AN EVALUATION OF PROBLEMS ASSOCIATED WITH 1000 VAC LONGWALL MINING SYSTEMS

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ABSTRACT

In the last several years, the coal mining industry has made significant changes in the design of longwall mining systems to make them more productive. Some changes, such as increasing the length of the coal faces and increasing the capacity of the face conveyors, have resulted in problems with achieving sufficient motor torque to start and operate the longwall face conveyors. Other changes, such as increasing the size and/or decreasing the percent impedance of longwall power center transformers, have resulted in problems with the short-circuit currents on the transformer secondaries exceeding the interrupting ratings of the molded case circuit breakers used to protect the secondary circuits. A survey of 24 longwall mining systems in northern West Virginia was conducted to investigate both problems. The results of the survey are presented and analyzed. Several recommended solutions to the problems are presented to enable the coal mining industry to design safe and productive longwall mining systems.

INTRODUCTION

Since 1970 the coal mining industry has installed over 100 longwall systems. At least 20 of these systems are located in northern West Virginia. In the last few years, the mining industry has been making major changes in the design of these systems to make them more productive. Some of the changes in design included increasing the width of the longwall mining face to 600-700 feet, increasing the capacities of the face conveyors, and increasing the lengths of the panels to 6000-8000 feet. This trend toward larger and more complex longwall systems has resulted in a corresponding increase in the size of conveyor motors, shearer motors and transformers and the adoption of a standard utilization voltage of 1000 Vac.

Mine Safety and Health Administration (MSHA) personnel recognized that the larger and more complex longwall systems could present safety problems and MSHA conducted a survey of the problems associated with longwall mining systems installed in northern West Virginia. This survey indicated that two major problems exist on longwall systems. One problem is inadequate startup or breakdown torques of the chain-conveyor motors. At times these motors would not start or would stall during operation. The mining industry, in an attempt to solve this problem, sometimes created a second problem by exceeding the interrupting ratings of 1000 Vac molded-case circuit breakers.

Technical evaluations of the problems identified during the survey indicated that practical solutions can be achieved with little or no detrimental effect on coal production.

DESCRIPTION OF LONGWALL MINING SYSTEMS

A typical longwall mining system consists of several electrically operated components, interfaced with the coal-haulage system for the time. The major components are the shearer unit, shearer controller, face conveyor unit, stage-loader unit, conveyor controller, hydraulic pump unit, pump controller, master controller and power center.

As shown in the layout drawing for longwall mining systems in figure 1, the size of the motors and the size and length of the cables can vary significantly. The total horsepower in the surveyed systems varied from 900 to 1750 hp and the power center rating varied from 750 to 1700 kVA. The systems had an average total horsepower of approximately 1100 hp and an average total power center rating of approximately 1250 kVA. These figures indicate that a diversity factor is not used by the mining industry to determine the size of the power centers. If a diversity factor were used the size of the power center would be less than the total horsepower. Instead, the mining industry uses other factors and conditions to determine the size of the power centers. The main factor used is one that allows sufficient voltage at the face chain conveyor motors to provide sufficient torque during start-up and to prevent stalling of the motors during operation.

The conveyor controller and shearer controller are usually located in the head-gate entry approximately 500 feet away from the power center and are usually provided with No. 4/0 AWG input cables. The controllers are equipped with protective devices for the conveyor and shearer motor circuits and are controlled by the master controller through 120 Vac control cables. The pump controller is usually located within 100 feet of the power center and may be provided with up to a No. 4/0 AWG input cable. Two conveyor motors were usually provided for each face chain conveyor. Six different motor sizes (125 hp, 175 hp, 200 hp, 250 hp, 300 hp, and 350 hp) were used to operate the face conveyors. The capacity of the face conveyors varied from 600-1200 tons per hour. The larger motors, 250 hp and above, were used to transport coal at production rates of 900-

1200 tons per hour. However, in most cases the outby haulage systems were not capable of handling these higher production rates and coal spillage has been a problem unless the mining rate is limited.

In addition to the face conveyor motors, a stage loader motor is usually provided to transfer the coal from the face conveyor to the outby belt conveyor system. The stage-loader and face-conveyor motors are usually Class C designs and are connected to the chain conveyors through fluid couplings. These fluid couplings are designed to transfer running torque to the load only after the motor has reached a specified speed, typically 1200-1600 r/min. If the couplings are overfilled, running torque is transferred to the load at a lower speed and if the couplings are under filled, running torque is transferred to the load at a higher speed. Also, the fluid levels of the couplings for two conveyor motors driving a face conveyor must be properly matched in order for both motors to equally share the load during operation of the face conveyor.

The shearer unit may consist of one or more motors which are started simultaneously by the shearer controller. The total horsepower of the shearer may vary from 225 to 600 hp. In most cases the shearer units are provided with automatic controls which limit the cutting rates so that the motors do not stall during operation. During start-up, the units are usually located so that the majority of starting load is from the inertia of the cutting drums. However, if the cutting drums are located against the coal face, the shearer unit would most likely not start, causing a stalled condition.

The hydraulic pump unit consists of two motors which provide independent operation of a hydraulic roof support system. The motors vary from 50 - 150 hp with most systems utilizing 100 hp motors with a Class C design.

TYPICAL SEQUENCE OF OPERATIONS

The following is a typical sequence of operations of a longwall mining system. First, the two pump motors are started with a 1 second delay between the first and second pump motors. This is usually done near the beginning of the shift and the pumps remain in continuous operation. Second, after the main belt conveyor has started operating, the stage loader is started. After a slight time delay the tail-face conveyor motor is energized. This step prevents slack in the chain from building up underneath the face conveyor. After a two-second time delay the head-face conveyor motor is energized providing two motors to operate the face conveyor. Last, the shearer unit is started and coal is mined from the longwall face. The shearer unit is not electrically interlocked with the chain conveyor motors. However, the shearer unit is not operated unless the face conveyor is operating. This prevents a buildup of coal on the face-conveyor which can cause the face-conveyor motors to stall upon start-up. Also, even though the hydraulic pump system is electrically independent of the rest of the longwall system, the pumps must be operated to assure proper roof support and provide advancement of the face equipment and the roof support system as the longwall face is mined.

EVALUATION OF MOTOR START-UP CONDITIONS

One of the main problems, observed during the survey of longwall mining systems, was the condition of low motor torque during start-up of the face conveyors. In order to evaluate the starting torque of the face conveyor motors, the systems electrical parameters had to be determined so that the motor voltage or motor current could be calculated. Since the motor torque is proportional to the square of the motor current, the impedances of the system from the utility source to the motors were calculated. The following conservative assumptions were made when calculating the impedances of the power systems:

1. The impedance of the utility should provide for little or no contribution from motors. In other words, use the values of impedances which provide the lowest short-circuit currents.

2. Ignore the contribution of motors that may act as generators during high-inrush currents.

3. The impedance of the outside substation transformer should be determined from the nameplate data. This assumes that the transformer is operating at the designed temperature. Typical oil-filled transformers are designed for a 55°C rise above ambient. Thus for a 40°C ambient temperature, the windings of the transformer are operating at 95°C.

4. The impedance of the high-voltage distribution system originating at the outside substation should be calculated for the maximum length of cable expected to be used. Also, the resistance should be based on the maximum temperature rating of the conductor insulation usually 90°C for portable cables. Impedance charts may be used for aerial transmission lines [1] and for portable power cables [2].

5. The impedance of the power-center transformer should be determined from information from the manufacturer based on a 150°-155°C rise above ambient temperature. Thus, the resistance will be based on an operating temperature of 170°-175°C.

6. The impedance of the 1000 Vac portable cables should be determined from impedance charts for portable cables [2] based on the maximum temperature rating of the conductors' insulation.

