1: Coosa Graphite Project – Site Layout



19 MARKET STUDIES AND CONTRACTS

The main applications for natural flake graphite are in refractories, batteries, crucibles, lubricants, and foundries.

China is the largest producer of flake graphite, accounting for approximately 66% of the annual production of approximately 460,000 tonnes in 2014. Brazil accounts for approximately 25% of annual production and the balance is made up by producers from India, Europe, North America, and Africa.

Regardless of whether the product is marketed as a flotation concentrate or value added product, the graphite market is characterized by direct relationships between the producer and buyer. Unlike other commodities such as copper, nickel, or gold, graphite is traded decentralized. Prices are impacted by flake sizes and carbon grades and are established directly between producers and buyers. Depending on the specific requirements of a graphite product, certification by the buyer will be carried out in a number of stages starting with small samples of less than 1 kg for initial evaluation to production size lots, which are several tons in some cases.

Natural graphite is sold as flake, spherical, micronized, expandable, and purified graphite. Prices range from $700/tonne for fine flaked graphite of -150 mesh grading 94-95% total carbon to $12,000/tonne for coated spherical graphite grading 99.95% total carbon (Summer 2015).

The graphite flotation concentrate for the Coosa County project with a grade of 95% total carbon constitutes a product that can be readily marketed to buyers. However, given the difference in value between natural and flake graphite concentrate grading 95% total carbon and value-added products, Alabama Graphite has decided to incorporate the purification, cutting, shaping, and coating value-add steps into the overall process flowsheet.

Alabama Graphite plans to purify the entire production followed by micronization. The intermediate product is then formed into spherical graphite followed by carbon coating. Since the micronization will also generate fines that cannot be formed into spherical graphite, this product will be marketed separately.

The only application for spherical, coated graphite is as anode material for lithium-ion batteries. While synthetic graphite is also suitable for battery applications, the cost of synthetic graphite is almost twice as high as coated spherical graphite derived from natural flake graphite.

The global lithium-ion battery market is expected to almost triple in size from approximately 35 GWH in 2013 to over 90 GWH in 2020 including producers such as Tesla, Panasonic, Foxconn, LG, BYD, and Boston Power. This anticipated increase in graphite consumption by the battery section will raise the demand for high-purity fine and medium sized graphite. Current and future battery manufacturers such as Tesla's Gigafactory in Nevada require approximately 0.62 kg of graphite/kWh storage capacity. At an annual nameplate capacity of 35 GWh, Tesla's Gigafactory will require approximately 21,700 tonnes/year of spherical coated graphite for this plant alone.

Any fines rejected from the forming process of the spherical graphite will be marketed as purified micronized graphite. The increased processing costs due to purification and grinding results in higher prices for micronized graphite compared to flake graphite. The current global market for micronized graphite is relatively small at approximately 25,000 tonnes.

Graphite concentrates from Alabama Graphite's Coosa Country project that will be generated in a flotation pilot plant that commenced in October 2015 will be provided to potential off-take parties and will also be subjected to value-add testwork to initiate the validation process. Some preliminary purification work that was completed using the proposed process, as described in Section 13, produced a concentrate grade of 99.971% purity using the Loss of Ignition (LOI) assay method. The sample that was used for the purification work was a composite of the flotation concentrates of the various tests on Coosa County samples prior to the flowsheet development program outlined in Section 13.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

Exploration, road access, and drilling at the Coosa Project is subject to environmental permits from the Alabama Department of Environmental Management (ADEM) including a Construction storm water permit under the National Pollutant Discharge Elimination System (NPDES), and General Permit number ALR1000000 authorizing discharges. A Construction Best Management Practices Plan (CBMPP) was prepared for storm water management, erosion and sediment control.

To minimize disturbance to the soil and natural seed bank, Alabama Graphite used a combination of forest mulchers and erosion control structures. Wildlife seed mix and straw has been placed on all new drill roads, drill pads, and trench right-of-ways to rehabilitate any disturbance and prevent erosion.

Monthly baseline water quality data has been collected from four locations at the Coosa site. Two sample locations are upstream and downstream of the project on Weogufka Creek and two locations are from ephemeral streams. Analysis is conducted at ALS Environmental in Middleton, Pennsylvania.

Weogufka Creek feeds into the Coosa River system. This is a priority item under consideration for all site layout activities, taking into account setbacks to prevent any disturbance of the local creek and river system.

Ongoing environmental work to advance the project includes waste effluent analysis for water chemistry, tailings solids, acid base accounting (ABA), and toxicity testing for local aquatic species. Material for this testwork is coming from the 2015 Q3 Pilot Plant Study at SGS Lakefield Research. This testwork is expected to be complete by the end of 2015.

Testing for dry stack tailings and water recycling systems is being initiated to determine if a zero discharge process is possible as well as eliminating the need for a tailings pond.

Biological Site Assessment, which includes a threatened & endangered species survey, wetland survey & delineation, and Phase I cultural survey, are required for the additional ADEM NPDES permitting. A comprehensive Reclamation Plan will need to be approved prior to start-up of the project.

Aerial Lidar Topographic Surveys will be completed during the winter of 2015-16. This information will be incorporated into mine site development plans, plant site layout, site access and mine roads, utilities corridors, tailings storage site, waste and overburden storage stockpiles, and appropriate storm and waste water control structure designs.

The schedule for total permitting, which is a largely Alabama State requirement, is typically 6-8 months in duration upon submission of the appropriate documentation.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

21.1.1 Summary

The capital costs for the Coosa Project are summarized in Table 21-1. The costs are based on installation of a 5,500 ton per annum concentrate operation in Year 1, followed by an expansion to 16,500 tons per annum in Year 7.

The project includes an open pit mine, a process plant with crushing/grinding/flotation and tailings dewatering operations plus a purification plant to allow production of micronized and coated, spherical, purified graphite (CSPG). Miscellaneous support infrastructure and a dry-stack tailings disposal facility are also included.

The mine has a 27-year processing life.

Table 21-1: Coosa Project Capital Cost Summary

Capital Category	Total Capital ($M)	Preproduction Capital Year -3 to Year -1 ($M)	Production Capital Year 1 ($M)	Sustaining Capital Year 2+ ($M)
Open Pit Mining	8.3	1.1	0.8	6.4
Milling & Flotation	33.6	11.7	0.0	21.9
Purification Plant	32.5	11.9	0.0	20.6
Infrastructure	8.9	2.4	0.3	6.3
Environmental	1.3	0.0	0.0	1.3
Indirects	24.0	10.0	0.1	14.0
Contingency	18.9	6.1	0.1	12.7
Total	**127.6**	**43.2**	**1.2**	**83.2**

Initial capital requirements (pre-production) are estimated to be $43.2 million. Production starts in Year 1 and the tail end of the start-up capital requirements will be partially offset by revenue in that year. Capital requirements for Year 1 total $1.2 million.

Leasing is applied to the overall mine capital requirements and only the 20% down payment of the initial capital cost is reflected in the mining capital cost summary. The indirect and contingency values vary by capital cost item, the values referred to in Table 21-2 are percentages of the direct capital numbers.

Table 21-2: Capital Category Indirect and Contingency Percentages

Capital Category	Indirects (%)	Contingency (%)
Open Pit Mining	5.0	10.0
Milling & Flotation		
5500 tpa phase	19.7	15.7
16500 tpa phase	19.9	19.4
Purification Plant		
5500 tpa phase	45.0	30.0
16500 tpa phase	39.0	30.0
Infrastructure	20.0	25.0
Environmental	5.0	10.0

21.1.2 Open Pit Mining Capital

The capital costs for the open pit mine are based on conventional small scale open pit mining equipment. The oxide zone is only being mined with no need for drill and blast. Larger, heavier equipment was specified to help with free digging the material. The uneven nature of the oxidation surface means that undulating floor conditions will exist that may it well suited for backhoe excavator and articulated truck mining.

Two articulated haulage trucks with 45-ton capacity were matched with a 6.0 cubic yard excavator initially. Support equipment includes track dozers, graders, and additional ancillary equipment. A 5.8 cubic yard loader will be located at the plant to tram material from a temporary stockpile on afternoon and evening shifts in addition to loading articulated trucks with the dry stack tailings for back haul to the dry stack tailings management area. Table 21-3 and Table 21-4 show the open pit capital breakdowns, full lease cost and capitalized down payments by equipment and by period, respectively.

Indirect and contingency costs for the open pit mine were estimated at 5% and 10% respectively.

Table 21-3: Major Equipment – Capital Cost, Full Lease Cost and Lease Down Payment

Equipment	Unit	Capacity	Unit Capital Cost ($)	Full Lease Cost ($)	Lease Down Payment ($)
Breaker Loader	yd^3	5.8	450,000	520,500	71,000
Backhoe Excavator	yd^3	6.0	919,000	1,065,600	162,100
Haulage Truck	t	45	575,000	663,000	65,000
Tracked Dozer – Large	HP	430	903,000	1,051,000	185,900
Tracked Dozer – Small	HP	90	107,000	124,700	22,400
Grader	HP	260	486,000	565,900	102,800

Table 21-4: Mining Capital by Period – Lease Down Payment

Equipment	Total Capital ($)	Preproduction Capital Year -3 to Year -1 ($)	Production Capital Year 1 ($)	Sustaining Capital Year 2+ ($)
Breaker Loader	355,800	-	71,000	284,800
Backhoe Excavator	1,458,700	-	162,100	1,296,600
Haulage Truck	1,039,400	-	129,900	909,500
Tracked Dozer – Large	929,200	-	185,800	743,400
Tracked Dozer – Small	67,200	-	22,400	44,800
Grader	514,000	-	102,800	411,200
Support Equipment	3,931,700	1,120,000	137,500	2,674,200
Total Mine Capital	**8,296,000**	**1,120,000**	**811,500**	**6,364,500**

21.1.3 Mill/Flotation Plant Capital

The mill/flotation process plant design described in Section 17 has been used as a basis for scoping-level plant capital cost estimates. An initial throughput rate of 565 tons per day has been used to achieve the desired 6,440 tons per annum of 95% graphite filter cake for delivery to the purification plant. Capital expansion to 16,500 tons per annum of final product occurs during year 7 as described in Section 22.

The cost estimate assumes that the plant is not fully enclosed and/or insulated. The project area does experience very high rainfall rates on occasion and thus a roof has been provided over much of the plant.

The mill/flotation process plant capital cost estimate is summarized in Table 21-5. The total direct cost for the process plant is $11.7M. Indirect costs total $2.31M, with the main sub components being the EPCM contract ($1.29M), first fill consumables and initial warehouse stocking ($260,000), and contractor's site mobilization/demobilization ($260,000).

A contingency equivalent to 15.7% of the direct capital costs has also been included.

Sustaining costs have been estimated simply as 1% of initial capital cost ($140,000).

Table 21-5 provides a breakdown of the capital cost estimate for the process plant.

The estimation methods for the processing plant are presented in Section 21.1.9.

The production schedules anticipate one throughput expansion during the life of mine (two were examined for a production rate trade-off discussed in Section 22). For the mill/flotation process, a new plant would be built in Year 7 to replace the existing unit and to allow a product rate of 16,500 tons per annum. Costs for this expansion is summarized in Table 21-6.

Table 21-5: Summary of Milling & Flotation Plant Capital Costs

Cost Category	Cost, $ 000's
Civil & Earthworks	1,511
Mechanicals – Supply	4,729
Mechanicals – Installation	946
Structural – Supply	603
Structural – Installation	332
Platework – Supply	528
Platework – Installation	269
Piping	676
Electrical & Instrumentation	1,182
Buildings	946
Subtotal	**11,721**
Indirects	2,309
Contingency	1,839
Sustaining Capital (1%)	136
Total	**15,869**

The old 6,440 tons per annum product plant might be decommissioned or used to process other feeds from nearby areas.

Table 21-6: Summary of Mill/Flotation Plant Capital Expansion Costs

Cost Category	Expansion Year 5 Cost $,000's
Direct Costs	21,696
Indirect Costs	4,310
Contingency	4,201
Total	**30,207**

21.1.4 Purification Plant Capital

The purification plant design described in Section 17 has been used as a basis for the high-level capital cost estimate summarized below. An initial throughput rate of 5,500 of final product per annum has been used to achieve the desired graphite concentrate production rate, with capacity expansions timed to match that of the mine and mill/flotation plant occurring in later years.

Several suitable industrial buildings have been identified in the vicinity of the Coosa project, and for this PEA it has been assumed that Alabama Graphite could lease a building for the initial phase at least. The leasing costs are included in the operating costs

Table 21-7: Summary of 5,500 tpa Purification Plant Capital Costs

Cost Category	Cost $,000's
Direct Costs	
Raw Material Handling and Storage	1,495
Purification Circuit Equipment	4,950
Spheronizing Circuit	3,690
Carbon Coating Circuit	515
Services + Utilities	1,215
Civils	45
Subtotal	**11,910**
Indirects	5,360
Contingency	3,573
Total	**20,843**

In line with the production expansion plans described above, a new purification plant would be built in Year 7 to complement the existing facility and to bring total product capacity up to 16,500 tpa. Capital cost for the expansion is summarized in Table 21-8 below.

Table 21-8: Summary of Purification Plant Capital Expansion Costs

Cost Category	16,500 tpa Expansion Cost $,000's
Direct Costs	20,582
Indirect Costs	8,027
Contingency	6,175
Total	**34,784**

21.1.5 Infrastructure Capital Cost

The infrastructure capital costs required by the mining and milling operations are listed in Table 21-9. These costs are based on information from vendor quotations, and work on similar operations.