7. The impedance of the face-conveyor motors should be determined from the value of current at the pickup speed of the fluid coupling, usually 70-85% of synchronous speed. In this range, the motor current will be 4 to 5 times the rated full-load current of the motor. The X/R ratio can be approximated by assuming that the power factor at pick-up speed is 0.50. The values listed in Table 1 were used to determine the currents and impedances of the motors at 70% of synchronous speed.

TABLE 1 - MOTOR IMPEDANCES			
SIZE (hp)	FULL-LOAD CURRENT (Amperes)	70% SPEED CURRENT (Amperes)	70% SPEED IMPEDANCE (Ohms)
125	69	345	0.7472 + J 1.3808
175	96	480	0.5701 + J 0.9875
200	107	535	0.5126 + J 0.8879
250	134	670	0.4093 + J 0.7090
300	161	805	0.3407 + J 0.5901
350	193	965	0.2848 + J 0.4933
400	212	1060	0.2587 + J 0.4481

An example of these calculations is provided in Appendix A. The system evaluated had approximately 120 MVA available at the primary of the outside substation and 31 MVA available at the primary of the power center. However, with the 1000 kVA 5.5% power center transformer and the portable cables as shown, the system had 12 MVA at the transformer motor. This resulted in a current for the tail-face conveyor motor of approximately 495 A at 70% of synchronous speed, or pickup speed of the fluid coupler. The actual motor current is only 89% of the design current at this speed.

If this value of current is reduced by 5% to account for utility system voltage variations, then the output torque of the motor would be 72% of the design torque at the pick-up speed of the fluid coupler.

If the head-face motor is started before the tail-face motor has reached its rated speed, then both motors will be operating near the pick-up speed of the fluid couplings causing the voltage at both motors to be further reduced. This condition would cause the torque of the tail-face and head-face motors to be reduced to 57% and 68%, respectively, of the design torque for the motors at pick-up speed. Thus, only 63% of the total design torque for both motors would be available to start and run the face conveyor. This condition may allow the motors to stall when the conveyors are fully loaded or during start up.

Appendix B contains a summary of the operating characteristics of the various longwall systems located in northern West Virginia. Of the 24 longwall systems evaluated, one system used 350 hp face-conveyor motors, three systems used 300 hp motors, 10 systems used 250 hp motors, 7 systems used 200 hp motors, 2 systems used 175 hp motors, and 1 system used 125 hp motors. The calculated torques of the tail-face conveyor motors varied from 67-89% of the design torques when the motor currents were reduced by 5% due to utility system variations. Twelve of the systems evaluated had calculated torques less than 75% of the design torques for the motors.

Appendix B also provides a torque constant K which was devised by the authors to normalize each of the systems to assist the coal industry in the calculation of starting torques for face-conveyor motors. The constant for any face-conveyor motor can be determined by dividing the short-circuit current available at the motor by 5 times the motor full-load current. Once K is calculated, Table II can be used to approximate the starting torque of the motor. If two motors are used to start the face-conveyor then the total torque should be reduced by 5-15%. This also approximates the amount of minimum breakdown torque necessary to prevent stalled-motor conditions.

TABLE II - MOTOR TORQUE CONSTANT

DESIGN TORQUE (% BREAKDOWN)	K
70	4.10
80	6.10
85	7.75
90	10.25

In the example of Appendix A, if the longwall system is to have at least 75% design torque with two-200 hp motors operating, then the tail-face motor should be capable of providing at least 80% of design torque when operating by itself during start-up. This would require that the torque constant K be at least 6.1. The minimum short-circuit current available at the tail-face motor should be 3386 (6.1 x 550) amperes. This is an increase of approximately 930 A in the short-circuit current at the tail-face conveyor motor.

Several ways are available to increase the torque:

1. Increase the primary system short-circuit MVA.

2. Increase the size of the power center transformer.

3. Decrease the impedance of the power center transformer.

4. Use larger capacity 1000 Vac portable cables to the motors.

5. Increase the ratio of the speed reducers of the face conveyors.

If the primary system short-circuit MVA is increased from 31 MVA to 40 MVA, then the short circuit current would increase to 2532 A. If the system was increased to 60 MVA, the short-circuit current would increase to only 2639 A. Thus, this method would achieve very small results when the high-voltage system is upgraded.

If only the size of the power-center transformer is increased from 1000 kVA to 1500 kVA the resulting short-circuit current would be approximately 2580 A. Again, the extra expense will achieve very little gain in starting torque.

If the impedance of the power-center transformer is decreased from 5.5 to 3.0% the short-circuit current available at the tail-face motor would be only 2624 A.

By installing a No. 4/0 AWG portable cable in parallel with the existing No. 4/0 AWG input cable to the conveyor controller, the short-circuit current available at the tail-face motor would be increased to approximately 2708 A. If the motor cable is increased in size from a No. 2 AWG to 1/0 AWG, the available short-circuit current would increase to approximately 2993 A without any change in the input power cable. If both cable changes were made the short-circuit current would increase to approximately 3370 A. This would increased K from 4.42 to 6.07 and result in a minimum starting torque of approximately 80% of design torque.

The load torque seen by the motor can be reduced by using speed reducers with higher gear ratios which will not affect the electrical characteristics of the system but will permit the load to be more evenly matched to the motors. In many cases the face conveyors have higher capacities (tons per hour) than the outby haulage system. Thus, in order to prevent coal spillage, the longwall face must often be slowed down or stopped completely resulting in a possible loss of coal production.

A combination of changes could be made to increase the short-circuit current at the tail-face motor. However, changes involving the high-voltage system may be quite expensive and very little advantage is obtained if the short-circuit MVA is increased over 40 MVA. This is shown in figure 2. Also, changes made to the high-voltage system or the power-center transformer may result in more expensive hardware, since higher short-circuit currents at the secondary of the transformer can result in more expensive hardware, since higher short-circuit currents at the secondary of the transformer can result in major modifications in the design of the power center. This will be discussed in detail in another section of the paper. The short-circuit current which is obtained at the tail-face motor by using a 1500 kVA - 3% power center transformer is only 2736 A. A new power center only achieved an increase of 280 A in the short-circuit current at the tail-face motor. This may be very uneconomical, since the increase in starting (breakdown) torque would be minimal at the tail-face motor. However, the torque at the head-face motor may have an appreciable increase, since it is located closer to the power center.

In addition to the theoretical evaluations on the 24 longwall systems, actual current-voltage measurements were conducted on three of these longwall systems. By measuring inrush currents of motors and voltages at the power center, the actual test results could be compared to the calculated results. The results of one series of tests are shown in Appendix B. Some of the findings include the following:

1. The calculated voltages at the power center are within ± 5% of the measured voltages. The power factor was not measured and was assumed to be 0.50.

2. The calculated system impedance was 0.1098 ohms and the measured system impedance averaged 0.100 ohms.

3. The shearer and pump motors reached operating speed within 25 cycles after start-up. However, both conveyor motors took up to three seconds to reach operating speed.

EVALUATION OF INTERRUPTING RATINGS OF 1000 VAC MOLDED-CASE CIRCUIT BREAKERS

The second problem evaluated was the condition of systems having available short-circuit currents which exceeded the interrupting ratings of 1000 Vac molded-case circuit breakers. In order to evaluate whether or not the systems surveyed were equipped with circuit breakers having interrupting ratings higher than available short-circuit currents, electrical parameters of the system had to be determined and compared with the interrupting rating of the various circuit breakers being used.

Presently, only two manufacturers supply the industry with 1000 Vac mine-duty circuit breakers. The interrupting ratings vary from 10 kA to 14 kA for the 100 to 1200 A frame circuit breakers. A higher interrupting rating can be obtained by using a larger 3000 A frame circuit breaker. (Note: In order to achieve the maximum interrupting rating the line leads must be rear connected to the circuit breaker.) Since standards are not available for testing the interrupting ratings of 1000 Vac molded-case circuit breakers, the interrupting ratings are listed for systems having short-circuits with a 0.50 power factor or higher. If the power factor is lower than 0.50, then the interrupting rating must be derated to account for lower power factors. The symmetrical rating is approximately equal to the asymmetrical rating at 0.50 pf.

Thus, both the maximum available fault current and the minimum power factor of system must be used in order to properly determine if the circuit breakers on a system have adequate interrupting capacity. The following conservative assumptions should be made when calculating the short-circuit currents available within the system:

1. The minimum utility impedance should be used.

2. Contribution from motor loads which are connected outby the power-center transformer should be included. In order to simplify the calculations, the motor contribution may be left out and the calculated short-circuit current increased by a multiplying factor.

3. The impedance of the outside substation transformer should be determined from the nameplate data. The resistance should be corrected to a no-load temperature of approximately 15-20°C above ambient. Thus, for a 55°C rise transformer, the resistance should be reduced by a factor of approximately 15%.

4. The impedance of the high-voltage distribution system originating at the outside substation should be calculated for a maximum conductor temperature of 40°C. Thus, the resistance should be reduced by a factor of at least 20%.

5. The impedance of the power-center transformer should be determined from information supplied by the manufacturer with a maximum winding temperature of 60°C. Thus, the resistance will be reduced by approximately 50%.

6. The impedance of the 1000 Vac portable cables should be determined from appropriate cable charts and the resistance reduced by approximately 30%. This assures that the resistance of the conductors is based on a conductor temperature of 20°C (Mine ambient).

7. The impedance of the motors can be approximated by assuming that the motors are operating at no-load and that the subtransient impedance equals the locked-rotor impedance [3]. However, the short-circuit current from motor contribution can be approximated by multiplying the full-load current by 3.6 [3] [4] [5] [6] [7]. At 1000 Vac, this is approximately two times the horsepower of the motors or if the total horsepower rating equals the power center kVA rating, then approximately two times the power center kVA rating.

This method was used to evaluate the amount of short-circuit current on seven of the 24 systems installed in northern West Virginia. It was noted that when lower (cold) values of resistances were used instead of the higher of both values, the amount of short-circuit current at the secondary of the power center increased only by 200-500 A. However, the power factors decreased 5-7% (Table III). The evaluation also indicated that the power factors for short-circuits at the secondaries of the power center transformers varied from 20 to 34%.

TABLE III - SUMMARY OF SHORT CIRCUIT SURVEY

COMPANY/MINE	HOT CONDUCTORS		COLD CONDUCTORS	
	I_{sc} (A)	Power Factors (pf)	I_{sc} (A)	Power Factors (pf)
A - 1	17412	0.230	17466	0.220
A - 2	11426	0.320	11665	0.266
B - 6 (1)	12257	0.330	12458	0.254
D - 1	12433	0.305	12662	0.253
F - 1 (1)	11910	0.390	12288	0.326
F - 1 (2)	11446	0.406	11849	0.340

Thus, the interrupting capacity of the circuit breakers on 1000 Vac systems must be derated since the circuit breakers were rated at a power factor of 50%. If circuit breakers are used in a system with power factor of 20% the interrupting ratings must be multiplied by a factor of .899 as shown in figure 3. This graph is based on [4] and [7]. Thus, a circuit breaker rated at 14 kA at 50% power factor would be derated to 12.58 kA at a power factor of 20%.

The evaluation also indicated that motor contribution can not be neglected in calculating the available short-circuit current even if the impedance of the primary system is assumed to be 0.0 ohms. This is shown in Appendix C and figure 4. Figure 4 can be used to determine either the recommended interrupting rating of circuit breakers if the impedance of the power center is known, or it may be used to determine the recommended power center impedance once the interrupting rating is derated for a power factor below 50%.

If a 1500 kVA-4% impedance transformer is used, then the interrupting rating should not be less than 25 times the full-load current (833 amperes) at the center. A more conservative value of 28.6 times the full-load current can be used to account for additional motor contribution. Thus, the interrupting rating would be 21-23.4 kA.

If a 1000 kVA power center is equipped with 12 kAIC circuit breakers with an assumed power factor of 0.20, then the ratio of the interrupting rating to the full-load current would be approximately 19.44. The graph in figure 4 indicates that the maximum impedance should not be less than 5.1%.

The values at this impedance are based on the assumption that the primary distribution system has a short-circuit MVA which equals or exceeds 75 times the power-center kVA rating (75 MVA). If the primary MVA is less than 75 times the power-center kVA rating, then a more detailed study would have to be conducted in order to determine the minimum impedance.

In the above example if the primary has 45 MVA, then the impedance can be reduced to 4.01%. However, this method would not provide any measure of safety if the power system is upgraded to provide more than 45 MVA or if the power center was moved to another location, i.e., closer to the main substation, a different mine, etc. It is the responsibility of all concerned, power-center manufacturers, circuit breaker manufacturers, mine operators, etc., to assure that the power center is always installed so that the interrupting ratings of circuit breakers are not exceeded.

If the interrupting rating is exceeded, several methods may be used to reduce the available short-circuit currents to values below the interrupting ratings of the circuit breakers:

1. Install air-core reactors between the transformers and the circuit breakers;

2. Decrease the available MVA of the primary system by installing transformers with higher impedance in the main substation;

3. Decrease the available MVA of the primary system by installing smaller transformers in the main substation;

4. Install two separate power transformers within the power center. One transformer can be used to supply the shearer and pump motor circuits and the other can be used to supply the conveyor motor circuits; or

5. Install circuit breakers with higher interrupting ratings.

Each of the above methods has been used in northern West Virginia. It should be noted that all but the last method may reduce the conveyor starting torque which was described earlier. Method 4 also has the advantage of not only decreasing the available short-circuit current from the power system, but it also decreases the amount of short-circuit current that is contributed by the other motors in the longwall system.

Mine Safety and Health Administration electrical personnel and various coal-mine electrical personnel also conducted tests to indirectly measure the available short-circuit current at the secondary of several power centers. These tests indicated that calculated values of short-circuit currents were within 5% of the measured values. The results of these tests are not included in this paper since the power systems at these mines are constantly changing. Tests were not conducted to measure short-circuit currents contributed by motors.

CONCLUSIONS

On several longwall mining systems, problems have been encountered with low torque during startup and operation of face conveyor motors. A torque constant, K, was developed to approximate the minimum amount of torque which is available during motor start-up conditions. This constant can be used to assist in the design of longwall mining systems. The following methods can be used to achieve higher starting torques:

1. The impedance of the high-voltage system can be reduced to provide a stiffer power system. However, very little advantage is obtained if the available short-circuit MVA at the primary of the power center transformer is increased above 40 MVA;

2. The impedance of the power center transformer can be reduced i.e. larger transformer size, lower percent impedance, or both; or

3. The impedance of the 1000 Vac portable cable can be reduced by the use of larger power cables or by paralleling existing cables.

Since the majority of the voltage drop, resulting in lower motor torque, occurs due to the impedance of the 1000 Vac distribution system, the third method can result in significantly higher increases in motor torque than the first two methods. It should also be noted that the first two methods will cause an increase in the short-circuit current available at the secondary of the power-center transformer.

The use of large transformers with low impedances to supply power to longwall mining systems has caused short-circuit currents at the secondary of these transformers to approach, and in some cases exceed, the interrupting ratings of 1000 Vac circuit breakers. The interrupting ratings of present circuit breakers must be derated if the short-circuit power factor is less than 0.50. Typical short-circuit power factors of 1000 Vac power systems vary from 0.20 to 0.34. Since, it has been the trend of the mining industry to provide stiffer power systems for 1000 Vac longwall mining industry to provide stiffer power systems for 1000 Vac longwall mining systems, it is recommend that the interrupting ratings of 1000 Vac circuit breakers be based on a maximum short-circuit power factor of 0.20.