Table 21-9: Infrastructure Capital Costs

Capital Category	Total Capital ($)	Capital, Year -3 to Year -1 ($)	Production Capital Year 1 ($)	Sustaining Capital Year 2+ ($)
Diesel Generator – 1 MW	250,000	250,000	-	-
Power Line – Mainline to Mill	750,000	-	-	750,000
Power Line – Tailings Settling pond	150,000	-	-	250,000
Site Main Substation	4,000,000	-	-	4,000,000
Fuel Storage	15,000	15,000	-	-
Shop and Garage	250,000	250,000	-	-

Capital Category	Total Capital ($)	Capital, Year -3 to Year -1 ($)	Production Capital Year 1 ($)	Sustaining Capital Year 2+ ($)
Freshwater Well and Pump System	250,000	250,000	-	-
Settling Pond Water Reclaim System	200,000	200,000	-	-
Settling Ponds	750,000	750,000	-	-
Site Mobile Equipment	250,000	250,000	-	-
Communications	50,000	50,000	-	-
Office/Dry/Warehouse	250,000	250,000	-	-
Camp Diesel Generator – 600 KW	100,000	100,000	-	-
Upgrade Access Roads to Site	335,000	335,000	-	-
Site Roads	1,315,000	-	199,000	1,116,000
Total	**8,915,000**	**2,700,000**	**199,000**	**6,116,000**

An indirect percentage of 20% was applied to the direct capital costs to cover items such as EPCM, contractor site mobilization and demobilization and various owner's costs. In addition, a contingency allowance of 25% of direct capital costs was included in the overall cost estimate.

Initial power to the plant will be with a 1 MW diesel generator with a 600 KW backup generator for site power. Grid power will be brought to the plant site with 3.75 miles of powerline from the mainline when the plant expansion to 16,500 tpa of final product is initiated. This will be constructed along existing roads when the expansion occurs in Year 7 at an estimated cost of $200,000 /mi. A tap off of this line for 0.75 miles will be constructed to provide power to the dry stack tailings facility settling pond.

The equipment maintenance facility has been sized to accommodate the small mobile equipment fleet anticipated. A full maintenance agreement has been included in the mine equipment operating cost and they would provide their own facility for maintenance.

Dry stack tailings will be stored in the dry stack tailings facility. Waste from the mining operation will also be stored in this location. Down the valley from this facility a settling pond will be located to collect runoff for plant operations and also sediment that may make its way down the valley. Settling/sediment control ponds will also be built to the north of the pit near the Weogufka creek and below the overburden storage facility. These will be cleaned as required to maintain their operational readiness.

Water well(s) sufficient for 20 gal/min of water for the plant will be developed with expansion possible to 65 gals/min when the process plant expands. Water from the dry stack tailings facility settling pond will also be used to supplement the well water to the plant.

Access roads to and around the site have been considered at a PEA level for costing. The cost of the access road was estimated considering 3.35 miles of road needing to be upgraded at a cost of $100,000 per mile. Site roads to access the various pits will need to be constructed over the mine life. These have been estimated to cost $150,000 per mile to construct these sufficient for the 45 ton articulated trucks use, with proper ditching and culverts.

21.1.6 *Environmental*

Daily environmental monitoring is included in the G&A cost category as an operating cost. Concurrent waste rock management facility re-sloping is considered a best practice for mining operations, and has been assumed and included as part of the mine operating cost. The final sloping and closure of the mine is shown as a capital cost item. This cost was estimated at $0.82 million. This includes fencing of the open pit upon closure, final re-sloping and seeding of the waste rock management area, removal of infrastructure and reclamation of site areas, and reclamation associated with the dry stack tailings management area.

Financial assurance regarding the reclamation has also been estimated, based on disturbed land on an annual basis. It is expected to cost $0.50 million for the bond interest.

21.1.7 *Indirects*

The indirect costs, as previously noted, vary between capital cost categories. Table 21-10 shows the percentages that were applied to or calculated for each area of the project. The indirect cost allowance accounts for various items, including (but not limited to) engineering and procurement, construction management, first fills of consumables and spares, plus contractor mobilization and demobilization. Also included in the indirect cost budget is an allowance for some of the Owner's costs associated with land purchases.

Table 21-10: Indirect Percentages Applied

Capital Category	Indirects (%)
Open Pit Mining	5.0
Milling & Flotation	
5500 tpa phase	19.7
16500 tpa phase	19.9
Purification Plant	
5500 tpa phase	45.0
16500 tpa phase	39.0
Infrastructure	20.0
Environmental	5.0

21.1.8 Contingency

Contingency costs have been estimated using various percentages by category and applied to the direct capital cost of a particular category. Table 21-11 shows the percentages, which illustrates the level of confidence in each of the direct capital cost estimates.

Table 21-11: Capital Cost Category Contingency Percentages

Capital Category	Contingency (%)
Open Pit Mining	10.0
Milling & Flotation	
5500 tpa phase	15.7
16500 tpa phase	19.4
Purification Plant	
5500 tpa phase	30.0
16500 tpa phase	30.0
Infrastructure	25.0
Environmental	10.0

21.1.9 Basis for Capital Estimates

Process Plant - Mechanical Equipment

- Estimated costs for mechanical equipment (i.e. crushers, mills, conveyors, thickeners, float cells, pumps, agitators etc.) were taken from recent budget quotations (ex-works) and from database information;
- Transportation costs for each piece of equipment using recent quotations for 20' and 40' containers. The transportation cost per item assumes a % of container volume, and thus cost.
- An installation rate for each equipment item was developed from an estimate of man-hours required and rates from similar projects in North America;

Process Plant - Civil Works

- An estimate of civil works costs was made by factoring from the mechanical equipment cost within each plant area, based on recent projects of similar size and scope.

Process Plant – Structural Steel and Platework

- An estimate of platework and structural steel costs was made by factoring from the mechanical equipment supply cost based on recent projects of similar size and scope.
- The direct cost includes transportation/delivery to site.
- Erection costs are factored from the structural supply cost.

Process Plant – Piping, Electrical and Instrumentation

- Factored per plant area using database information from other studies.
- The direct cost includes transportation to site and installation.

Building Costs

- Factored using database information from other studies.
- The direct cost includes transportation to site and installation.

21.2 Operating Costs

21.2.1 Summary

Operating costs were developed for an initial 550 t/d mining and milling operation with a 27-year milling life. In Year 7 the mill be expanded to a rate of 1,650 t/d.

All prices in the PEA study are quoted in Q3 2015 U.S. dollars unless otherwise noted. Diesel fuel pricing is estimated at $2.00 /gal. This estimate was derived from current prices in the area for off-road diesel fuel. The price of diesel power generation for the first phase of the plant was estimated at $0.13/kWhr. The price for electrical power was set at $40 per MWh, based on current Alabama industrial pricing.

The Coosa pit will be developed using conventional open pit technology, with small scale equipment. This includes a backhoe excavator loading 45 ton articulated trucks. The open pit has a LOM strip ratio of 0.11:1. A total of 12.5 million tons of Indicated material and 2.6 million tons of Inferred material will be fed to the mill over the 27-year processing life. A further 1.7 million tons of material will be placed in the waste rock management facility.

Mill feed material will be available immediately from Year 1 due to the near surface nature of the deposit. Only minor overburden/organics are required to be removed initial and they will be stockpiled to the west of the pits. Processing will average 191,000 tons of material to the mill in Years 1-5. In Year 6 the plant will start its ramp-up to produce 16,500 tpa of final product. This will require on average 661,000 tons per annum of feed to the mill from Year 7 until the end of the mine life.

The plant will use conventional grinding and flotation to generate a flake concentrate. This concentrate will then be trucked 19 miles to the town of Rockford where the Purification plant is considered to be located. Tailings from the process plant will be filtered and stored in the dry stack tailings facility in the valley adjacent to the south of the plant.

G&A costs are based on an average of 7 people; 2 staff and 5 hourly. Additional charges for public relations, recruitment, logistics, bussing, etc. are also included in the G&A costs. Mine employees will be located in the immediate area, and no camp will be provided or required.

No costs are allowed for product transportation or shipping: the product sales are FOB customer's vehicle at the Coosa purification plant loading bay.

Table 21-12 shows a summary of all the operating cost categories on a cost per ton (or ton) final product.

Table 21-12: Total Final Product Operating Costs – Years 1 to 27

Cost Category	Per Dry Ton Product ($/ton)	Per Dry Ton Product ($/ton)
Open Pit Mining – Mill Feed and Waste	301	332
Processing	360	397
G&A	84	93
Subtotal On-Site Costs	746	822
Filter Cake Trucking	3	3
Purification Plant	662	730
Subtotal Off-Site Costs	665	733
Total	**1,410**	**1,555**

21.2.2 Open Pit Mining

Mine operating costs were developed from base principles using vendor rates on equipment and current production rates from North American operations.

Key inputs into the mine operating cost estimate are fuel, electricity, and labour. The diesel fuel cost for this study was estimated at $2/gallon based on local diesel rates. The price of diesel power generation for the first phase of the plant was estimated at $0.13/kWhr. The price for electrical power was set at $40 per MWh, based on current Alabama industrial pricing.

Labour costs for the various job classifications were estimated using InfoMine U.S.A. Inc. 2014 "Cost Mine" information from mines currently in operation in Alabama. Burdens were calculated to average 25% for both salaried and hourly personnel. Mine shifts are using an 8-hour, day shift only schedule.

Open pit mining utilizes proven technology and equipment. No drilling is required. Larger, heavier equipment has been specified to help with free digging the oxide zone with a dozer supporting the loading completed by a backhoe excavator. Material loading is primarily accomplished with a backhoe excavator with a bucket capacity of 6.0 yd^3. A loader with a 5.8 yd^3 bucket will assist on afternoon and night shifts to ensure the crusher is full with material from a temporary stockpile. Rock haulage is completed with 2 - 45 ton articulated trucks initially, increasing to 4 units by Year 15 due to longer hauls.

Track dozers, and graders round out the major equipment list. Support equipment includes a water truck, small backhoe, utility loader, pickup trucks, pumps, and light plants.

Mine equipment requirements are minimal to start even with mining only occurring on dayshift, Monday to Friday. This includes a 6.0 yd3 backhoe, two 45-ton articulated trucks and a 430 Hp dozer. With the plant expansion in Year 7, an additional truck is required. As the haulage distance increases later in the mine schedule (both in distance and lift), another truck would be required. The trucks are assumed to be replaced as part of the lease terms each 5 years.

The large backhoe initially purchased is supplemented with another backhoe in Year 5 and two backhoes are used for the remainder of the mine life. This also allows multiple pits to be developed to blend mill feed if required.

Table 21-13: Major Mine Equipment Requirements

Equipment	Unit	Capacity	Year 1	Year 5	Year 10	Year 15
Breaker Loader	yd^3	5.8	1	1	1	1
Backhoe Excavator	yd^3	6.0	1	2	2	2
Haulage Truck	t	45	2	3	3	4
Tracked Dozer - Large	kW	433	1	1	1	1
Tracked Dozer – Small	kW	231	1	1	1	1
Grader	kW	233	1	1	1	1

The front-end loader will be used at the primary crusher, and tramming material from a temporary pile to ensure the primary crusher/conveyor system is properly charged during the afternoon and evening shifts. This loader will also load the trucks with dry stack tailings for their back haul to the tailings facility. This loader will operate around the clock.

Mine engineering and general operating costs are included in the mine operating cost. This covers a mine foreman, surveyor, grade control geologist and a secretary. Salaries for two general labourers are also included in this category. One of the labourers is responsible for the lube truck each shift to fuel and lube the dozers and backhoe as they will be located in the pits at night.

Loading costs were estimated using the backhoe excavator as the primary material mover. The larger excavator working in this configuration will be better able to break the material out without blasting. A 450 Hp dozer will assist the excavator with ripping and dozing of material for ease of loading. The average loading percentages are shown in Table 21-14.

The trucks present at the loading unit refers to the percentage of time that a truck is available to be loaded. To maximize truck productivity and reduce operating costs, it is more efficient to slightly under-truck the shovel. The value of 75% comes from the standby time expected to be encountered due to a lack of trucks.

Table 21-14: Loading Parameters

	Unit	Excavator
Waste Tonnage Loaded	%	100
Mill Feed Tonnage Mined	%	100
Bucket Fill Factor	%	90
Cycle Time	Sec	35
Trucks Present at the Loading Unit	%	75
Loading Time	min	3.03

Haulage profiles were determined for each pit phase for the primary crusher, the waste rock management facility destinations, overburden stockpile and dry stack tailings facility. From these profiles, Caterpillar's FPC software (FPC) was used to determine haulage cycle times. These cycle times were applied to the appropriate yearly tonnage by destination and phase to estimate the haulage costs.

Support equipment costs were determined using a percentage applied to either the truck hours or the loading hours. As indicated earlier, these percentages resulted in the need for one large track dozer, one smaller track dozer, and one grader. Their tasks include ripping and dozing for the backhoe, cleanup of the roads, waste storage facility, and cleanup of the settling ponds. The graders will maintain the plant feed and waste haul routes as well as the mine access road. In addition, a water truck has responsibility for controlling fugitive dust for safety and environmental reasons.

The equipment rates applied, less the operating labour costs, are shown in Table 21-15. All rates include consumables such as fuel, electricity, tires, drill steel, bits, and required maintenance parts. Maintenance costs are based on MARC pricing quotations from local vendors. Fuel consumption is estimated from basic principles and, where possible, with the FPC software as a check. Operating labour is calculated separately.

Table 21-15: Major Equipment Hourly Operating Rates

Equipment	Hourly Rate ($/h)
Breaker Loader	$65
Backhoe Excavator	$105
Haulage Truck	$60
Tracked Dozer – Large	$89
Tracked Dozer - Small	$26
Grader	$51

Leasing of the mining fleet was included in the operating cost estimate. If the fleet is leased, initial mine capital is reduced but the operating cost increases for the associated lease payments. The leasing terms were based on 20% down payment, and current vendor terms. The entire major mine equipment was leased and the majority of the support equipment where it was considered reasonable. The operating cost would vary annually depending on the equipment replacement schedule and timing of the leases.

The mining cost is calculated by year to take into account changing haulage routes, and the resulting equipment requirements. Sampling costs were added to the mine operating cost. These are based on two grab samples per day from the working face. An annual drill program to outline the next years mining block is included as well in this cost. That is estimated to cost $75,000 per year.

The LOM average cost for the total material moved is shown in Table 21-16. This includes the plant loader and the haulage of dry stack tailings to the dry stack tailings management facility.

Table 21-16: Open Pit Mine Operating Costs

Open Pit Operating Category	LOM Cost ($/t Total Material)
General Mine and Engineering	1.25
Drilling	-
Blasting	-
Loading	0.57
Hauling	0.83
Support	2.15
Leasing	1.73
Sampling	0.19
Total	**6.72**

21.2.3 *Mill/Flotation Plant Operating Cost*

The process plant operating cost is estimated at $3.37 million per annum, or $16.51 per ton processed. The majority of costs are confined to three areas: power, reagents, and grinding media. A breakdown of these costs by area is provided in Table 21-17.

Labour

Labour costs were calculated using typical plant staffing levels. Pay scales were based on recent database rates. The plant schedule is assumed to be 12-hour shifts, with two shifts at site and two shifts on leave at all times. Table 21-18 shows the estimated labour breakdowns.

The mill is estimated to require a total complement of 26 persons at an annual cost of $2.19 million, or $10.73 per ton.

Table 21-17: Operating Cost Estimate for the Mill/Flotation Plant

Area	Cost	
	($/a)	($/t)
Labour	2,189,491	10.73
Power	496,972	2.44
Reagents	136,999	0.67
Mill Media	52,634	0.26
Mill Liners	63,368	0.31
Plant Maintenance Spares	287,451	1.41
Electrical/Instrumentation	34,494	0.17
Piping	14,803	0.07
Miscellaneous Items	11,102	0.05
Plant Assay Laboratory	50,000	0.25
Safety Equipment	31,200	0.15
Total	**3,368,515**	**16.51**

Table 21-18: Mill/Flotation Plant Labour Cost Summary

Area	No. of Persons	Shifts	Total Persons	Total Cost ($/a)
Plant Management/Admin				
Plant Superintendent.	1	1	1	121,638
Plant Engineer	1	1	1	106,434
Plant Metallurgist	1	1	1	76,024
Met Clerk	1	1	1	60,819
Plant Operation				
Plant Foreman (Shift)	1	4	4	389,243
Control Room Operators	1	4	4	328,424
Plant Operators	1	4	4	304,096
Labourers	1	4	4	243,277
Maintenance				
Maintenance Foreman	1	1	1	97,311
Electrician	1	1	1	82,106
Instrument/Control Tech	1	1	1	76,024
Labourers	1	2	2	121,638
Laboratory				
Chief Chemist	1	1	1	106,434
Analytical	1	1	1	76,024
Total			**26**	**2,189,491**

Mill Power

Power is the largest single operating cost, accounting for roughly 26% of the overall total. A summary of the total cost calculation is presented in Table 21-19.

Table 21-19: Mill/Flotation Plant Power Cost Estimate

Item	Unit	Value
Total Connected Power	kW	1,224
Load Factor	%	40.3
Estimated Power Consumed	kW	493
Annual Running Time	H	7,750
Annual Consumption	MWh	3,823
Cost per MWh	$	130
Total	**$**	**496,972**

The total connected power is taken from the mechanical equipment list, with load factors applied for each piece of equipment. A power supply rate of $130 per MWh has been estimated for diesel supplied power. This equates to an annual operating cost of $2.69 per ton of mill feed treated.

Flotation Reagents

Reagent costs were estimated using unit costs from vendors and consumption rates from the lab testwork. A summary of the costs for each reagent is presented in Table 21-20. The total reagent cost amounts to $0.74/t of mill feed.

Table 21-20: Summary of Estimated Reagent Operating Costs

Reagent Consumption	Annual Cost ($)
Lime	32,683
MIBC	52,736
Fuel Oil	21,974
Flocculant	29,606
Subtotal	**136,999**

Grinding Media and Liners

Table 21-21 shows the cost of grinding balls and crusher/mill liners which were estimated using quoted supply rates and consumption estimates from similar projects.

Table 21-21: Grinding Media and Liner Operating Costs

Steel Consumption	Annual Cost ($)
Rod Mill Rods	24,500
Ball Mill Balls	28,134
Crusher Liners	55,611
Mill Liners	7,757
Total	**116,002**

Maintenance and Supplies

An allowance for plant maintenance was factored from the mechanical supply cost for each area. The cost includes transportation, but labour costs for replacement, installation, and maintenance are included in the labour allowances shown earlier in this section.

Similar amounts were factored from the overall mechanical supply cost to cover electrical/instrumentation maintenance and piping.

Fixed allowances were included to cover minor costs associated with miscellaneous items, assay lab supplies, and safety equipment. Costs in this area include training, monitoring equipment, first aid supplies, and personal protective equipment.

Tailings Management

The tailings management cost is independent of the process plant cost. The cost for the tailings is covered in the mine operating cost including the haulage, placement and sloping of the material.

Plant Expansions

As the process plant is expanded in Y7, operating costs will vary. Table 21-22 summarizes the adjustments to OPEX as the plant expansion is completed.

Table 21-22: Operating Cost for Mill/Flotation Plant Expansions

Area	Expansion to 16,500	
	Cost, $/a	Cost, $/t
Total	**5,233,891**	**8.44**

21.2.4 Purification Plant Operating Cost

The purification plant operating cost is estimated at $4.77 million per annum, or $740 per dry ton of flotation concentrate processed. The majority of costs are confined to three areas: power, raw materials, and labour. A breakdown of these costs by area is provided in Table 21-23.

Table 21-23: Operating Cost Estimate for the Coosa Purification Plant

Area	Cost ($/a)	Cost ($/t)
Leasing	60,000	9.3
Raw Materials	712,919	110.7
Power	555,794	86.3
Labour	884,000	137.3
Furnace Parts & Liners	1,600,000	248.4
Contingency	953,178	148.0
Total	**3,368,515**	**$740**

Labour

Labour costs were calculated using estimated plant staffing levels. Pay scales were based on recent database rates for the region. The plant schedule is assumed to be 12-hour shifts, with two shifts at site and two shifts on leave at all times.

At the 5,500 tpa production scale, the purification plant is estimated to require a total complement of 25 persons at an annual cost of $884,000 or $137.30 per ton of purification plant feed.

Purification Plant Power

Power accounts for roughly 12% of the overall total. A summary of the total cost calculation is presented in Table 21-24.

Table 21-24: Purification Plant Power Cost Estimate

Item	Unit	Value
Total Connected Power	kW	2150
Load Factor	%	88
Estimated Power Consumed	kW	1892
Annual Running Time	H	7344
Annual Consumption	MWh	13895
Cost per MWh	$	40
Total	**$**	**555,800**

The total connected power is taken from the mechanical equipment list, with load factors applied for each piece of equipment. A power supply rate of $40 per MWh has been estimated for line power. This equates to an annual operating cost of $86.31 per dry ton of graphite concentrate treated.

Purification Plant Raw Materials

Raw materials for the purification plant include Chlorine, Nitrogen (for purging furnaces etc.), Oxygen and Toluene. Consumptions have been estimated using the purification testwork and theoretical calculations. Unit prices for these materials were taken from budget quotations obtained from suppliers.

Contingency

Contingencies are not often added to operating cost estimates. For the purification plant, AGP has elected to include a budget for contingency, to allow for the preliminary nature of testwork in certain areas of the process. The budget is based on 25% of calculated OPEX.

Plant Expansion

As the process plant is expanded in Y7 operating costs will vary. Table 21-25 summarizes the adjustments to OPEX as the plant expansions are completed.

Table 21-25: Operating Cost for Purification Plant Expansion

Area	Expansion	
	Cost, $/a	Cost, $/t
Total	**5,907,842**	**552**

21.2.5 General and Administrative

G&A costs include the cost of the 2 staff and 4 hourly employees initially rising to 5 hourly with the plant expansion in Year 7. Employees will be located in the immediate area, and no camp is planned or required.

The G&A cost in the cash flow starts in Year 1 for the initial plant size at an annual amount of $1.1 million per year. With the plant expansion that rises to $1.2 million per year. The cost for G&A in Year 1 is $5.19/t mill feed, but drops to $1.89/t mill feed when the plant expansion occurs. A breakdown of the cost for Year 2 is shown in Table 21-26.

Table 21-26: G&A Cost Calculation (Year 2)

Category	Annual Costs Year 2 ($)
Salaried Staff	308,000
Hourly Personnel	301,000
Site Operations and Maintenance Supplies	50,000
Local Office	20,000
Information Systems (hardware/software)	10,000
Communications	10,000
Public/Community Relations	10,000
Recruitment and Training	10,000

Category	Annual Costs Year 2 ($)
Safety and Medical Supplies	10,000
Consultants	65,000
Legal and Audit Fees	20,000
Taxes and Insurance	225,000
Office Supplies	10,000
Environmental Monitoring	50,000
Total G&A	**1,099,000**
Tonnage Milled	211,826
G&A Costs ($/t)	5.19

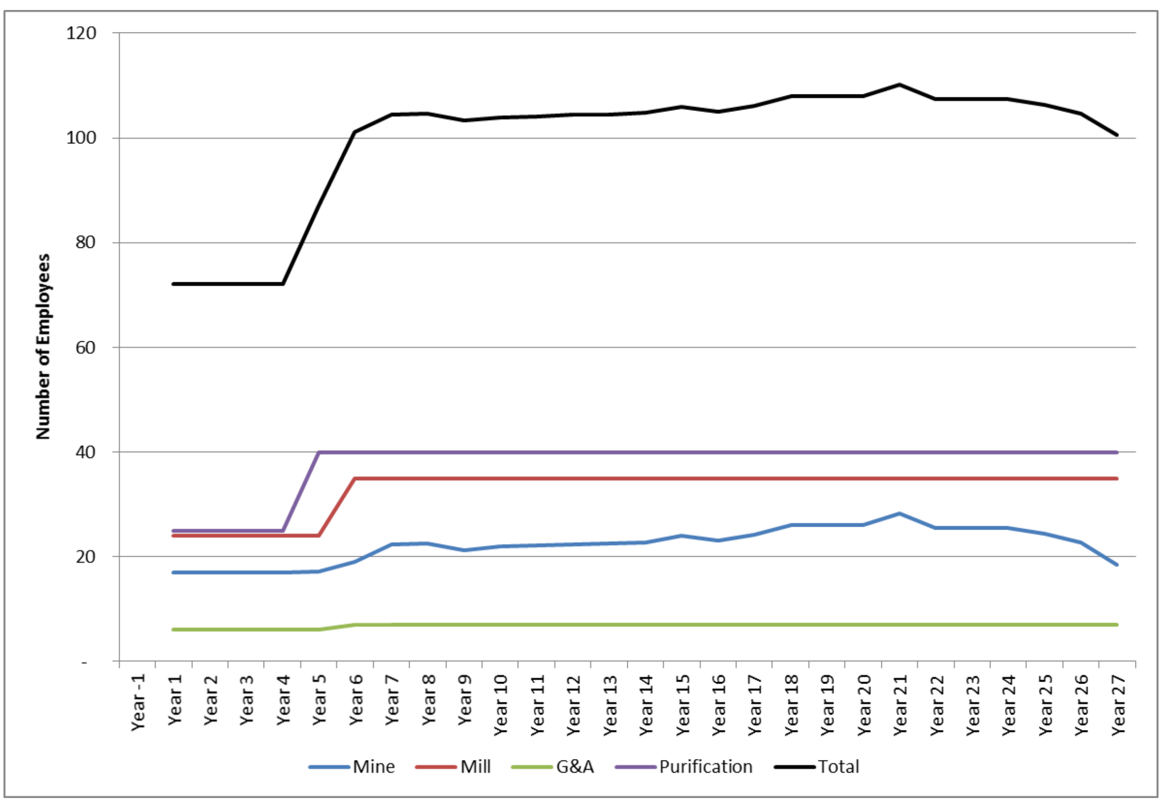
21.2.6 Products Transportation

Products are sold at the gate and therefore no costs have been allowed for this item.

21.2.7 Mine Labour Force

Labour force requirements for the Coosa Graphite Project vary from year to year by department, depending on the level of plant production. The detail on the labour force has been discussed in the appropriate sections, but a summary is presented graphically in Figure 21-1

Figure 21-1: Annual Labour Force Levels



22 ECONOMIC ANALYSIS

22.1 Production Rate Trade-off Analysis

The Coosa Graphite project was examined from the perspective of a staged approach to develop the extensive resource. Industrial mineral projects are more constrained with market demand than traditional copper or gold projects. For this reason, a focus on annual tonnage produced is crucial to the success of the project to ensure the economics are not based on unrealistic market expectations.

Alabama Graphite, from the onset considered a staged approach to allow a phased entry to the specialty graphite market. This provides for the opportunity to establish the operation at a much lower capital cost, then add capacity as market conditions permit. This also allows the ability to do two things; use free cash flow to fund further expansion plus provide an opportunity for site experience to be incorporated in later phase expansions.

Three different production scenarios were outlined at the onset:

1. 5,500 tons per annum of final product
2. 16,500 tons per annum of final product
3. 27,500 tons per annum of final product

Mine schedules were developed for each of the three cases. These used the same pit phase designs described in Section 16. The 5,500 ton per annum case (called 5K for the study) only utilized the first four phases to provide a mine life of 25 years. Additional phases could have been added which would have extended the mine life beyond 30 years but AGP believed that the impact on the overall project NPV would be negligible.

The two other cases, 16,500 tpa (15K) and 27,500 tpa (25K) were scheduled as expansions to the 5K case. They both incorporated the full number of pit phases designed in the oxide zone. The timing of the first expansion was estimated to occur for Year 5 and the second in Year 10. Initial estimates suggested that this may be when sufficient free cash flow was available to pay for the expansions without affecting payback period of the project.