The calculation of available short-circuit currents on 1000 Vac systems should use impedances based on temperature which are less than the maximum operating temperature of the conductors. Although the short-circuit current is increased only slightly, the short-circuit power factor is decreased 5-7%. Since the no-load voltage may be 1100 Vac or more, the calculated short-circuit current should not exceed 90% of the interrupting rating of the circuit breakers. This will also provide a margin of safety if the interrupting rating of a circuit breaker decreases due to contamination from the mine environment.

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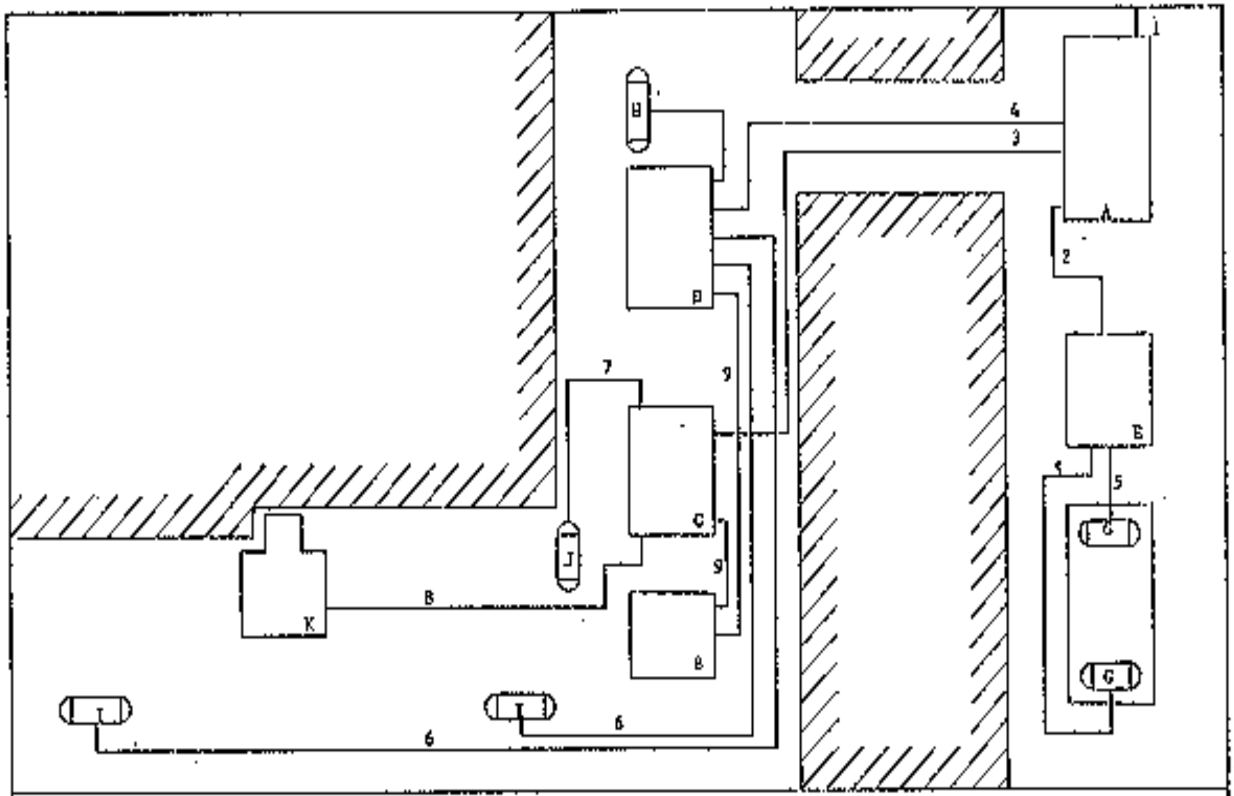

Components

A. Power Center (750-1700kVA)
B. Master Controller
C. Shearer Controller
D. Conveyor Controller
E. Pump Controller
G. Pump Motor (50-150hp)
H. Stage Loader Motor (50-150hp)
I. Conveyor Motor (125-350hp)
J. Crusher Motor (100-200hp)
K. Shearer Motor (210-300hp)

Cables

1. High Voltage Distribution Cable (1/0-4/0 AWG SHD-GC)
2. 2KV Pump Feeder Cable (#2 - 4/0 AWG SHD-GC, 50-500 feet)
3. 2KV Shearer Feeder Cable (4/0 AWG - 350 MCM - SHD-GC, 500-1000 feet)
4. 2KV Conveyor Feeder Cable (4/0 AWG - 350 MCM - SHD-GC, 500-1000 feet)
5. 2KV Pump Motor Cable (#4 - 1/0 AWG, SHD-GC, 10-50 feet)
6. 2KV Conveyor Motor Cable (#2 - 4/0 AWG, SHD-GC, 30-700 feet)
7. 2KV Crusher Motor Cable (#2 - 1/0 AWG, SHD-GC, 20-50 feet)
8. 2KV Shearer Motor Cable (#2 - 4/0 AWG, SHD-GC, 350-700 feet)
9. 600V Control Cable