Capital and operating costs were estimated for each case to put into a discounted cash flow model to compare. The costs were the same estimates used for the final cost estimate. The processing and purification plant estimates included the addition of another step in production from 16,500 tpa to 27,500 tpa to properly assess that second expansion.

The 5K to 15K expansion in Year 5 showed that the cash flow had a "double dip" where the net cash flow was negative for a second time. Based on this, two additional mine schedules were created and evaluated to calculate the NPV with the first expansion at Year 6 or Year 7.

The net result was that the expansion in Year 7 provided the lowest upfront capital, shortest payback period and sufficient free cash flow to fund the expansion. The second expansion in Year 10 was unaffected as a result of the significant increase in annual after tax cash flow after the first expansion was completed.

Graphite prices for the comparison used the Base Case pricing (CSPG = $8,165/ton, PMG = $1,814/ton). The NPV was determined at an 8% discount rate. The results of the evaluation are shown below in Table 22-1.

Table 22-1: Trade-off Study Results – Base Case Pricing

Category	Unit	Case 5K	Case 5K – 15K	Case 5K-15K-25K
Production Parameters				
Annual Final Product Production	Tons	5,500	5,500 -16,500	5,500-16,500-27,500
Year of First Expansion	Year	-	Year 7	Year 7
Year of Second Expansion	Year	-	-	Year 10
Mine Life	Years	25	27	20
Cash Flow				
Operating- LOM	(M$)	293.0	532.8	468.0
Capital – LOM	(M$)	52.8	127.6	176.7
Revenue (less Royalty)	(M$)	854.7	2439.5	2439.1
Pre-Tax Cash Flow (Revenue-Operating-Capital)	(M$)	508.7	1,779.1	1794.5
Total Tax	(M$)	119.0	480.4	489.9
Post-Tax Net Cash Flow	(M$)	389.7	1,298.7	1,304.6
Net Present Value (Pre-Tax)				
NPV @ 8%	(M$)	157	444	512
IRR	(%)	47.8	52.2	53.0
Payback Period	Years	1.9	1.9	1.9
Net Present Value (Post-Tax)				
NPV @ 8%	(M$)	120	320	368
IRR	(%)	42.4	45.7	46.6
Payback Period	Years	2.0	2.0	2.0

The results of the analysis indicated that the 5K-15K-25K case had a higher NPV and IRR. But only marginally over the 5K-15K case. When considering the marketing of 27,500 tons of final product, Alabama Graphite decided to approach the market in a more realistic manner and limit the project at this time to the 5K-15K case. The ability to rapidly expand production of the mine to the 27,500 tons exists if market conditions permit.

The final economics for the project were then based on the 5K-15K case producing 16,500 tons (15,000 tonnes) per year of final product.

22.2 Discounted Cash Flow Analysis

This PEA is preliminary in nature, and it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The tons and grades reported in Section 16 for the open pit phases were used in the discounted cash flow (DCF) analysis. Section 22.1 described the process to determine the production rate for the PEA plan. Mill throughput over the 27-year mine life is 15.2 million tons. The breakdown of the Indicated and Inferred material utilized LOM in the analysis is:

- Indicated = 12.6 million tons grading 2.85% Cg
- Inferred = 2.6 million tons grading 2.95% Cg

The Indicated tonnage accounted for 83% of the mill feed with the Inferred at 17% of the tonnage. This is a result of the near surface mining of the resource initially.

The first ten years of mining had elevated grades as a result of the mining sequence. The tons and grade for that period of time are:

- Indicated = 3.3 million tons grading 3.18% Cg
- Inferred = 0.4 million tons grading 3.35% Cg

All prices are quoted in 3Q 2015 U.S. dollars unless otherwise noted.

The graphite prices used were based on the latest market research from UK based Benchmark Mineral Intelligence. Selling prices for coated spherical graphite (CSPG) ranged from $6,350 - $10,890 per ton ($7,000 - $12,000 per tonne). A median price of $8,165 per ton ($9,000 per tonne) was used for the base case CSPG price

Micronized purified graphite (PMG) has selling prices in the range of $1,630 - $2,540 per ton ($1,800 - $2,800 per tonne). A median price of $1,814 per ton ($2,000 per tonne) was used as the base case price for PMG.

The results of the Base Case analysis indicate the potential for a Pre-tax NPV at 8% discount rate of $444 million with an internal rate of return (IRR) of 52.2%. The payback period is 1.9 years. The post-tax NPV at 8% discount rate is $320 million with an internal IRR of 45.7%. The Post-tax payback is 2.0 years.

The discounted cash flow results in more detail are outlined in Table 22-2.

Table 22-2: Discounted Cash Flow Results

Cost Category	Unit	Low Case	Base Case	High Case
Metal Prices				
CSPG	$/ton	7,257	8,165	9,072
PMG	$/ton	907	1,814	2,722
Operating Costs				
Open Pit Mining	(M$)	113.9	113.9	113.9
Processing	(M$)	136.0	136.0	136.0
G&A	(M$)	31.8	31.8	31.8
Filter Cake Trucking – to Purification Plant	(M$)	1.0	1.0	1.0
Purification Plant	(M$)	250.1	250.1	250.1
Subtotal Operating Costs	**(M$)**	**532.8**	**532.8**	**532.8**
Capital Costs				
Open Pit Mining	(M$)	8.3	8.3	8.3
Processing Plant	(M$)	33.6	33.6	33.6
Purification Plant	(M$)	32.5	32.5	32.5
Infrastructure	(M$)	8.9	8.9	8.9
Environmental Costs	(M$)	1.3	1.3	1.3
Indirect	(M$)	24.0	24.0	24.0
Contingency	(M$)	18.9	18.9	18.9
Subtotal Capital Costs	**(M$)**	**127.6**	**127.6**	**127.6**
Revenue	(M$)	2,141.9	2,233.5	2,827.3
Royalties (2.5%)	(M$)	38.2	45.1	51.9
Net Revenue(less Royalties)	(M$)	2,103.7	2,439.5	2,775.3
Pre-Tax Net Cash Flow (Revenue-Operating-Capital)	(M$)	1,443.2	1,779.1	2,114.9
Total Tax	(M$)	377.8	480.4	583.1
Post-Tax Net Cash Flow	**(M$)**	**1,065.4**	**1,298.7**	**1,531.8**
Net Present Value (Pre-Tax)				
NPV @ 0%	(M$)	1,443	1,779	2,115
NPV @ 8%	(M$)	351	444	537
NPV @ 10%	(M$)	257	329	400
NPV @ 12%	(M$)	191	247	303
IRR	(%)	43.6	52.2	60.6
Payback Period	Years	2.4	1.9	1.6
Net Present Value (Post-Tax)				
NPV @ 0%	(M$)	1,065	1,299	1,532
NPV @ 8%	(M$)	255	320	385
NPV @ 10%	(M$)	186	236	286
NPV @ 12%	(M$)	137	176	215
IRR	(%)	38.4	45.7	52.8
Payback Period	Years	6.0	2.0	1.7

The key economic parameters and cash cost calculation for various periods of the mine life have been shown in Table 22-3.

Table 22-3: Key Economic Parameters and Cash Cost Calculations – United States and Metric

Cost Category	Units	Year 1-5	Year 6-27	Year 1-27
CSPG Produced – Average Annual Production	tons	4,100	11,900	10,500
CSPG - Total Life of Mine Production	tons			283,300
PMG Produced – Average Annual Production	tons	1,400	4,000	3,500
PMG – Total Life of Mine Production	tons			94,400
Mining	$ /ton	574	280	301
Milling and Flotation	$/ton	572	343	360
G&A	$/ton	180	77	84
Sub-total Operating Cost	**$/ton**	**1,326**	**700**	**746**
Filter Cake Trucking	$/ton	3	3	3
Purification	$/ton	865	646	662
Total Production Cost	**$/ton**	**2,194**	**1,349**	**1,410**
CSPG Produced – Average Annual Production	tonnes	3,700	10,800	9,500
CSPG - Total Life of Mine Production	tonnes			257,000
PMG Produced – Average Annual Production	tonnes	1,200	3,600	3,200
PMG – Total Life of Mine Production	tonnes			85,700
Mining	$/tonne	632	309	332
Milling and Flotation	$/tonne	631	378	397
G&A	$/tonne	198	85	93
Sub-total Operating Cost	**$/tonne**	**1,462**	**772**	**822**
Filter Cake Trucking	$/tonne	3	3	3
Purification	$/tonne	953	712	730
Total Production Cost	**$/tonne**	**2,418**	**1,487**	**1,555**

22.3 Sensitivity Analysis

The project sensitivity to various inputs was examined on the Base Case. The items that were varied were recovery, metal prices, capital cost, and operating cost.

Figure 22-1: Spider Graph of Sensitivity of NPV at 8% (Post-Tax)

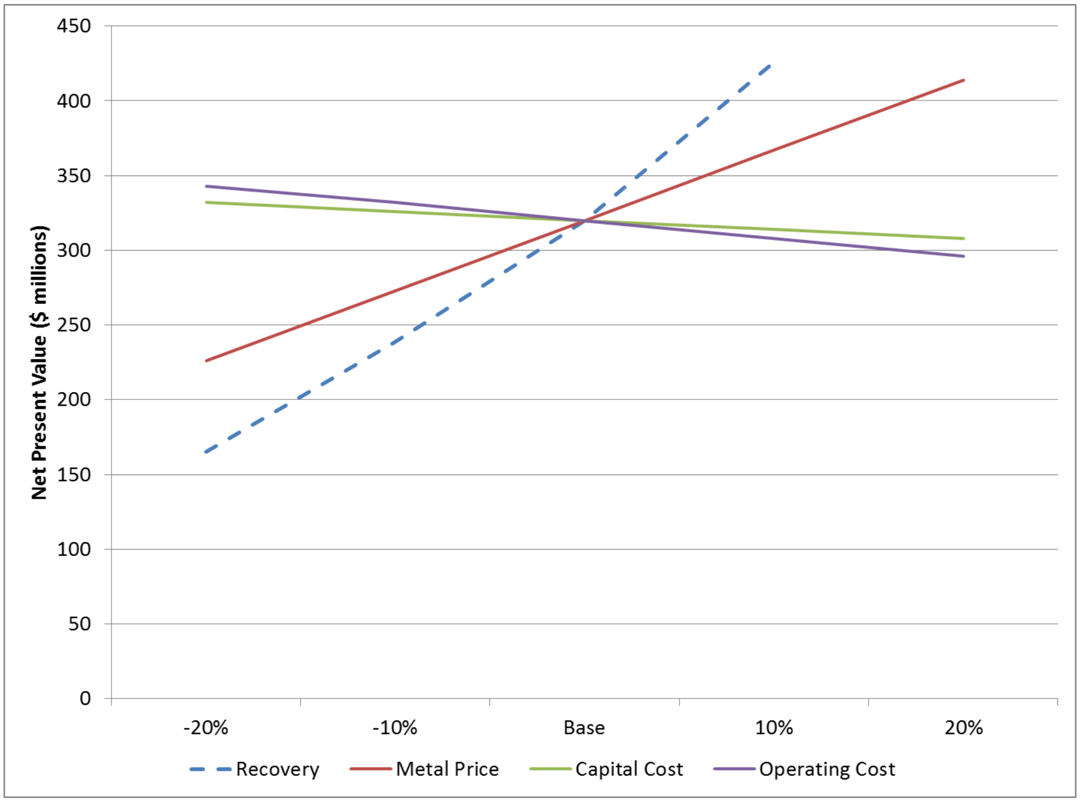
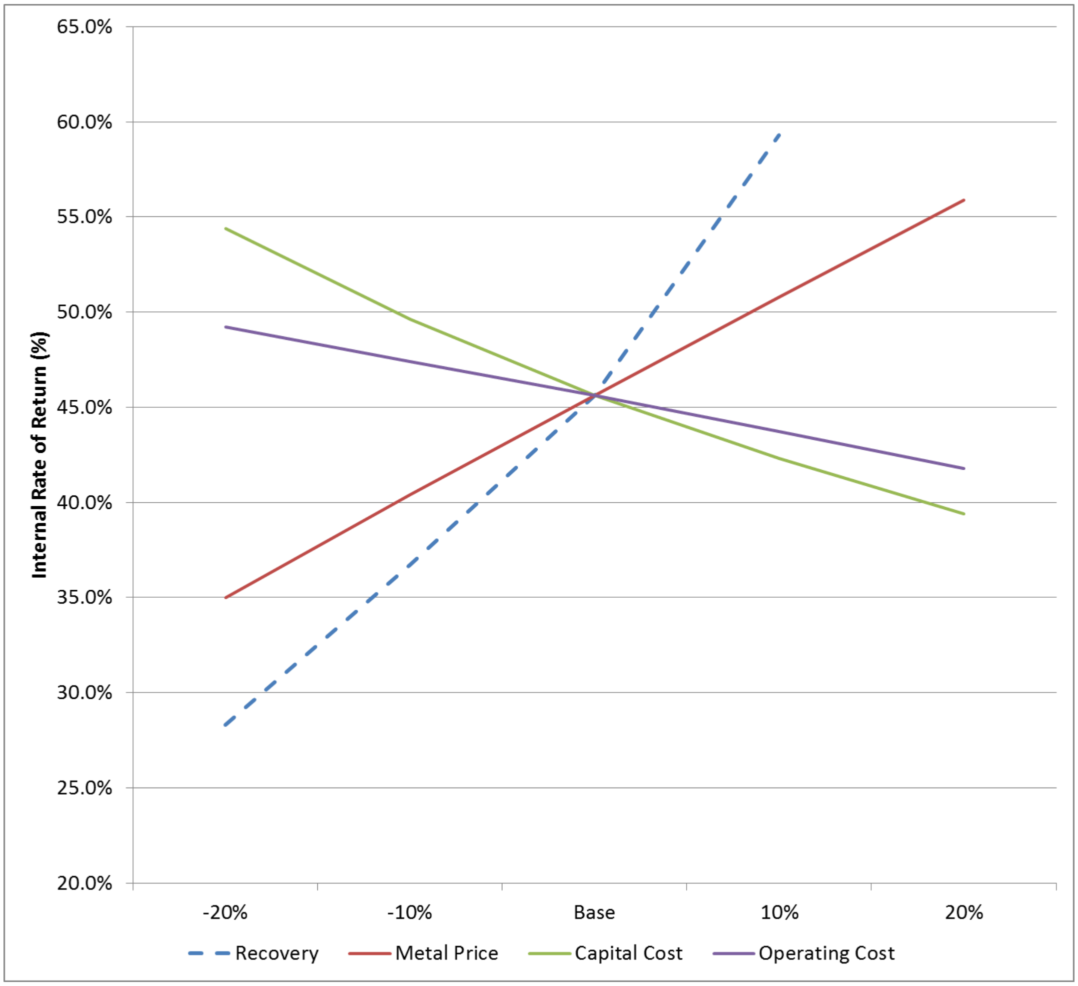


Figure 22-2: Spider Graph of IRR Sensitivity (Post-Tax)



The greatest sensitivity in the project is to recovery. To calculate the sensitivity to recovery, a percentage factor was applied to each graphite recovery in the same proportion. Therefore, while sensitivity exists, actual practice may show less fluctuation than is considered in this analysis. The +10% and +20% values are somewhat meaningless as they show recoveries greater than 100% due to the already elevated recovery. Recoveries on the negative side may also not be practical as the testwork has shown less fluctuation that the percentage variations here would indicate; but they do indicate a steeper slope showing a greater sensitivity. This large a swing in recovery has the obvious effect of influencing the project, but may not be realistic.

The next most sensitive item is the graphite price. While the project still maintains a positive NPV and elevated IRR, the reduction in the graphite price has a dramatic impact. A 20% drop in the metal price drops the IRR by 10% and the NPV by $94 million.

Operating costs are the next most sensitive. A 20% increase in these results in the $24 million decrease in the NPV.

The sensitivity to capital cost is the least sensitive. The cost of the plant is where this fluctuation may occur as it represents the majority of the capital expenditure. An increase in the capital costs by 20% drops the NPV by $12 million.

22.4 Detailed Cash Flow

The base case cash flow has been included in Table 22-4 and Table 22-5. This includes the mine schedule, concentrate and purification tonnages, operating and capital costs, revenue estimates and cumulative cash flow on which the NPV and IRR have been determined for both the Pre-Tax and Post-Tax cases.

22.5 Taxes and Royalties

Federal and Alabama State taxes have been considered applying, among other matters, the appropriate depreciation and depletion calculations.

Taxable income for income tax purposes is as defined in the Internal Revenue Code and regulations issued by the Department of Treasury and the Internal Revenue Service. The Federal income tax rate is approximately 35% in accordance with Internal Revenue Service Publication 542.

For the purposes of the PEA, the depletion deduction has been applied to the purification plant in addition to the mining. The IRC did not specifically state this could not be completed in this manner and needs to be verified going forward into the FS.

A 2% NSR royalty is payable by Alabama Graphite to the lessor from the commercial production and sale of graphite from the properties, as well as royalties for any precious metals, mica, iron, magnetite, manganese, calcium carbonate, copper, tantalum and rare earths commercially produced and sold from the properties. Only the royalty for graphite has been incorporated into the cash flow model.

Alabama Graphite is obliged to pay an independent contractor, an additional $100,000 upon receipt of a bankable FS and a further $150,000 upon full permitting of the Leased Property. The Company is also obliged to pay Merchant an NSR of 0.5% per year, up to $150,000 if and when graphite mining operations commence on the Leased Property.

Table 22-4: Detailed Cash Flow (Year -3 to Year 12)

		Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Mill Production																	
Mill Feed - Indicated	ton	12,584,056	-	-	-	202,808	186,563	109,187	64,758	63,974	335,495	607,750	637,451	540,634	587,013	633,505	429,174
DCg%	%	2.849	-	-	-	3.133	3.331	3.360	3.346	3.274	3.261	3.118	2.976	3.328	3.175	3.010	3.201
Mill Feed - Inferred	ton	2,616,353	-	-	-	-	5,141	77,421	117,078	128,220	24,949	3,330	3,974	36,452	13,407	16	161,732
DCg%	%	2.946	-	-	-	-	2.726	3.469	3.577	3.322	3.042	3.463	2.433	2.948	3.216	3.248	3.294
Mill Feed	ton	15,200,409	-	-	-	202,808	191,704	186,608	181,836	192,195	360,444	611,080	641,425	577,086	600,420	633,521	590,906
DCg%	%	2.87	-	-	-	3.133	3.315	3.405	3.495	3.306	3.246	3.120	2.973	3.304	3.176	3.010	3.227
Graphite Concentrate																	
Insitu Graphite	dry tons	435,600	-	-	-	6,355	6,355	6,355	6,355	6,355	11,700	19,067	19,067	19,067	19,067	19,066	19,067
Graphite Recovery	%	92.0%	0.0%	0.0%	0.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%
Dry Filter cake (@ carbon content)	dry tons	421,844	-	-	-	6,154	6,154	6,154	6,154	6,154	11,330	18,464	18,465	18,465	18,465	18,464	18,464
Moisture Content	%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%
Filter Cake	wet tons	496,287	-	-	-	7,240	7,240	7,240	7,240	7,240	13,329	21,723	21,723	21,723	21,723	21,723	21,723
Transit Losses	%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%
Delivered to Purification Plant (less losses)	wet tons	493,805	-	-	-	7,204	7,204	7,204	7,204	7,204	13,263	21,614	21,615	21,614	21,614	21,614	21,614
	dry tons	419,735	-	-	-	6,123	6,123	6,123	6,123	6,123	11,273	18,372	18,372	18,372	18,372	18,372	18,372
Purified Product																	
CSPG (15 microns)	dry tons	283,321	-	-	-	4,133	4,133	4,133	4,133	4,133	7,610	12,401	12,401	12,401	12,401	12,401	12,401
Micronized (5 microns >80%)	dry tons	94,440	-	-	-	1,378	1,378	1,378	1,378	1,378	2,537	4,134	4,134	4,134	4,134	4,134	4,134
Total Product	dry tons	377,761	-	-	-	5,511	5,511	5,511	5,511	5,511	10,146	16,535	16,535	16,535	16,535	16,535	16,535
Mine Production - Open Pit																	
Feed to Mill	ton	15,200,409	-	-	-	202,808	191,704	186,608	181,836	192,195	360,444	611,080	641,425	577,086	600,420	633,521	590,906
Waste	ton	1,742,363	-	-	-	16,707	10,354	13,322	10,530	10,928	96,694	45,091	60,737	39,626	26,100	28,210	200,171
Total Material	ton	16,942,772	-	-	-	219,515	202,058	199,930	192,366	203,123	457,138	656,170	702,162	616,712	626,520	661,731	791,077
Strip Ratio		0.11	-	-	-	0.08	0.05	0.07	0.06	0.06	0.27	0.07	0.09	0.07	0.04	0.04	0.34
Operating Cost				Mine Cost Moved to Capitalized Prestrip													
Open Pit Mining	dollars	113,923,816				3,167,194	3,151,587	3,151,108	3,146,596	3,193,303	3,749,158	4,256,027	4,316,472	4,178,588	4,241,981	4,414,753	4,504,557
Processing	dollars	135,954,956	0	0	0	3,348,522	3,165,174	3,081,044	3,002,244	3,173,283	3,041,001	5,155,569	5,411,587	4,868,770	5,065,635	5,344,905	4,985,366
G&A	dollars	31,829,782	-	-	-	1,052,214	994,601	968,164	943,403	997,149	679,992	1,152,826	1,210,074	1,088,696	1,132,716	1,195,163	1,114,767
Filter Cake Trucking	dollars	1,037,239	-	-	-	15,132	15,132	15,132	15,132	15,132	27,859	45,401	45,401	45,401	45,401	45,401	45,401
Purification	dollars	250,068,178	-	-	-	4,766,965	4,767,019	4,766,997	4,766,965	4,766,989	6,554,363	10,681,557	10,681,698	10,681,589	10,681,639	10,681,493	10,681,566
Subtotal Operating	dollars	532,813,971	0	0	0	12,350,027	12,093,513	11,982,444	11,874,339	12,145,855	14,052,373	21,291,380	21,665,233	20,863,043	21,167,372	21,681,714	21,331,656
Capital Cost				Includes Any Capitalized Prestrip													
Open Pit Mining	dollars	8,296,040	0	0	1,120,000	811,520	0	0	180,000	118,800	1,078,640	0	180,000	28,800	380,000	1,009,760	380,000
Underground Mining	dollars	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	66,128,100	0	4,722,000	18,888,000	0	0	0	0	21,141,000	21,164,610	0	23,610	0	23,610	0	23,610
Infrastructure	dollars	8,915,057	0	257,500	2,092,500	299,006	13,636	250,000	114,347	4,181,960	862,358	0	27,557	65,625	190,625	27,415	163,920
Environment Costs	dollars	1,347,712	0	0	3,424	3,880	4,073	4,261	6,940	7,550	10,063	10,557	11,000	13,811	15,418	15,965	17,619
Indirect	dollars	23,998,693	0	2,459,380	7,498,208	59,801	2,727	50,000	26,933	7,077,927	6,347,620	0	9,575	13,125	118,968	5,483	39,356
Contingency	dollars	18,935,068	0	1,146,355	4,997,421	74,751	3,409	62,500	36,714	6,375,480	5,412,806	0	15,016	16,406	209,341	6,854	54,124
Subtotal Capital	dollars	127,620,671	0	8,585,235	34,599,554	1,248,959	23,846	366,761	364,934	38,902,717	34,876,098	10,557	266,758	137,767	937,962	1,065,477	678,629
Revenue (after Purification)																	
CSPG (15 microns)	dollars	2,313,219,589	-	-	-	33,746,249.51	33,746,633	33,746,471	33,746,247	33,746,415	62,129,466	101,251,551	101,252,885	101,251,849	101,252,322	101,250,943	101,251,633
Micronized (5 microns >80%)	dollars	171,349,599	-	-	-	2,499,722.19	2,499,751	2,499,739	2,499,722	2,499,734	4,602,183	7,500,115	7,500,214	7,500,137	7,500,172	7,500,070	7,500,121
Subtotal		2,484,569,188	-	-	-	36,245,972	36,246,384	36,246,210	36,245,969	36,246,149	66,731,649	108,751,666	108,753,099	108,751,986	108,752,495	108,751,013	108,751,754
Royalty 2% NSR to Lessor	dollars	44,669,275	-	-	-	629,277	629,285	629,282	629,277	629,281	1,202,989	1,960,494	1,960,520	1,960,500	1,960,509	1,960,482	1,960,496
Royalty 0.5% to total of $150K	dollars	150,000	-	-	-	150,000	-	-	-	-	-	-	-	-	-	-	-
Lump sum to Independent Contractor	dollars	250,000	100,000		150,000												
Subtotal Royalty	dollars	45,069,275	100,000	-	150,000	779,277	629,285	629,282	629,277	629,281	1,202,989	1,960,494	1,960,520	1,960,500	1,960,509	1,960,482	1,960,496
Net Revenue	dollars	2,439,499,912	-100,000	0	-150,000	35,466,694	35,617,099	35,616,928	35,616,692	35,616,869	65,528,660	106,791,172	106,792,579	106,791,486	106,791,985	106,790,530	106,791,258
Pre-Tax Cashflow																	
Operating Cost	dollars	532,811,000	0	0	0	12,350,000	12,094,000	11,982,000	11,874,000	12,146,000	14,052,000	21,291,000	21,665,000	20,863,000	21,167,000	21,682,000	21,332,000
Capital Cost	dollars	127,624,000	0	8,585,000	34,600,000	1,249,000	24,000	367,000	365,000	38,903,000	34,876,000	11,000	267,000	138,000	938,000	1,065,000	679,000
Revenue	dollars	2,439,501,000	-100,000	0	-150,000	35,467,000	35,617,000	35,617,000	35,617,000	35,617,000	65,529,000	106,791,000	106,793,000	106,791,000	106,792,000	106,791,000	106,791,000
Pre-Tax Cashflow	dollars	1,779,066,000	-100,000	-8,585,000	-34,750,000	21,868,000	23,499,000	23,268,000	23,378,000	-15,432,000	16,601,000	85,489,000	84,861,000	85,790,000	84,687,000	84,044,000	84,780,000
Pre-Tax Cumulative	dollars		-100,000	-8,685,000	-43,435,000	-21,567,000	1,932,000	25,200,000	48,578,000	33,146,000	49,747,000	135,236,000	220,097,000	305,887,000	390,574,000	474,618,000	559,398,000
Post-Tax Cashflow																	
Alabama Taxes	dollars	79,645,506	-	-	-	-	-	390,800	791,164	661,124	1,984,669	3,659,974	3,678,091	3,752,781	3,731,328	3,684,892	3,715,881
Federal Taxes	dollars	400,707,535	-	-	-	380,245	1,516,447	3,994,575	3,919,116	3,438,929	8,851,918	16,623,677	18,524,114	18,919,953	18,778,292	18,535,763	18,718,876
Total Tax	dollars	480,353,041	-	-	-	380,245	1,516,447	4,385,375	4,710,279	4,100,053	10,836,587	20,283,651	22,202,205	22,672,734	22,509,620	22,220,655	22,434,756
Post-Tax Cashflow	dollars	1,298,712,959	-100,000	-8,585,000	-34,750,000	21,487,755	21,982,553	18,882,625	18,667,721	-19,532,053	5,764,413	65,205,349	62,658,795	63,117,266	62,177,380	61,823,345	62,345,244
Post-Tax Cumulative	dollars		-100,000	-8,685,000	-43,435,000	-21,947,245	35,308	18,917,933	37,585,654	18,053,601	23,818,014	89,023,363	151,682,158	214,799,424	276,976,805	338,800,149	401,145,393