FIGURE 1 - Layout of Typical Longwall Systems

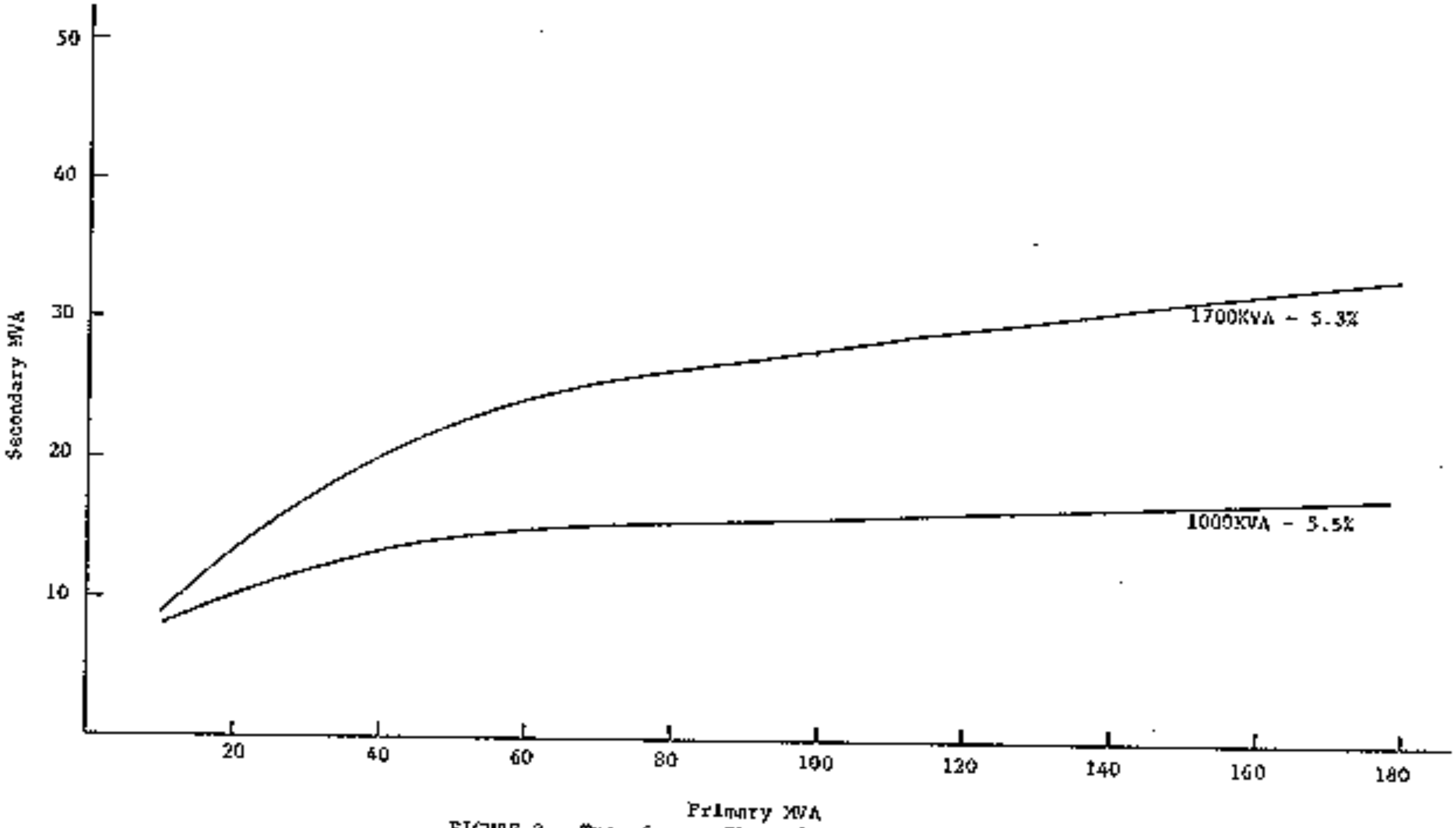

FIGURE 2 - Transformer Short-Circuit Characteristics

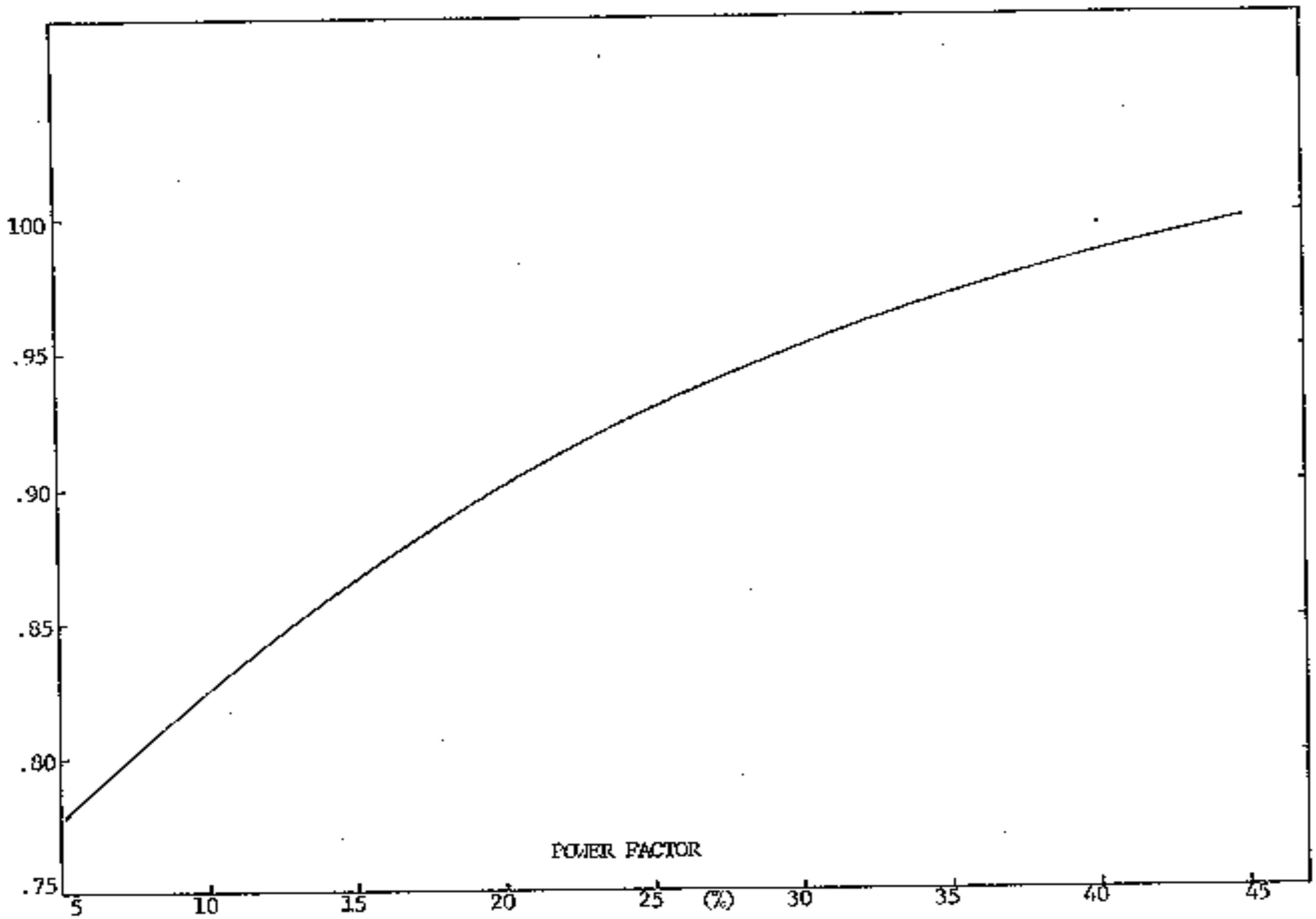

FIGURE 3. DERATING FACTORS FOR MOLDED CASE CIRCUIT BREAKERS

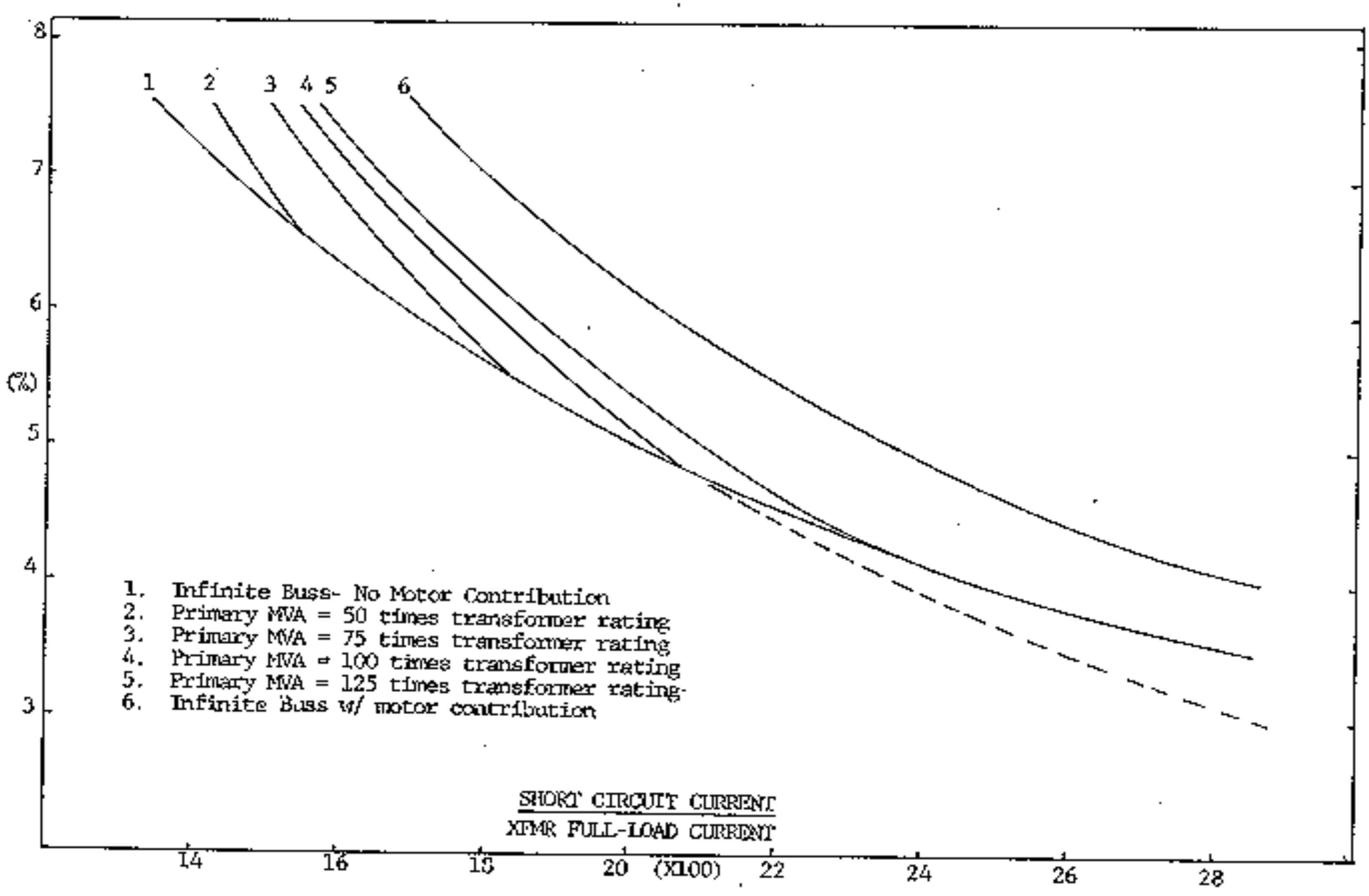

FIGURE 4. POWER CENTER CHARACTERISTICS

APPENDIX A

Theoretical Calculations of Company B
Mine 9 Longwall Mining System

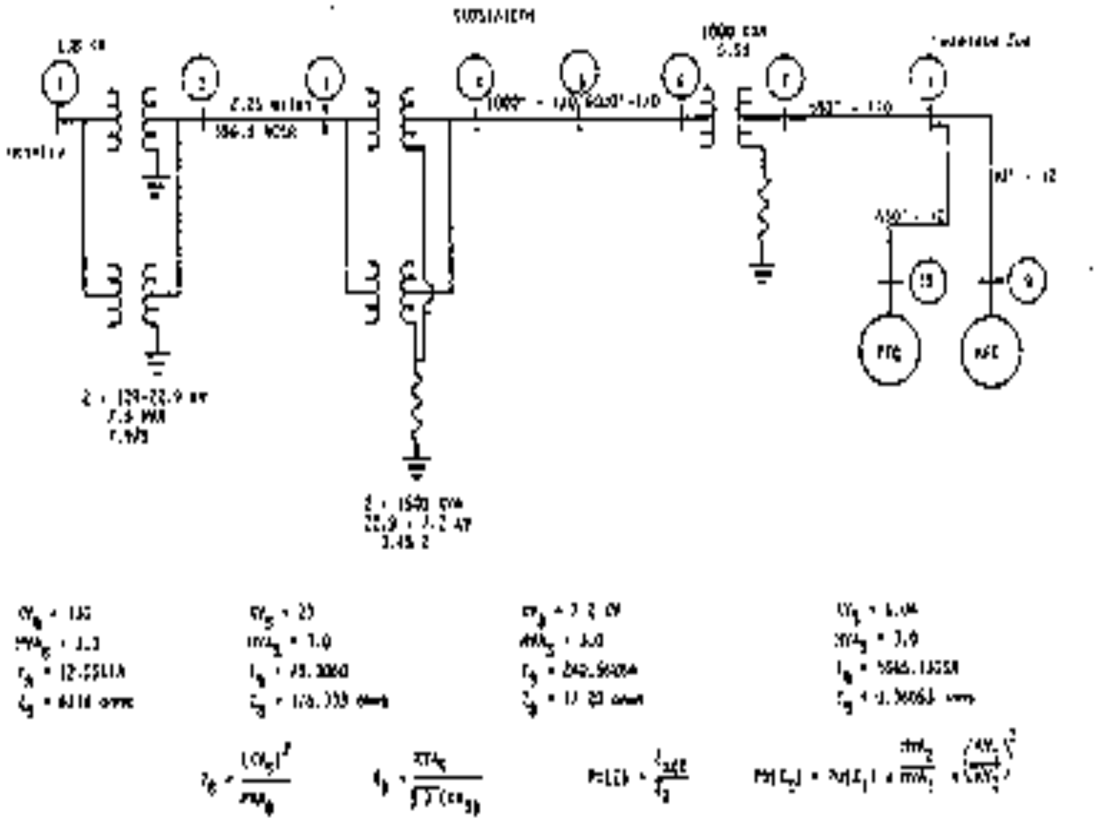

Figure A-1

IMPEDANCE CALCULATIONS

Utility

I = 7455 Amps (X/R=6)
V = 138 kV
Z = (1.04) (138 kV) / (1.732) &1955)
Z = 10.4166
R = 1.7124
X = 10.2745
Z (pu) = 0.0002698 + j 0.0016185

Aerial Lines

Z_{11} = $R_a + j (X_a + X_j)$
For 556.5 kcmil ACSR - 5 foot spacing
R_a = 0.168 ohm/mile
X_a = 0.4146 ohm/mile
X_a = 0.2233 ohm/mile
Z_{11} = 0.168 + j 0.63798 ohm/mile
Z (pu) = 0.00214 + j 0.0081406

High Voltage Cable

I/0 SHD-GC 8 kV (90°C Insulation)
Z = 0.134 + j 0.037 ohm/1000'
Z (pu) = 0.05428 + j 0.01499

Portable Cable

4/0 SHD-GC - 2 kV (90°C Insulation)
Z = 0.068 + j 0.029 ohm/1000'
Z (pu) = 0.23360 + j 0.05228

Conveyor Motor (Pickup Speed)

I = 5 x 111 = 555 amps
pf = 0.5
Z = (548.5)/555 = .98826 ohms
Z = 0.49412 + j 0.85586
Z (pu) = 1.37057 + j 2.373890

Main Transformers

Assume
R = 1%
Z = 7.97%
Z (pu) = 0.01 + j .07907 on 7.5 MVA Base

Substation Transformers

Assume
R = 1%
X = 3.4%
Z (pu) = 0.01 + f 0.03249 on 1.5 MVA Base
Z (pu) = 0.02 + j 0.06898 on 3 MVA Base

Power Center - Transformers

Assume
R = 1%
Z = 5.5%
Z = 0.01 + j 0.05408 on 1 MVA Base
Z (pu) = 0.03 + j 0.16225

Portable Cable

#2 SHD-GC - 2 kV (90° Insulation)
Z = 0.218 + j 0.033 ohm/1000'
Z (pu) = 0.39303 + j 0.05950 (Tail-Face)
Z (pu) = 0.01814 + j 0.00495 (Head-Face)

All ten system busses were analyzed for three phase faults (TPF) and power factor (pf). The results are shown in Table A-1.

TABLE A-1

BUSS	kV	R(PU)	X(PU)	Z(PU)	TPF(PU)	TPF	pf
1	138.	.000270	.001619	.001641	633.824	7955	.164
2	23.	.002420	.017433	.017600	59.159	4455	.134
3	23.	.004560	.025578	.025950	40.068	3017	.176
4	7.2	.014560	.058068	.059830	17.383	4181	.243
5	7.2	.022315	.060209	.064160	16.209	3899	.352
6	7.2	.068840	.073056	.100380	10.371	2494	.686
7	1.04	.098840	.235306	.255166	4.075	6786	.387
8	1.04	.221440	.287590	.362876	2.866	4772	.610
9	1.04	.227980	.288574	.367763	2.828	4709	.620
10	1.04	.614447	.347086	.705596	1.474	2454	.871

The theoretical short-circuit current at the end of the tailgate conveyor motor cable is 2454 A. The motor starting-center can be calculated as shown below.

$$\begin{aligned} Z_T &= Z_{10} + Z_M \\ &= (0.614447 + j\ 0.347084) + (1.37057 + j\ 2.37384) \\ &= 1.984017 + j\ 2.720974 \\ &= 3.3681 \angle\ 53.9^\circ\ (pu) \\ I_m &= 494.5 \angle\ -53.9^\circ \end{aligned}$$

The voltage at the motor terminals is 846 V which reduces the breakdown torque by approximately 21% ($T/T_{BD} = 0.79$).