Table 22-5: Detailed Cash Flow (Year 13 – 29)

		Total	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29
Mill Production																			
Mill Feed - Indicated	ton	12,584,056	119,571	524,277	503,278	608,840	649,895	630,872	355,890	522,942	646,564	723,270	684,953	648,616	611,747	573,549	381,480	-	-
DCg%	%	2.849	2.797	2.866	2.868	2.785	2.751	2.626	2.741	2.669	2.450	2.523	2.563	2.638	2.722	2.798	2.715	-	-
Mill Feed - Inferred	ton	2,616,353	520,035	135,169	153,346	76,198	45,597	103,954	350,479	206,102	105,308	29,085	50,808	66,161	82,120	104,946	15,326	-	-
DCg%	%	2.946	3.023	2.989	3.020	2.771	2.611	2.406	2.657	2.479	3.063	2.816	2.976	2.956	2.938	2.880	2.840	-	-
Mill Feed	ton	15,200,409	639,605	659,447	656,623	685,038	695,491	734,826	706,370	729,044	751,872	752,355	735,761	714,777	693,867	678,495	396,806	-	-
DCg%	%	2.87	2.981	2.891	2.904	2.783	2.741	2.595	2.699	2.615	2.536	2.534	2.591	2.667	2.748	2.810	2.720	-	-
Graphite Concentrate																			
Insitu Graphite	dry tons	435,600	19,067	19,067	19,067	19,067	19,066	19,067	19,066	19,067	19,067	19,067	19,066	19,067	19,067	19,067	10,793	-	-
Graphite Recovery	%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	0.0%	0.0%
Dry Filter cake (@ carbon content)	dry tons	421,844	18,465	18,465	18,465	18,465	18,464	18,465	18,464	18,465	18,465	18,464	18,464	18,465	18,465	18,465	10,452	-	-
Moisture Content	%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%
Filter Cake	wet tons	496,287	21,723	21,723	21,723	21,723	21,723	21,723	21,723	21,723	21,723	21,723	21,723	21,723	21,723	21,724	12,297	-	-
Transit Losses	%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%	0.50%
Delivered to Purification Plant (less losses)	wet tons	493,805	21,615	21,615	21,615	21,614	21,614	21,614	21,614	21,614	21,614	21,614	21,614	21,614	21,615	21,615	12,235	-	-
	dry tons	419,735	18,372	18,372	18,373	18,372	18,372	18,372	18,372	18,372	18,372	18,372	18,372	18,372	18,372	18,373	10,400	-	-
Purified Product																			
CSPG (15 microns)	dry tons	283,321	12,401	12,401	12,401	12,401	12,401	12,401	12,401	12,401	12,401	12,401	12,401	12,401	12,401	12,402	7,020	-	-
Micronized (5 microns >80%)	dry tons	94,440	4,134	4,134	4,134	4,134	4,134	4,134	4,134	4,134	4,134	4,134	4,134	4,134	4,134	4,134	2,340	-	-
Total Product	dry tons	377,761	16,535	16,535	16,535	16,535	16,535	16,535	16,535	16,535	16,535	16,535	16,535	16,535	16,535	16,535	9,360	-	-
Mine Production - Open Pit																			
Feed to Mill	ton	15,200,409	639,605	659,447	656,623	685,038	695,491	734,826	706,370	729,044	751,872	752,355	735,761	714,777	693,867	678,495	396,806	-	-
Waste	ton	1,742,363	186,627	82,436	102,297	52,772	52,059	109,157	125,609	122,996	173,792	48,911	43,960	38,910	32,614	10,053	1,701	-	-
Total Material	ton	16,942,772	826,233	741,882	758,921	737,810	747,551	843,983	831,978	852,040	925,664	801,266	779,721	753,687	726,481	688,548	398,506	-	-
Strip Ratio		0.11	0.29	0.13	0.16	0.08	0.07	0.15	0.18	0.17	0.23	0.07	0.06	0.05	0.05	0.01	0.00	-	-
Operating Cost																			
Open Pit Mining	dollars	113,923,816	4,572,401	4,581,408	4,680,639	4,618,352	4,744,388	4,988,851	4,966,303	4,976,955	5,311,766	4,963,423	4,954,547	4,933,107	4,864,240	3,330,429	1,913,612	26,035	26,035
Processing	dollars	135,954,956	5,396,236	5,563,632	5,539,813	5,779,543	5,867,736	6,199,596	5,959,514	6,150,817	6,343,411	6,347,486	6,207,481	6,030,446	5,854,029	5,724,339	3,347,778	0	0
G&A	dollars	31,829,782	1,206,641	1,244,072	1,238,746	1,292,352	1,312,072	1,386,279	1,332,595	1,375,371	1,418,437	1,419,348	1,388,042	1,348,456	1,309,007	1,280,008	748,590	-	-
Filter Cake Trucking	dollars	1,037,239	45,401	45,402	45,402	45,401	45,400	45,401	45,401	45,401	45,401	45,401	45,401	45,401	45,402	45,402	25,700	-	-
Purification	dollars	250,068,178	10,681,676	10,681,725	10,681,781	10,681,611	10,681,420	10,681,607	10,681,516	10,681,649	10,681,586	10,681,546	10,681,486	10,681,609	10,681,737	10,681,896	6,046,492	-	-
Subtotal Operating	dollars	532,813,971	21,902,357	22,116,238	22,186,381	22,417,259	22,651,016	23,301,734	22,985,329	23,230,193	23,800,601	23,457,204	23,276,956	23,039,017	22,754,415	21,062,074	12,082,173	26,035	26,035
Capital Cost																			
Open Pit Mining	dollars	8,296,040	28,800	0	90,000	1,123,600	28,800	200,000	0	270,000	1,103,520	0	0	135,000	28,800	0	0	0	0
Underground Mining	dollars	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	66,128,100	0	23,610	0	23,610	0	23,610	0	23,610	0	23,610	0	23,610	0	0	0	0	0
Infrastructure	dollars	8,915,057	0	0	0	0	50,000	161,648	103,693	49,716	0	0	0	3,551	0	0	0	0	0
Environment Costs	dollars	1,347,712	20,184	20,712	21,227	21,748	22,327	22,899	25,723	33,561	34,041	34,509	34,962	35,398	35,811	36,050	404,800	419,200	0
Indirect	dollars	23,998,693	0	0	76,112	4,064	10,000	34,838	20,739	98,429	0	0	0	4,209	0	10,300	10,300	20,600	0
Contingency	dollars	18,935,068	0	0	152,223	8,127	12,500	45,429	25,923	189,400	0	0	0	7,886	0	20,600	20,600	41,200	0
Subtotal Capital	dollars	127,620,671	48,984	44,322	339,562	1,181,149	123,627	488,423	176,078	664,716	1,137,561	58,119	34,962	209,655	64,611	66,950	435,700	481,000	0
Revenue (after Purification)																			
CSPG (15 microns)	dollars	2,313,219,589	101,252,681	101,253,137	101,253,675	101,252,065	101,250,254	101,252,020	101,251,165	101,252,418	101,251,827	101,251,443	101,250,875	101,252,038	101,253,260	101,254,758	57,315,308	-	-
Micronized (5 microns >80%)	dollars	171,349,599	7,500,199	7,500,232	7,500,272	7,500,153	7,500,019	7,500,150	7,500,086	7,500,179	7,500,135	7,500,107	7,500,065	7,500,151	7,500,241	7,500,352	4,245,578	-	-
Subtotal		2,484,569,188	108,752,880	108,753,370	108,753,948	108,752,218	108,750,273	108,752,169	108,751,251	108,752,597	108,751,962	108,751,550	108,750,939	108,752,189	108,753,501	108,755,110	61,560,886	-	-
Royalty 2% NSR to Lessor	dollars	44,669,275	1,960,516	1,960,525	1,960,535	1,960,504	1,960,469	1,960,503	1,960,487	1,960,511	1,960,500	1,960,492	1,960,481	1,960,504	1,960,527	1,960,556	1,109,774	-	-
Royalty 0.5% to total of $150K	dollars	150,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Lump sum to Independent Contractor	dollars	250,000																	
Subtotal Royalty	dollars	45,069,275	1,960,516	1,960,525	1,960,535	1,960,504	1,960,469	1,960,503	1,960,487	1,960,511	1,960,500	1,960,492	1,960,481	1,960,504	1,960,527	1,960,556	1,109,774	-	-
Net Revenue	dollars	2,439,499,912	106,792,364	106,792,845	106,793,412	106,791,714	106,789,804	106,791,666	106,790,764	106,792,086	106,791,463	106,791,058	106,790,458	106,791,685	106,792,974	106,794,554	60,451,112	0	0
Pre-Tax Cashflow																			
Operating Cost	dollars	532,811,000	21,902,000	22,116,000	22,186,000	22,417,000	22,651,000	23,302,000	22,985,000	23,230,000	23,801,000	23,457,000	23,277,000	23,039,000	22,754,000	21,062,000	12,082,000	26,000	26,000
Capital Cost	dollars	127,624,000	49,000	44,000	340,000	1,181,000	124,000	488,000	176,000	665,000	1,138,000	58,000	35,000	210,000	65,000	67,000	436,000	481,000	0
Revenue	dollars	2,439,501,000	106,792,000	106,793,000	106,793,000	106,792,000	106,790,000	106,792,000	106,791,000	106,792,000	106,791,000	106,791,000	106,790,000	106,792,000	106,793,000	106,795,000	60,451,000	0	0
Pre-Tax Cashflow	dollars	1,779,066,000	84,841,000	84,633,000	84,267,000	83,194,000	84,015,000	83,002,000	83,630,000	82,897,000	81,852,000	83,276,000	83,478,000	83,543,000	83,974,000	85,666,000	47,933,000	-507,000	-26,000
Pre-Tax Cumulative	dollars		644,239,000	728,872,000	813,139,000	896,333,000	980,348,000	1,063,350,000	1,146,980,000	1,229,877,000	1,311,729,000	1,395,005,000	1,478,483,000	1,562,026,000	1,646,000,000	1,731,666,000	1,779,599,000	1,779,092,000	1,779,066,000
Post-Tax Cashflow																			
Alabama Taxes	dollars	79,645,506	3,709,691	3,708,350	3,707,336	3,685,394	3,666,871	3,627,339	3,661,104	3,644,178	3,595,716	3,616,497	3,643,877	3,690,509	3,736,930	3,848,896	2,117,916	- 65,802	-
Federal Taxes	dollars	400,707,535	18,674,702	18,669,649	18,664,655	18,546,863	18,454,800	18,248,420	18,444,068	18,341,113	18,086,084	18,214,945	18,355,101	18,596,616	18,830,255	19,416,901	10,057,050	- 1,095,590	-
Total Tax	dollars	480,353,041	22,384,394	22,377,999	22,371,991	22,232,257	22,121,670	21,875,758	22,105,172	21,985,291	21,681,799	21,831,442	21,998,978	22,287,126	22,567,186	23,265,797	12,174,966	- 1,161,392	-
Post-Tax Cashflow	dollars	1,298,712,959	62,456,606	62,255,001	61,895,009	60,961,743	61,893,330	61,126,242	61,524,828	60,911,709	60,170,201	61,444,558	61,479,022	61,255,874	61,406,814	62,400,203	35,758,034	654,392	-26,000
Post-Tax Cumulative	dollars		463,601,999	525,857,000	587,752,009	648,713,752	710,607,081	771,733,323	833,258,151	894,169,860	954,340,061	1,015,784,619	1,077,263,641	1,138,519,516	1,199,926,330	1,262,326,533	1,298,084,567	1,298,738,959	1,298,712,959

23 ADJACENT PROPERTIES

There are no adjacent properties to the Coosa project that have published NI 43-101 technical reports.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other information or explanation necessary to make the Technical Report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology

The Coosa Project is located within the Alabama Graphite Belt that has a known history of graphite producing mines in Coosa County, Alabama. The Alabama Graphite Belt is associated with the southernmost part of the metamorphosed and folded belt of the Appalachian Mountains. The Coosa Project is hosted in the Higgins Ferry Group which contains bands of graphitic schist that exhibit continuity for up to 15 mi (24.1 km) or more along strike. Within the Coosa Project property, only one graphitic schist belt has been explored in detail, however surface sampling has indicated the presence of additional belts with resource potential.

Exploration results at the Coosa Project has defined a graphitic schist band with a strike length of 5,980 ft. (1,822.7 m) with a true width of 1,200 ft. (365.8 m). Graphitic material is present in two types of schist, quartz-graphite schist (QGS) and intermediate QGS to quartz muscovite-biotite-graphite-schist (INT), that generally have grades > 1% Cg. The graphitic rich band is overlain and underlain by quartz-biotite-graphite-schist (QMBGS). This usually has grade <1% Cg but with locally higher grades. The graphitic schist band has not been fully tested along strike and remains open in both directions.

MMC and Stewart Redwood reviewed pertinent data from Coosa regarding exploration data and methods and resource estimates. The resource prepared by MMC for Alabama Graphite's Coosa Project is in accordance with Canadian National Instrument 43-101 as set forth in the CIM standards on Resources and Reserves, Definitions and Guidelines (2005). MMC and Stewart Redwood completed their review of the project in preparation for this Technical Report. MMC and Stewart Redwood conclude:

Assaying and drill hole surveys have been carried out in accordance with best industry standard practices and are suitable to support resource estimates.

Sampling and assaying includes quality assurance procedures, including submission of blanks and coarse reject duplicates and execution of check assays by a second laboratory.

The Coosa Project resource model was developed using industry accepted methods.

Mineral Resources are classified as Indicated and Inferred Mineral Resources. Resource classification criteria are appropriate in terms of the confidence in grade estimates and geological continuity and meet the requirements of National Instrument 43-101 and CIM Standards on Resources and Reserves, Definitions and Guidelines (2005).

The Coosa Project is a property of Merit and has enough mineral potential that future exploration is warranted. The property has merit and potential for mineable graphite and should be explored in more detail.

MMC and Stewart Redwood met their objectives for Alabama Graphite regarding the Coosa Project.

25.2 Geotechnical

Geotechnical work has not been completed on the site at the time of this report. The next stage of study, Feasibility, will need to examine the potential wall slopes, and stability of the waste and dry stack storage facilities. In addition, the overburden stockpile will need to be examined from a stability perspective.

25.3 Mining

The Coosa Project uses Indicated and Inferred Mineral Resources in the overall mine plan. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. The open pit plans described in this study are preliminary in nature and there is no certainty that the plans will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The LOM mill feed percentages are 83% Indicated Mineral Resources and 17% Inferred Mineral Resources for the base case schedule (16,535 tpa product).

The total mill throughput in the PEA base case mine plan is estimated to be 12.6 Mt at 2.85% Cg (diluted) Indicated material, and 2.6 Mt at 2.95% Cg (diluted) of Inferred material. The strip ratio for the life of the mine is 0.11:1. The total waste is estimated to consist of 761 ktons of overburden and 981 ktons of rock. Waste stripping is at its lowest level from Years 1 to Year 4 with an average of 13 kt/a mined. The stripping level increases as the schedule transitions from a mill production rate of 5511 tpa up to 16535 tpa product. Years 12 to 21 have the highest waste production with an average of 121 kt of waste and overburden moved per year. After Year 21, the waste stripping requirements tail off for the remainder of the mine plan. The pit design is comprised of four initial phases located in the northern part of the project, followed by six additional lower grade pits to the south.

Operating costs for the open pit are expected to average $6.72/t total material over the LOM or $7.49/t of mill feed. The primary mining fleet will consist of a 6.0 cubic yard excavator, a 400+ horsepower dozer and 45 ton articulated trucks. Support to the mine and plant will be provided by graders and smaller dozers to dress roads and waste/overburden storage facilities. A smaller backhoe will also be used to ensure the settling ponds are cleaned regularly to avoid mine material leaving the project site.

The rock in the project is assumed to be non-acid generating. This assumption needs to be verified in further levels of study but is not expected to be a concern. Waste rock material will be stored along with the dry tailings in a storage facility located east of the pits and south of the process plant. The final design elevation for this waste facility was 670 ft. Overburden material has been sent to a storage facility located to the west of the pits, primarily with downhill hauls to the final design elevation of 500 ft.

Additional geotechnical review is required to enhance the pit slopes with consideration for design ramps.

Condemnation drilling is required in all areas currently proposed for waste or tailings placement.

25.4 Mineral Processing

The Coosa County mineralization displays metallurgical properties that are consistent with other graphite projects that contain interlayered graphite. In this case, the graphite flakes consist of alternating layers of graphite and gangue minerals. While the traditional graphite grinding approach using ceramic media fails to produce acceptable concentrate grades in the smaller size fractions, a high-shear grinding application using a modified stirred mill with ceramic media has produced very good concentrate grades in excess of 97% total carbon. The validity of this grinding technology has been demonstrated in a number of other graphite projects that encountered interlayered graphite.

The scoping level tests revealed a noticeable variation in the metallurgical response of the Coosa County mineralization with regards to concentrate grade and flake size distribution. In order to address this, the proposed flowsheet has been designed to be robust enough to compensate for those variations. While the master composite that was employed in the flowsheet development included six samples from three different areas of the Coosa County target, a full-scale variability flotation program will be required to ensure the robustness of the flowsheet.

Since Alabama Graphite is planning to include value-add processing in the overall flowsheet and is targeting the spherical cut and coated graphite market, flake size preservation is not as critical as in most other graphite applications. This allows the application of more energy in the cleaning circuit to improve mineral liberation and to ensure that concentrate grade targets are met or exceeded.

25.5 Purification Processes

Preliminary testwork on Coosa County flotation concentrates has been completed at an independent North American laboratory and has demonstrated the technical viability of a pyrometallurgical chlorination purification process. When this purification step is combined with other shaping, sizing and coating processes, then several speciality products are generated to increase project revenues, including coated, spherical, purified graphite (CSPG) and purified micronized flake graphite.

Including purification, the overall graphite recovery is estimated to be 85.5% (95% in flotation and 90% in purification). Ultra high grade products (99.95%) are achievable. The PEA has assumed that 75% of material fed to the purification plant would be of suitable size post-micronization for CSPG. This assumption should be verified as part of future development programs on more sizeable samples of flotation concentrate (i.e. pilot products).

25.6 Infrastructure and Site Layout

The infrastructure and site plan designs are based on information from published data and from previous work on similar operations. Several assumptions were made in the costing of this portion of the Project, and subsequent testing will provide more accurate data for refining the site plan design and associated infrastructure costs.

The site for the mill and operations was chosen for the following reasons:

- relative proximity to the open pit
- site is on top of a relatively flat height of land

For the purposes of this study, water sources required for operations will be supplied by nearby wells, and recycle water from the dry stack tailings facility settling pond. Approximately 90,000 gals/d will be required after the plant expansion.

The mine office and shop will be smaller portable offices and sprung structures. The mine equipment is leased and the vendor will provide his own facility on site. The proximity to the vendor shops in the nearby city will reduce the warehousing requirement to immediate parts storage in the sea containers.

This study assumed that the electrical capacity at point of connection to the Alabama Power Energy grid (3.75 mi away) can be made available for this project. Initial power to the project until the first plant expansion is with a 1 MW diesel generator.

The dry stack tailings storage facility is located to the south of the plant location. Waste rock will be mixed with this material to have a common facility. A tailings density of 112 lbs/cu ft (1.80 t/m^3) was used for the volume calculation and slopes of 27 degrees upon completion. Total tailings storage is 8.7 million cubic yards and waste storage is a further 620,000 cubic yards.

Control of sediment from mining activities is important to the project and Alabama Graphite. Three sets of settling ponds are created below the mine, overburden stockpile and dry stack tailings facility. These will contain any sediment that may be present and will be cleaned annually or more frequently if required. Water from these ponds will also be used to water haulroads to control fugitive dust.

25.7 Economic Analysis

The PEA has shown that a mine life of 27 years is possible with a 16,500 tpa final product operation. The Project has a pre-tax NPV of $444 million with an 8% discount rate (post-tax $320 million). The IRR for the Project is 52.2% on a pre-tax basis (post-tax 45.7%). The graphite prices used to determine the above Base Case values were $8,165 /ton ($9,000 /tonne) for coated spherical graphite (CSPG) and $1,814 /ton ($2,000/tonne) for purified micronized graphite (PMG). The pre-tax payback period is estimated at 1.9 years from the start of milling operations (post –tax 2.0 years).

The greatest sensitivity in the project is to recovery. While sensitivity exists, actual practice may show less fluctuation than is considered in this analysis. The +10% and +20% values are somewhat meaningless as they show recoveries greater than 100% due to the already elevated recovery. Recoveries on the negative side may also not be practical as the testwork has shown less fluctuation that the percentage variations here would indicate. But they do indicate a steeper slope showing a greater sensitivity. This large a swing in recovery has the obvious effect of influencing the project, but may not be realistic.

The next most sensitive item is the graphite price. While the project still maintains a positive NPV and elevated IRR, the reduction in the graphite price has a dramatic impact. A 20% drop in the metal price drops the IRR by 10% and the NPV by $94 million.

Operating costs are the next most sensitive. A 20% increase in these results in the $24 million decrease in the NPV.

The sensitivity to capital cost is the least sensitive. The cost of the plant is where this fluctuation may occur as it represents the majority of the capital expenditure. An increase in the capital costs by 20% drops the NPV by $12 million.

The project development options are sufficiently understood and the project shows positive economics to support a decision to proceed to a FS.

The PEA is preliminary in nature, it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The results of the economic analysis constitute forward-looking statements within the meaning of the United States Private Securities Litigation Reform Act of 1995 and forward-looking information within the meaning of applicable Canadian securities laws. While these forward-looking statements are based on expectations about future events as at the effective date of this Report, the statements are not a guarantee of the Company's future performance and are subject to risks, uncertainties, assumptions and other factors, which could cause actual results to differ materially from future results expressed or implied by such forward-looking statements. Such risks, uncertainties, factors and assumptions include, amongst others but not limited to metal prices, mineral resources, labour rates, consumable costs and equipment pricing. There can be no assurance that forward-looking statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements.

26 RECOMMENDATIONS

The project development options are sufficiently understood and the project shows positive economics to support a decision to proceed to a FS. As part of the FS, additional trenching is required to define and demonstrate grade continuity. The recommendations and associated budgets are described below.

26.1 Geology

Based on the favorable exploration results to date, the Coosa Project is of sufficient merit to warrant further exploration and mineral resource definition. In order to better define the high grade trends identified in the resource area, a surface trenching program consisting of 15,000 ft. of trenching and sampling is recommended to help define and demonstrated grade continuity between existing drill holes and trenches in the oxide and transitional material. The data collected would be utilized to update the geologic model and further increase the confidence in the graphite resource. The program is estimated to cost $230,000 as shown in Table 26-1.

Table 26-1: Phase 1 Exploration Program Costs

2013 Exploration Program	Unit Cost	Total Cost
Excavator (trench and backfill)	$3/ft.	$45,000
Mulching	$3/ft.	$45,000
Labor (Geologist + 2 Assistants)	$3/ft.	$45,000
Assays + Shipping	$6.25/ft.	$95,000
Project Total ($ USD)		**US $230,000**

All trenches should be constructed between existing drill hole grids and perpendicular to the known strike of the graphitic schist. The program can be completed in three phases as follows:

Phase 1: Central/grid area

Phase 2: North of central/grid area.

Phase 3: South of central/grid area.

26.2 Open Pit Mining

ARD testwork needs to be completed on the material from the pit and potential quarry sites to ensure that there are no long-term waste storage issues that may exist. From this, variations in the waste storage options may need to be considered. Humidity cell testwork takes six months or longer to develop accurate predictions.

Other areas for waste storage that may improve haulage times need to be examined for their suitability.

Comprehensive costing of labour and equipment needs to be undertaken. Upon completion, a review of owner versus contractor mining should then be conducted so that the most effective mining strategy is implemented.

Condemnation drilling needs to be considered under all waste storage locations, including the tailings dry stack facility. The plant site and other facilities should also be considered when developing a condemnation drilling program.

26.3 Processing - Flotation

The metallurgical test work to date has been encouraging, and the following recommendations are made for future metallurgical test programs:

- Optimization of the proposed process with regards to the processing of the 100 mesh screen undersize. The high-shear grinding application is characterized by a number of variables that require optimization for each deposit. While very high concentrate grades of more than 97% total carbon were obtained for this product, a process optimization will ensure that the best concentrate grades are achieved with minimum fines generation.
- Variability program using samples from different areas of the Coosa County mineralization. These variability samples should be subjected to the proposed flowsheet to validate its robustness. The variability samples should be selected taking into account different mineral domains, grade ranges, and planned mining sequence.
- Locked cycle flotation tests to establish closed circuit metallurgical performance. These locked cycle tests should be conducted on a composite reflecting the projected mine life as best as possible as well as selected variability composites.
- Comminution testing to establish crushing and grinding energy requirements. These comminution tests should include Low Energy Impact, Bond ball mill, Bond rod mill, and Bond abrasion tests.
- Pilot plant campaign to demonstrate the proposed process on a larger scale. This will also generate larger quantities of a flotation concentrate for downstream processing and evaluation by potential off-takers. The pilot plant will also allow to demonstrate the attrition technology on a larger and continuous scale.
- Environmental characterization of the process tailings. The environmental tests should be conducted on products from the pilot plant campaign and consider all tests required for environmental permitting.

26.4 Processing - Purification

The purification work to date is preliminary in nature and thus it is recommended that additional development work be undertaken using samples of pilot plant flotation concentrate. This work should include a program of laboratory test work to:

- Determine the optimal operating conditions for purification of the AG graphite and in particular to define the relationship between the quantity of chlorine and graphite purity. Excessive chlorine consumption impacts on both opex and capex and methods should be tested to reduce and optimize the required chlorine.
- Better understand the proposed method of thermal oxidation of the chloride off-gas stream for recycling the chlorine and recovery of a stable mixed oxide residue.

The laboratory program should then be expanded to pilot scale, and should include test work to:

1. Demonstrate the viability of the shaft furnace concept and generate performance data to assist refining the cost estimates.
2. Demonstrate the viability of the off-gas treatment and chlorine recycling concept.
3. Lastly, collaboration with established suppliers is recommended to further develop the technical aspects and refine the costs for:
    a. the graphite material handling system
    b. the shaft furnace graphite liners
    c. the chlorine recycling system

26.5 Infrastructure and Site Layout

Additional testing and data is required to further define the infrastructure and site layout requirements and associated costs in these areas:

- Topographical information to provide more detail for infrastructure design, including power line routing, access road rehabilitation, and building placement.
- Geotechnical testing and data to further develop construction requirements for the mill, tailings and services sites.
- Hydrogeology and water quality testing on water sources surrounding the mine site to determine available volumes and quality of water required to support the mill and services infrastructure.
- Additional electrical study is necessary in order to validate the conceptual design and the assumed load capacity. The local electrical utility can perform an Electrical Service Agreement (ESA) study in order to determine the best option for the expanded plant's power requirements.

- Materials testing and quarry location scoping for design and construction cost estimates.
- Foundation testing, quarry placement, and material suitability studies need to be completed for the plant site and all the settling pond locations.
- Condemnation drilling under the plant, WRM, and dry stack tailings area.
- Environmental and socioeconomic studies.

26.6 Waste and Water Management

A complete hydrogeological review of water in the mining area needs to be undertaken. This entails both quantity and quality sufficient for operation of the process plant and mine. In addition, the precipitation and drainage areas need to be determined for proper estimating of diversion dam/ditches/settling ponds to minimize contact of fresh water with mining areas.