BREAKDOWN TORQUES WITH BOTH MOTORS OPERATING

$$Z_{TF} = 1.7636 + j\ 2.4334 = 3.0053 \angle 54.1^\circ$$

$$Z_{hF} = 1.3887 + j\ 2.3788 = 2.7545 \angle 59.7^\circ$$

$$Z_{eq} = 1.4390\ \ 57.0^\circ = 0.7837 + j\ 1.2068$$

$$Z_{total} = 1.0051 + j\ 1.4944 = 1.8010 \angle 56.1^\circ$$

$$I_{total} = 924.7 \angle -56.1^\circ\ A$$

$$E_8 = 0.7989 + j\ 0.0130 = .7990 \angle 0.9^\circ$$

$$I_{TF} = 442.8\ A \qquad T/T_{BD} = 0.64\ (0.57)$$

$$I_{HF} = 483.1\ A \qquad T/T_{BD} = 0.75\ (0.58)$$

IMPEDANCES FROM SYSTEM CHANGES

A. 31 MVA to 40 MVA

Z = 0.01546 + j 0.016431

I_{sc} = 2532 A

K = 4.56

B. 31 MVA to 60 MVA

Z = 0.03327 + j 0.03531

I_{sc} = 2626 A

K = 4.73

C. 1000 kVA/5.5% to 1500 kVA/5.5%

Z = 0.01 + j 0.05408

I_{sc} = 2580 A

K = 4.65

D. 1000 kVA/5.5% to 1000 kVA/3%

Z = .01364 + j 0.07376

I_{sc} = 2624 A

K = 4.65

E. 4/0 Cable to two 4/0 Input Cable
Z = .0613 + j 0.02614
I_{sc} = 2708 A
K = 4.88

F. #2 Cable to 1/0 Motor Cable
Z = .44783 + j .004913
I_{sc} = 2993 A
K = 5.39

G. Changes E & F Above
Z = .20913 + j .031053
I_{sc} = 3370 A
K = 6.07

H. 1000 kVA/5.5% - 1500 kVA/3%
Z = 0.0189 + j 0.1322
I_{sc} = 2736 A
K = 4.93

I. Changes E, F & H Above
Z = 0.22803 + j 0.16325
I_{sc} = 4047 A
K = 7.29

In G, the motor current, voltage and torque are:
I_m = $517 \angle -56.6^\circ$ Amperes
E_m = 885 volts
T_m = 87% T_{BD}

In H, the motor current, voltage and torque are:
I_m = $512 \angle -52.8^\circ$ Amperes
E_m = 877 volts
T_m = 85% T_{BD}

APPENDIX B. SUMMARY OF OPERATING CHARACTERISTICS OF LONGWALL SYSTEMS LOCATED IN NORTHERN WEST VIRGINIA

COMPANY/ MINE	SECONDARY MVA (MVA-pf)	TAIL-FACE TPF (Amperes)	T.F. (70%) CURRENT (Amperes)	SIZE OF CONVEYOR MOTOR (hp)	T/T_{BD} (%)	K**
A-1	25.3 - .250	5123	865-910	350	86-86	5.51
A-2	16.1 - .320	4220	711-750	300	78-87	5.24
B-1 (1)	18.0 - .34	3781	692-728	300	74-82	4.70
B-1 (2)	19.4 - .33	4013	695-732	300	75-83	4.99
B-2 (1)	8.5 - .22	2951	462-486	200	75-83	5.52
B-2 (2)	6.6 - .21	2561	449-473	200	71-78	4.79
B-3	9.2 - .305	2911	556-585	250	67-76	4.34
B-4 (1)	10.2 - .290	3090	561-591	250	70-78	4.61
B-4 (2)	14.9 - .280	2695	465-490	200	76-84	5.02
B-5	10.7 - .243	3210	566-596	250	71-79	4.79
B-6 (1)	18 - .300	5900	618-651	250	85-94	8.81
B-6 (2)	10.6 - .347	3932	584-615	250	76-84	5.87
B-7	11.2 - .328	4092	588-619	250	77-85	6.11
B-8	12.2 - .272	4395	594-625	250	78-87	6.56
*B-9	11.8 - .355	2454	470-495	200	72-79	4.42
B-10 (1)	11.8 - .396	2642	496-522	200	86-95	4.94
B-10 (2)	10.9 - .326	2514	488-514	200	83-92	4.70
C-1 (1)	11.3 - .283	2751	404-425	175	74-85	5.85
C-1 (2)	12.5 - .351	3240	413-435	175	77-86	6.89
D-1	17.9 - .305	4546	493-519	200	85-94	8.50
E-1	9.5	3976	322-340	125	89-100	11.7
F-1 (1)	17.2 - .39	4028	593-624	250	78-87	6.01
F-1 (2)	16.4 - .40	3935	591-622	250	78-86	5.87
G-1	14.9 - .203	3667	581-612	250	75-84	5.47

* A full-load current of 110 amperes was used to calculate the motor impedance instead of using the impedance listed in Table 2.

** K = Tail-Face TPF (A) ÷ five times full-load current of Motor (A) from Chart II.

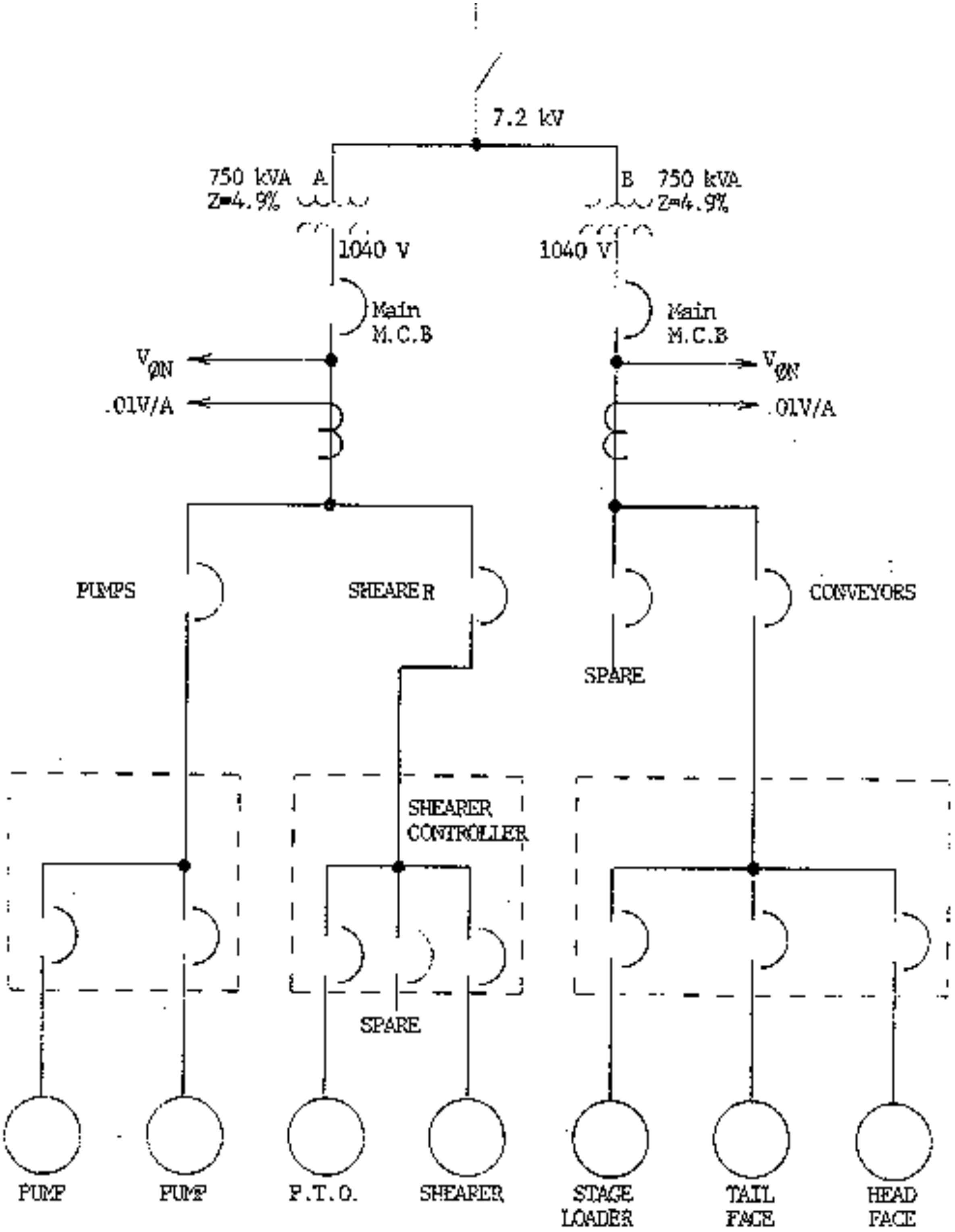
7.2 kV
750 kVA
Z=4.9%
A
B
750 kVA
Z=4.9%
1040 V
1040 V
Main
M.C.B
Main
M.C.B
$V_{\emptyset N}$
.01V/A
$V_{\emptyset N}$
.01V/A
PUMPS
SHEARER
SPARE
CONVEYORS
SHEARER
CONTROLLER
SPARE
PUMP
PUMP
P.T.O.
SHEARER
STAGE
LOADER
TAIL
FACE
HEAD
FACE

FIGURE B-1

COMPARISON OF TEST RESULTS WITH THEORETICAL RESULTS

COMPANY B MINE 4(1)

TEST NO. 3 - Tail-Face Motor Start-Up

Measured values at power center :

Transformer A	Transformer B
I_A = 0.0 A	I_B = 789 A
V_A = 583 V	V_B = 531 V

$V_{no\text{-}load}$ = 604 V

$$I_{sc} = 789 \left(\frac{604}{72.5} \right) = 6573 \ A$$

Theoretical Calculations:

$$\begin{aligned} Z &= 0.1098 \ \angle 72.6^\circ \ ohms \\ Assume \ I &= 789 \ \angle -60^\circ \ A \\ V_{drop} &= (789 \ \angle -60^\circ) \ (.1098 \ \angle 72.6^\circ) \\ &= 86.6 \ \angle 12.6^\circ \ V \\ &= 84.5 + j \ 18.9 \\ V_B &= 519.5 - j \ 18.9 = 520 \ volts \end{aligned}$$

TEST NO. 4 - Head-Face Motor Start-Up

Measured values at power center:

Transformer A	Transformer B
I_A = 0.0 A	I_B = 837 A
V_A = ------	V_A = 515 V

$V_{no\text{-}load}$ = 604 V

$$I_{sc} = 837 \left(\frac{604}{89} \right) = 5680 \ Amps$$

Theoretical Calculations:

$$\begin{aligned} Z &= 0.1098 \ \angle 72.6^\circ \ ohms \\ Assume\ I &= 837 \ \angle -60^\circ \ A \\ V_{drop} &= (837)\ (.1098)\ \angle 12.6^\circ \\ &= 91.9 \ volts \ \angle 12.6^\circ \\ &= 89.7 + j\ 20.0 \\ V_a &= 514.3 - j\ 20.0 = 515 \ volts \end{aligned}$$

TEST NO. 5A - Tail-Face and Head-Face Motors Start-up

Tail-Face Motor Only

$$\begin{aligned} I_B &= 768 \ A \\ V_B &= 520 \ V \\ V_{no-load} &= 593 \ V \\ I_{sc} &= 6240 \ A \end{aligned}$$

Theoretical Calculations

$$\begin{aligned} V_{drop} &= (768)\ (.1098) = 84 \quad \angle 12.6^\circ V \\ &= 82 + j\ 18.3 \\ V_B &= 509 - j\ 15.3 = 523 \ V \end{aligned}$$

Measured Values

$$\begin{aligned} I_B &= 1182 \ A \\ V_B &= 478 \ V \\ I_{sc} &= 6148 \ A \end{aligned}$$

Theoretical Calculations

$$\begin{aligned} V_{Drop} &= 129.8 \ \angle \ 12.6^\circ \ V \\ &= 126.9 + j\ 28.3 \\ V_B &= 466 - j\ 18.9 = 466 \ V \end{aligned}$$

TEST NO. 6 - Shearer Motor Start-Up

Measured Values

$$\begin{aligned} I &= 1171\ A \\ V_A &= 479\ V \\ V_{no-load} &= 128.6\ \angle\ +\ 12.6^\circ \\ I_{sc} &= 5326\ A \end{aligned}$$

Theoretical Calculations

$$\begin{aligned} V_{no-load} &= 128.6\ \angle\ +\ 12.6^\circ \\ &= 125.5\ +\ j\ 26.7 \\ V_B &= 489\ -\ j\ 26.7\ =\ 490\ V \end{aligned}$$

Average I_{sc} 6000 A from measured values.

Theoretical value -5700 A

APPENDIX C

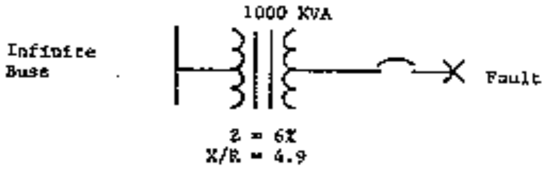

Figure C-1

$I_{sc} = \frac{555}{0.06} = 9250$ A

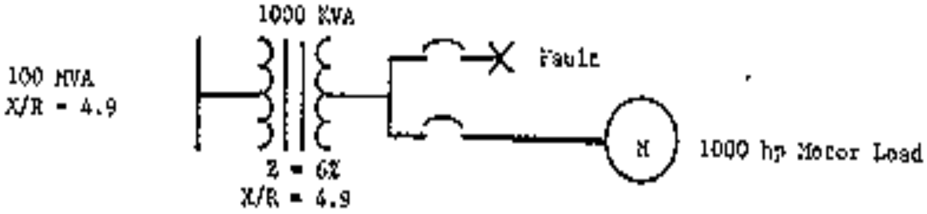

Figure C-2

$Z_T = 0.01 + 0.06$

$Z_T = 0.07$

$I_{sc} = \frac{555}{0.07} + 3.6\ (555)$

$= 7929 + 2000$

$= 9929$ A

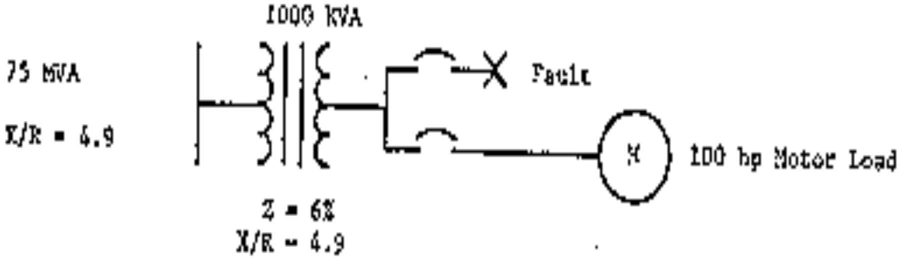

Figure C-3

$Z_T = 0.0133 + 0.05 = 0.0733$

$I_{sc} = \frac{555}{0.0733} = 9572$ A