The dry stack tailings management facility and waste rock management facility areas need to have complete hydrological evaluations completed for surface runoff, ground water, and seepage.

ARD and metal leaching testwork needs to be developed and completed for proper waste rock characterization, dry stack tailings and development of storage options.

26.7 Environmental

With the PEA completed and some understanding of the potential size and scope of the project now understood, the data collection and testwork can begin in earnest to prepare for the next stage of study and ultimately permit applications.

Basic data collection needs to commence immediately covering a wide range of diverse subjects, including: weather, water flows, vegetation, wildlife, and socio-economic. A comprehensive program will need to be established to collect the required information necessary to comply with the respective agency permit application requirements. This is of critical importance to ensure that the permits may be issued in a timely manner. Baseline environmental studies, including Biology (vegetation and wildlife), Cultural Resources, and Waters of the United States & Wetland Delineation will need to be completed.

A detailed FS plan will also be required to build upon the other information collected. These two items will require advancing forward in parallel with the technical experts providing input into the permit application process. Detailed environmental and engineering information must be collected in at least the following areas:

- air quality
- cultural resources
- noxious weeds, invasive, and non-native species

- environmental justice
- floodplains
- geology and geochemistry
- land use authorization
- livestock grazing management
- meteorology
- migratory birds
- minerals
- native American religious concerns
- paleontological resources
- social and economic values
- soils
- special status plants and animals
- threatened and endangered species
- vegetation
- visual resources
- wastes (solid and hazardous)
- water (surface and ground)
- wild pigs
- wildlife

Seasonal data of at least twelve months may be required for some of the elements above. The programs should start as soon as possible.

26.8 Feasibility Study

AGP recommends that Alabama Graphite proceed with activities that will allow completion of a FS. AGP estimates that the cost of a FS for the Coosa Graphite Project would be in the range of $3-6 million to complete, although the scope of the FS remains to be thoroughly reviewed and a detailed budget determined. The FS would cover areas such as:

- resource estimate update
- metallurgical testwork, pilot plant, product testing
- geotechnical and hydrogeological studies
- condemnation drilling
- dry stack tailings management facility design, material characterization and site geotechnical
- environmental management studies and data collection
- graphite marketing and sales studies
- capital and operating cost estimation
- infrastructure evaluation and costing
- financial evaluation
- project management and administration

Aspects of the study can take place concurrent with the exploration budget. Environmental and social work and some of the geotechnical and metallurgical work would also occur at this time.

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28 CERTIFICATE OF AUTHORS

28.1 Scott E. Wilson, CPG

I, Scott E. Wilson, of Highlands Ranch, Colorado, do hereby certify:

- I am currently employed as President by Metal Mining Consultants Inc., 9137 S. Ridgeline Blvd., Suite 140, Highlands Ranch, Colorado 80129.
- I am a co-author of the technical report titled "Preliminary Economic Assessment (PEA) on the Coosa Graphite Project, Alabama, USA" (the Technical Report) dated 27 November 2015.
- I graduated with a Bachelor of Arts degree in Geology from the California State University, Sacramento in 1989.
- I am a Certified Professional Geologist and member of the American Institute of Professional Geologists (CPG #10965) and a Registered Member (#4025107) of the Society for Mining, Metallurgy and Exploration, Inc.
- I have been employed as either a geologist or an engineer continuously for a total of 24 years. My experience included resource estimation, mine planning geological modeling and geostatistical evaluations of numerous projects throughout North and South America.
- I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
- I did not make a personal inspection on the property for personal reasons and I relied on the expertise of my colleague and coauthor, Dr. Stewart Redwood, who made a personal inspection of the project on June 8-10, 2015.
- I am responsible for Sections 14-1, 25-1, and 26-1 of the Technical Report.
- As of the date of the report, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- My prior involvement with the property was as co-author of the Technical Report dated 16 July 2013.
- That I have read NI 43-101 and Form 43-101F1, and that this technical report was prepared in compliance with NI 43-101.
- I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 27th day of November 2015 at Peterborough, Ontario.

"Original Document signed and sealed
by Scott E. Wilson, CPG"

Signature

28.2 Stewart D. Redwood, FIMMM

I, Stewart D. Redwood, FIMMM, hereby certify that:

- I am a Consulting Geologist with address at P.O. Box 0832-1784, World Trade Center, Panama City, Republic of Panama.
- I am a co-author of the technical report titled "Preliminary Economic Assessment (PEA) on the Coosa Graphite Project, Alabama, USA" (the Technical Report) dated 27 November 2015.
- I graduated from Glasgow University with a First Class Honors Bachelor of Science degree in Geology in 1982, and from Aberdeen University with a Doctorate in Geology in 1986.
- I am a Fellow of The Institute of Materials, Minerals and Mining, #47017.
- I have more than 30 years' experience as a geologist working in mineral exploration, mine geology, mineral resource and reserve estimations and feasibility studies on projects in North America, Latin America, the Caribbean, Europe, Africa, Asia and Australia.
- I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional organization (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
- My most recent personal inspection of the Coosa Project was on June 8-10, 2015.
- I am responsible for Sections 4-1, and 5 to 12 of the Technical Report.
- I am independent of the Alabama Graphite Corp. applying all of the tests in Section 1.5 of NI 43-101.
- My prior involvement with the property was as co-author of the Technical Report dated 16 July 2013 and to advise Alabama Graphite Corp. about sampling, analyses and QA/QC.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with that instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 27th day of November 2015 at Peterborough, Ontario.

"Original Document signed and sealed
by Stewart D. Redwood, FIMMM"

Signature

28.3 Gordon Zurowski, P.Eng.

I, Gordon Zurowski, P.Eng., of Stouffville, Ontario, as a Qualified Person (QP) of this technical report titled "Preliminary Economic Assessment (PEA) on the Coosa Graphite Project, Alabama, USA" with an effective date of 27 November 2015 do hereby certify the following statements:

- I am a Principal Mining Engineer with AGP Mining Consultants Inc., with a business address at #246-132 Commerce Park Dr., Unit K, Barrie, ON L4N 0Z7 Canada.
- I am a graduate of the University of Saskatchewan, B.Sc. Geological Engineering, 1989.
- I am a member in good standing of the Professional Engineers of Ontario (Membership #100077750).
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101 or "the instrument") and certify that, by reason of my education, affiliation with a professional association (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.
- My relevant experience includes the design and evaluation of open pit mines for over 20 years.
- I am responsible for the content of Sections 1, 2, 3, 15, 18, 20, 21.1, 21.1.1 to 21.1.2, 21.1.5 to 21.1.8, 21.2.1, 21.2.2, 21.2.5 to 21.2.7, 22 to 24, 25.2, 25.6, 25.7, and 26.5 to 26.8 of this technical report.
- I conducted a site visit of the property on July 9 – 10, 2015.
- I am independent of the issuer as defined by Section 1.5 of the instrument.
- I have read NI 43-101 and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed and dated this 27th day of November 2015 at Stouffville, Ontario.

"Original Document signed and sealed
by Gordon Zurowski, P.Eng."

Signature

28.4 Willie Hamilton, P.Eng.

I, Willie Hamilton, P.Eng., of Calgary, Alberta, as a Qualified Person (QP) of this technical report titled "Preliminary Economic Assessment (PEA) on the Coosa Graphite Project, Alabama, USA" with an effective date of 27 November 2015 do hereby certify the following statements:

- I am a Senior Associate Mine Engineer with AGP Mining Consultants Inc., with a business address at #246-132 Commerce Park Dr., Unit K, Barrie ON L4N 0Z7 Canada.
- I am a graduate of the University of Alberta with a Bachelor of Science in Mining Engineering in 1988 and a Master of Science in Mining Engineering in 1990.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of Alberta (Membership #47481) and the Association of Professional Engineers and Geoscientists of British Columbia (Membership #110011).
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101 or "the instrument") and certify that, by reason of my education, affiliation with a professional association (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.
- My relevant experience includes the design and evaluation of open pit mining operations and projects for more than 25 years.
- I am responsible for the content of Sections 16, 25.3, and 26.2 of this technical report.
- I have not conducted a site visit of the property.
- I am independent of the issuer as defined by Section 1.5 of the instrument.
- I have read NI 43-101 and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed and dated this 27th day of November 2015 at Calgary, Alberta.

"Original Document signed and sealed
by Willie Hamilton, P.Eng."

Signature

28.5 Andy Holloway, CEng., P.Eng.

I, Andrew Holloway, CEng., P.Eng., of Peterborough, Ontario, as a Qualified Person (QP) of this technical report titled "Preliminary Economic Assessment (PEA) on the Coosa Graphite Project, Alabama, USA" with an effective date of 27 November 2015 do hereby certify the following statements:

- I am a Principal Process Engineer with AGP Mining Consultants Inc., with a business address at #246-132 Commerce Park Dr., Unit K, Barrie ON, L4N 0Z7 Canada.
- I am a graduate of the University of Newcastle upon Tyne, England, B.Eng. (Hons), 1989.
- I am a member in good standing with the Professional Engineers of Ontario (Membership #100082475).
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101 or "the instrument") and certify that, by reason of my education, affiliation with a professional association (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.
- My relevant experience with respect to metallurgy and mining project management includes 25 years' experience in the mining sector covering mineral processing, process plant operation, design engineering, and operations and project management. I have been involved in numerous projects around the world in both base metal, industrial mineral and precious metal deposits.
- I am responsible for the content of Sections 17.5, and 21.1.4 of this technical report.
- I have not conducted a site visit of the property.
- I am independent of the issuer as defined by Section 1.5 of the instrument.
- I have read NI 43-101 and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed and dated this 27th day of November 2015 at Peterborough, Ontario.

"Original Document signed and sealed
by Andy Holloway, CEng., P.Eng."

Signature

28.6 Phillip Mackey, P.Eng.

I, Phillip Mackey, P.Eng., of Kirkland, Quebec, as a Qualified Person (QP) of this technical report titled "Preliminary Economic Assessment (PEA) on the Coosa Graphite Project, Alabama, USA" with an effective date of 27 November 2015 do hereby certify the following statements:

- I am President of P.J. Mackey Technology, Inc., with a business address at 295 Kirkland Blvd., Kirkland QC, H9J 1P7 Canada.
- I am a graduate of the University of New South Wales, Sydney Australia, B.Sc. (Hons.), 1964 and Ph.D., 1969.
- I am member in good standing of the Professional Engineers of Ontario (Membership #28236016).
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101 or "the instrument") and certify that, by reason of my education, affiliation with a professional association (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.
- My relevant experience with respect to metallurgy and metallurgical engineering includes over 45 years of experience in operations, research, engineering and development, due diligence studies of mining and metallurgical projects, and project management. I have been involved in numerous mining and metallurgical projects around the world in a range of base metals.
- I am responsible for the content of Sections 13.6, 13.7.2, 21.2.4, 25.5, and 26.4 of this technical report.
- I have not conducted a site visit of the property.
- I am independent of the issuer as defined by Section 1.5 of the instrument.
- I have read NI 43-101 and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the parts of this Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed and dated this 27th day of November 2015 at Kirkland, Quebec.

"Original Document signed and sealed by
Phillip Mackey, P. Eng."

Signature

28.7 Lyn Jones, P.Eng.

I, Lyn Jones, P.Eng., of Peterborough, Ontario, as a Qualified Person (QP) of this technical report titled "Preliminary Economic Assessment (PEA) on the Coosa Graphite Project, Alabama, USA" with an effective date of 27 November 2015 I do hereby certify the following statements:

- I am a Senior Process Engineer with AGP Mining Consultants Inc., with a business address at #246-132 Commerce Park Dr., Barrie ON, L4N 0Z7 Canada.
- I am a graduate of the University of British Columbia, Metals and Materials Engineering, 1998.
- I am a member in good standing of the Professional Engineers of Ontario (Membership #100067095).
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101 or "the instrument") and certify that, by reason of my education, affiliation with a professional association (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.
- My relevant experience with respect to metallurgy and mining project management includes 17 years' of work experience in plant operations, metallurgical testing, flowsheet development, and process engineering.
- I am responsible for the content of Sections 17.1 to 17.4, 21.1.3, 21.1.9, 21.2.3, 25.4, and 26.3 of this technical report.
- I have not conducted a site visit of the property.
- I am independent of the issuer as defined by Section 1.5 of the instrument.
- I have read NI 43-101 and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the parts of this Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed and dated this 27th day of November 2015 at Peterborough, Ontario.

"Original Document signed and sealed
by Lyn Jones, P.Eng."

Signature

28.8 Oliver Peters, P.Eng.

I, Oliver Peters, P.Eng., of Peterborough, Ontario, as a Qualified Person (QP) of this technical report titled "Preliminary Economic Assessment (PEA) on the Coosa Graphite Project, Alabama, USA" with an effective date of 27 November 2015 do hereby certify the following statements:

- I am an Associate Process Engineer with AGP Mining Consultants Inc., with a business address at #246-132 Commerce Park D., Unit K, Barrie, ON L4N 0Z7 Canada.
- I am a graduate of the Technical University of Aachen, Germany, 1998.
- I am a member in good standing of the Professional Engineers of Ontario (Membership #100078050).
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101 or "the instrument") and certify that, by reason of my education, affiliation with a professional association (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.
- My relevant experience with respect to metallurgy and mining project management includes 17 years' experience in the mining sector covering mineral processing, process engineering, and operations and project management.
- I am responsible for the content of Sections 13.1 to 13.5, 13.7.1, 19, 25.4, and 26.3 of this technical report.
- I have not conducted a site visit of the property.
- I am independent of the issuer as defined by Section 1.5 of the instrument.
- I have read NI 43-101 and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, the parts of this Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed and dated this 27th day of November 2015 at Peterborough, Ontario.

"Original Document signed and sealed by Oliver Peters, P.Eng."

Signature